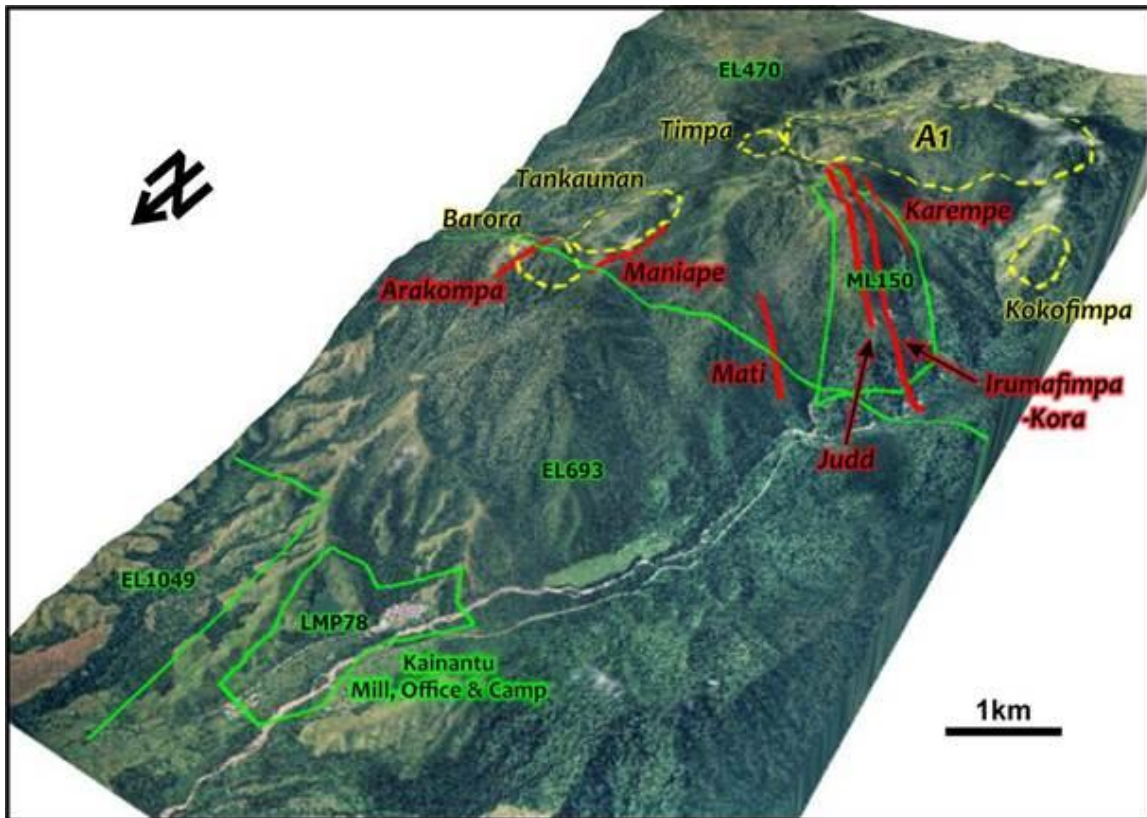


INDEPENDENT TECHNICAL REPORT
MINERAL RESOURCE ESTIMATE UPDATE AND PRELIMINARY
ECONOMIC ASSESSMENT OF IRUMAFIMPA AND KORA GOLD
DEPOSITS,
KAINANTU PROJECT, PAPUA NEW GUINEA



Prepared by Nolidan Mineral Consultants

Australian Mine Design and Development

and Mincore

for

K92 Mining Inc.

Authors:

Anthony Woodward BSc Hons., M.Sc., MAIG

Christopher Desoe BE (Min)(Hons), FAusIMM, RPEQ, MMICA

Lisa J Park GAICD FAusIMM

Effective Date: 02 March 2017

Submitted Date: 02 March 2017

Reference: NMC1701

Table of Contents

1.	SUMMARY	8
1.1	INTRODUCTION	8
1.2	GEOLOGY AND MINERALIZATION,	9
1.3	RESOURCE ESTIMATE	10
1.4	EXPLORATION TARGETS	11
1.5	PREVIOUS MINING AND PROCESSING	12
1.6	EXPLORATION	13
1.7	SCOPING STUDIES	13
1.7.1	Irumafimpa	14
1.7.2	Kora	14
1.7.3	Treatment Plant Upgrade	15
1.8	RECOMMENDATIONS	15
1.8.1	Exploration	15
1.8.2	Mine	15
1.8.3	Treatment Plant	16
2	INTRODUCTION	17
2.1	ISSUER	17
2.2	TERMS OF REFERENCE AND PURPOSE	17
2.3	INFORMATION USED	17
2.4	SITE VISIT BY QUALIFIED PERSONS	18
3	RELIANCE ON OTHER EXPERTS	18
4	PROPERTY DESCRIPTION AND LOCATION	18
4.1	TENURE	20
4.1.1	Exploration Licence	20
4.1.2	Mining Lease	21
4.1.3	Mining Lease No 150 Renewal Conditions	22
4.1.4	Expenditure Commitments	22
4.1.5	Reporting Requirements	22
4.2	ROYALTIES	23
4.2.1	States Right to Acquire 30% Interest In Mining Projects	23
4.3	CARE AND MAINTENANCE	23
4.4	OTHER SIGNIFICANT FACTORS AND RISKS	24
5	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY	24
5.1	PHYSIOGRAPHY	24
5.2	ACCESS	24
5.3	CLIMATE	25
5.4	LOCAL RESOURCES	25
5.4.1	Yonki Dam and Ramu Hydro Electric Power Station	25
5.4.2	Gusap Airstrip	26
5.5	INFRASTRUCTURE	26
6	HISTORY	26
6.1	PREVIOUS OWNERSHIP	27
6.2	HISTORICAL EXPLORATION 1928-2012	27
6.3	ML150 (IRUMAFIMPA, KORA, JUDD AND KAREMPE)	29
6.3.1	Kora and Irumafimpa	29
6.3.2	Judd	32
6.4	HISTORICAL EXPLORATION REVIEWS	33
6.4.1	Corbett (2009)	33
6.4.2	Tosdale (2012)	34
6.5	HISTORICAL ESTIMATES	34
6.5.1	Historical Estimates Irumafimpa-Kora	34
6.5.2	Historical Estimates – Arakompa and Maniape	35
6.6	HISTORIC PRODUCTION	35
6.6.1	Irumafimpa-Kora	35
6.6.2	Other sites	36
6.7	HISTORICAL PERFORMANCE AND RECONCILIATION REVIEWS	36

7	GEOLOGICAL SETTING AND MINERALIZATION	37
7.1	REGIONAL GEOLOGY	37
7.2	PROPERTY GEOLOGY	38
7.3	MINERALIZATION OVERVIEW	40
7.4	IRUMAFIMPA-KORA VEIN SYSTEM	43
7.4.1	Host rocks	44
7.4.2	Controls	44
7.4.3	Dimensions and Continuity	44
7.5	OTHER VEIN SYSTEMS	45
7.5.1	Judd	45
7.5.2	Karampe	45
7.6	PORPHYRY SYSTEMS	46
8	DEPOSIT TYPES	46
9	EXPLORATION	47
10	DRILLING	48
11	SAMPLE PREPARATION, ANALYSES AND SECURITY	48
11.1	SAMPLE PREPARATION	48
11.1.1	Drill core (HPL)	48
11.1.2	Mine Grade Control (HPL)	48
11.1.3	Drill core (Barrick)	49
11.1.4	Drill core (K92ML)	49
11.2	SAMPLE SECURITY	49
11.3	SAMPLE ANALYSES	49
11.3.1	Laboratory Independence and Certification	49
11.4	QUALITY ASSURANCE AND QUALITY CONTROL	50
11.5	QC PROGRAMS	50
11.5.1	1992-2002 Exploration	50
11.5.2	2001-2005 Exploration	51
11.5.3	2006-2008 Mine Sampling	51
11.5.4	2004-2006 Exploration	51
11.5.5	After 2008 Barrick Exploration	51
11.6	ADEQUACY OPINION	52
12	DATA VERIFICATION	53
12.1	DATA VERIFICATION PROCEDURES	53
12.1.1	Drillhole Database	53
12.1.2	Face Samples	53
12.1.3	Site Visits	53
12.1.4	Independent Samples	54
12.2	LIMITATIONS	55
12.3	VERIFICATION OPINION	56
13	MINERAL PROCESSING AND METALLURGICAL TESTING	56
13.1	MINERALIZATION CHARACTERISTICS	56
13.2	NATURE OF TESTING AND RESULTS	56
13.2.1	Samples 2000	56
13.2.2	Irumafimpa Samples - March 2002	56
13.2.3	Kora Testwork 2009	56
13.3	ORIGINAL PROCESS SELECTION AND DESIGN	57
13.4	RECOVERY ASSUMPTIONS	57
13.5	REPRESENTIVITY	58
13.6	FACTORS AFFECTING POTENTIAL ECONOMIC EXTRACTION	58
14	MINERAL RESOURCE ESTIMATE	58
14.1	APPROACH	59
14.2	SUPPLIED DATA	59
14.3	DIMENSIONS	60
14.4	GEOLOGIC INTERPRETATION	62
14.5	DATA PREPARATION AND STATISTICAL ANALYSIS	63
14.5.1	Drillhole Spacing	63
14.5.2	Domains & Stationarity	63
14.5.3	Compositing	64
14.5.4	Basic Statistics	64
14.5.5	Grade Capping	65

14.6	VARIOGRAPHY.....	66
14.6.1	Methodology.....	67
14.6.2	Variogram Models and Parameters	68
14.7	GRADE ESTIMATION.....	68
14.7.1	Methodology.....	68
14.7.2	Block Model	70
14.7.3	Informing Samples and Search Parameters	71
14.7.4	Discretisation	71
14.7.5	Block Model Attributes	71
14.8	VALIDATION AND COMPARISON WITH ALTERNATIVE ESTIMATES	72
14.9	ECONOMIC CUT-OFF PARAMETERS	74
14.10	BULK DENSITY	76
14.11	MOISTURE	76
14.12	MINING & METALLURGICAL FACTORS	76
14.13	ASSUMPTIONS FOR 'REASONABLE PROSPECTS FOR EVENTUAL ECONOMIC EXTRACTION'	76
14.14	RESOURCE CLASSIFICATION	76
14.14.1	Indicated Mineral Resource	78
14.14.2	Inferred Mineral Resource	78
14.15	DISCUSSION ON FACTORS POTENTIALLY AFFECTING MATERIALITY OF RESOURCES	78
14.16	MINERAL RESOURCE ESTIMATE STATEMENT.....	78
14.16.1	Notes to accompany resource statement:.....	79
15	MINERAL RESERVE ESTIMATES	79
16	MINING METHODS	79
16.1	IRUMAFIMPA	80
16.1.1	Irumafimpa Scoping Study Inputs	80
16.1.2	Irumafimpa Cutoff Grades	81
16.1.3	Irumafimpa Mining Method.....	81
16.1.4	Irumafimpa Development Concept.....	81
16.1.5	Irumafimpa Backfill Strategy	82
16.1.6	Irumafimpa Ventilation	82
16.1.7	Irumafimpa Stope Design.....	83
16.1.8	Irumafimpa Design Quantities	86
16.1.9	Irumafimpa Mine Schedule	87
16.2	KORA	89
16.2.1	Kora Scoping Study Inputs.....	89
16.2.2	Kora Cutoff Grades.....	91
16.2.3	Kora Mining Method	91
16.2.4	Kora Development Concept	92
16.2.5	Kora Backfill Strategy	94
16.2.6	Kora Ventilation	94
16.2.7	Kora Stope Design	95
16.2.8	Mine Design Quantities.....	98
16.2.9	Kora Mine Schedule	99
17	RECOVERY METHODS	101
17.1	INTRODUCTION	101
17.2	CURRENT PLANT CONDITION	104
17.3	PLANT UPGRADE SCOPING STUDY	104
18	PROJECT INFRASTRUCTURE	107
18.1	POWER AND WATER	107
18.1.1	Power	107
18.1.2	Water	108
18.2	MINE	108
18.2.1	Lower 800 Portal And Workshop	108
18.2.2	Underground Mine	108
18.2.3	Upper 1300 Portal.....	108
18.2.4	1400 Level Camp	109
18.3	PROCESSING PLANT	109
18.4	OFFICE	109
18.5	EXPLORATION AREA.....	109
18.6	ACCOMMODATION CAMP	110
19	MARKET STUDIES AND CONTRACTS.....	111
20	ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT	111

20.1	ENVIRONMENTAL LIABILITIES AND MINE CLOSURE.....	111
20.2	TAILINGS STORAGE FACILITY.....	112
20.2.1	Tailings Disposal.....	113
20.2.2	Future Tailings Capacity.....	113
20.3	REQUIRED PERMITS.....	113
20.4	ENVIRONMENTAL PERMITS.....	114
20.7	OTHER SIGNIFICANT FACTORS AND RISKS.....	116
21	CAPITAL AND OPERATING COSTS.....	117
21.1	KORA CAPITAL EXPENDITURE EQUIPMENT AND FACILITIES.....	117
21.2	KORA CAPITALISED DEVELOPMENT COSTS.....	117
21.3	KORA MOBILE FLEET.....	118
21.4	IRUMAFIMPA OPERATING COSTS.....	119
21.5	KORA OPERATING COSTS.....	119
22	ECONOMIC ANALYSIS.....	119
22.1	IRUMAFIMPA MINE PLAN.....	119
22.2	KORA MINE PLAN.....	122
22.2.1	15m Level Case.....	123
22.2.2	25m Level Case.....	125
23	ADJACENT PROPERTIES.....	126
24	OTHER RELEVANT DATA AND INFORMATION.....	126
25	INTERPRETATION AND CONCLUSIONS.....	127
25.1	EXPLORATION POTENTIAL.....	127
25.1.1	ML150.....	127
25.2	SCOPING STUDY RESULTS.....	129
25.2.1	Irumafimpa.....	129
25.2.2	Kora.....	129
25.2.3	Treatment Plant Upgrade.....	130
25.3	RISK ASSESSMENT.....	130
26	RECOMMENDATIONS.....	131
26.1	EXPLORATION.....	131
26.2	MINE.....	131
26.3	TREATMENT PLANT.....	132
27	REFERENCES.....	133
28	CERTIFICATE OF QUALIFIED PERSON.....	136
29	APPENDIX 1: GLOSSARY OF TECHNICAL TERMS AND ABBREVIATIONS.....	139

FIGURES

Figure 1.	Kainantu Project Location and Tenements.....	20
Figure 2.	Oblique View of Northern Part of Property, Showing Relief and Location of Main Infrastructure.....	24
Figure 3.	Kora long section showing potential depth extents of mineralization.....	29
Figure 4.	Local geology and Barrick drillholes location plan at Kora and Karempé.....	30
Figure 5.	Cross section 58600mN at Irumafimpa showing consistency of high grade, particularly within the Robinson lode.....	31
Figure 6.	Surface drilling traces showing surface projections of >100g/t Au and >10% Cu.....	31
Figure 7.	Long section view of Kainantu Resource Areas with Vein Composites colour coded for Au Eq.....	32
Figure 8.	Cross section showing narrow mineralization intersected at Judd lode.....	33
Figure 9.	Tectonic Framework of Papua New Guinea, Showing Location of Kainantu Property.....	38
Figure 10.	Kainantu property geology and known vein and porphyry deposits and prospects.....	40
Figure 11.	Diagram illustrating grade distribution within an original 0.7m sample.....	44
Figure 12.	Karempé location plan showing mapped veins and rock chip results.....	45
Figure 13.	Karempé long section showing strike extent of known surface footprint.....	46
Figure 14.	Conceptual model for porphyry and related low and high sulphidation mineralization.....	47
Figure 15.	Scatter of of Repeat Data, Astrolabe Laboratory 1992-2002.....	52
Figure 16.	Scatterplot of ALS vs Genalysis Results, 2002-2005.....	52
Figure 17.	Scatterplot of ALS Repeat Assays, 2002-2005.....	52
Figure 18.	Example CRM Control Chart (2002-2005).....	52
Figure 19.	Core Logging facilities.....	54

Figure 20: Almonte Automatic core saw and brick saw	54
Figure 21: BKDD0027 461.6 to 462.6 m 16.3 g/t Au, 131 g.t Ag and 6.48 g/t Cu	55
Figure 22: BKDD0026 583.25 to 584.2 m 0.49 g/t Au, 19.4 g/t Ag and 3.4% Cu	55
Figure 23: Plan view of the Irumafimpa-Kora Resource drilling, coloured by drillhole type.	61
Figure 24: Long section view of Kainantu Resource Areas with Vein Composites colour coded for AuEq	62
Figure 25: Vein arrangement at Kora, plan view at 1745mRL	62
Figure 26: Vein arrangement at Kora, section view at 58600N, looking north	62
Figure 27: Comparison of Vein Chemistries	64
Figure 28 Variograms generated for veins.	67
Figure 29: Trend analysis by northing for Ag g/t. (Kora to the left).....	73
Figure 30: Trend analysis by northing for Cu % (Kora to the left)	73
Figure 31: Trend analysis by northing for Au g/t (Kora to the left)	74
Figure 32: Trend analysis by northing for Au g/t at Irumafimpa with Judd and Mill veins separated	74
Figure 33: Irumafimpa-Kora Grade Tonnage Charts.....	75
Figure 34: ML150 Long Section with blocks coloured by resource category (looking west)	77
Figure 35: Avoca Benching method	82
Figure 36: MSO shapes at Irumafimpa with current voids and existing development.....	84
Figure 37: Level 1310RL showing voids and MSO stopes	85
Figure 38: MSO shapes selected for the 3 Year Mine Plan schedule.....	85
Figure 39: Monthly production from development and stopes for 3 Year Mine Plan	88
Figure 40 Monthly development metres for 3 Year Mine Plan	88
Figure 41 Annual LOM Irumafimpa production from development and stopes	89
Figure 42: Longitudinal projection looking west showing proposed Kora development in relation to existing Irumafimpa development.....	93
Figure 43: Kora development concept, longitudinal projection looking west.....	93
Figure 44: Ventilation concept for Kora development	95
Figure 45: Kora MSO shapes, 15m levels (longitudinal projection looking west)	97
Figure 46: Kora MSO stope shapes with mineralised veins, typical plan view at 1700mRL	98
Figure 47: Mill feed tonnes and development metres	100
Figure 48: Mill feed tonnes and grade.....	101
Figure 49: Process Plant Flowsheet 2014	103
Figure 50: Photograph of the processing plant (2008)	104
Figure 51: Photograph of the Crushing Circuit (2012)	104
Figure 52: Upgraded Process Plant Flowsheet	106
Figure 53. Oblique view of Process Plant and office infrastructure area (<i>circa 2012</i>)	109
Figure 54. Profile view of Exploration Coreshed (left foreground), warehouse (left background) and offices (right foreground) (<i>circa 2014</i>).....	110
Figure 55 Aerial and Ground view of Kumian Accommodation Camp (<i>circa 2009</i>)	110
Figure 56. Aerial view of Process Plant, Tailings Storage Facility and Accommodation Camp (<i>circa</i> <i>2009</i>)	111
Figure 57: Kora Annual cashflow, cumulative cashflow and DCF; 15m Levels	124
Figure 58: Kora 25m Level Case Annual cashflow, cumulative cashflow and DCF	125
Figure 59. Location of Kainantu project and gold deposits within major mineralized province.	127
Figure 60: Kainantu Exploration Targets.....	128

TABLES

Table 1. Project Tenure Details.....	20
Table 2. Local Resources to Property	25
Table 3. Kainantu exploration statistics, Barrick and historical.	26
Table 4. Significant intercepts, Barrick drilling (> 1 g/t Au) at Kora.....	30
Table 5 Historical estimates reported for the Arakompa, Maniape and Aifunka deposits	35
Table 6. Irumafimpa - Stope by stope comparison of Reserve estimate against Grade Control estimate (November 2008).....	35
Table 7. Historic mill production for Irumafimpa	36
Table 8. Summary operations timeline for the Project	37
Table 9. Summary of main regional rock units identified within Kainantu area.	39

Table 10. Local deformation history for the Kainantu area.....	39
Table 11. Summary of mineralization, host rocks, dimensions and continuity for main Kainantu deposits and prospects	41
Table 12. Mineralization and alteration paragenesis in the Irumafimpa-Kora vein system.....	43
Table 13: Kora recoveries adjusted for saleable concentrate.	57
Table 14. Master Database Structure	59
Table 15: Mineralized samples outside vein tags.	59
Table 16: Database Extents	60
Table 17: Univariate uncapped statistics for gold, silver and copper by vein	65
Table 18: Grade caps for gold by vein	66
Table 19: Semivariogram Parameters used for Irumafimpa-Kora estimation	68
Table 20: Block Model Extents	70
Table 21: Search Parameters.....	71
Table 22: Block Model Attributes	72
Table 23: Alternate estimation results at a 5g/t Au cut-off (drill samples only)	72
Table 24. Resources by Deposit, Mining Method and Category	77
Table 25: ML150 Resources by Deposit and Category	78
Table 26: Key project assumptions for Irumafimpa stope cut-off grade.....	80
Table 27: Base mining cost allowances, \$/t, by activity	80
Table 28: Irumafimpa Cutoff grades.....	81
Table 29: MSO Parameters.....	83
Table 30: Irumafimpa MSO stope shape quantities	83
Table 31: Irumafimpa MSO stope shape quantities	86
Table 32: Irumafimpa Possible active stopes	86
Table 33 Schedule quantities by month for Irumafimpa 3 Year Mine Plan.....	87
Table 34: Annual Irumafimpa LOM Schedule quantities	89
Table 35: Key project assumptions for Kora stope cutoff grade	90
Table 36: Kora AuEq Cutoff grades.....	91
Table 37: MSO Parameters.....	95
Table 38: MSO stope shape quantities – 15m level case	96
Table 39: Kora MSO stope shape quantities – 25m level case	96
Table 40: Development quantities for 15m levels.....	98
Table 41: Schedule Parameters	99
Table 42 Mine Closure Costs - Barrick 2010	112
Table 43: Kora Capital Expenditure – Equipment and Facilities	117
Table 44: Kora Capital Expenditure - Development	117
Table 45: Kora Capital Expenditure – Mobile Plant	118
Table 46: Operating costs for the Irumafimpa Life Of Mine Plan	119
Table 47: Operating costs for the Kora Mine Plan	119
Table 48: Simplistic Cashflow Model for 3 Year Mine Plan	120
Table 49: Simplistic Irumafimpa LOM cashflow model	121
Table 50: Kora Annual LOM Schedule; 15m Levels.....	124
Table 51: Kora Annual LOM Schedule; 25m Levels.....	125
Table 52. ML150 Exploration Prospect Ranking Vein Targets	128

1. SUMMARY

1.1 INTRODUCTION

This report is an independent technical report dated 02 March 2017 of the geology, exploration, mineral resource estimates, and mining scoping studies for the Irumafimpa and Kora gold deposits at the Kainantu project. The Kainantu property covers a total area of 405 sq.km and is located in the Eastern Highlands Province of Papua New Guinea, approximately 180 km west-northwest of Lae.

In October 2016 Mr John Lewins, Chief Operating Officer of K92 Mining Inc. (“K92”), requested Nolidan Mineral Consultants (“Nolidan”), Australian Mine Design and Development (“AMDAD”), and Mincore Pty Ltd (“Mincore”) to prepare a report in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) incorporating the results of recently completed mineral resource estimates and mine scoping studies of the Irumafimpa and Kora gold deposits and plant upgrade studies.

Nolidan was engaged by K92 to update the mineral resource estimates for the Irumafimpa and Kora deposits which were previously reported in May 2015. AMDAD was initially engaged in 2016 by K92ML to compile a 3 Year Mine Plan for mining of the Irumafimpa deposit. Later in 2016 AMDAD was engaged to undertake a Scoping Study for the development of the Kora deposit. In conjunction with the Kora Scoping Study, Mincore was engaged to carry out a detailed study on the potential expansion of the existing processing plant to treat 400,000tpa of ore primarily from the Kora deposit. As part of the Irumafimpa and Kora studies AMDAD prepared conceptual cashflows to provide guidance in relation to the economic viability of those mine plans. Those cashflows are the basis of the Preliminary Economic Assessment presented in this Technical Report.

This assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessments will be realized.

The Project as described herein is 100% owned by K92 Mining Limited (“K92ML”) (formerly Barrick (Kainantu) Limited); a company incorporated in Papua New Guinea, which is 100% owned by K92 Holdings (PNG) Limited (“K92PNG”), a 100% owned subsidiary of K92 Holdings International Limited (“K92 Holdings”).

K92PNG acquired K92ML from Barrick (Niugini) Limited (“Barrick”) pursuant to an agreement dated June 11, 2014 (the “K92ML Purchase Agreement”) (which closed March 6, 2015), for the sum of US\$2,000,000. Under the terms of that agreement K92PNG is obligated to make additional payments of up to US\$60,000,000 as follows:

- (i) US\$20,000,000 upon K92PNG determining 1,000,000 ounces of gold equivalent (based on in-situ and mined product classified as measured mineral resource, indicated mineral resource, probable ore reserve or proven ore reserve); and
- (ii) US\$5,000,000 upon upon K92PNG determining each additional 250,000 ounces of gold equivalent (on the same bases as stated above) up to an aggregate of 3,000,000 ounces.

The obligation to pay additional payments will cease on March 6, 2025.

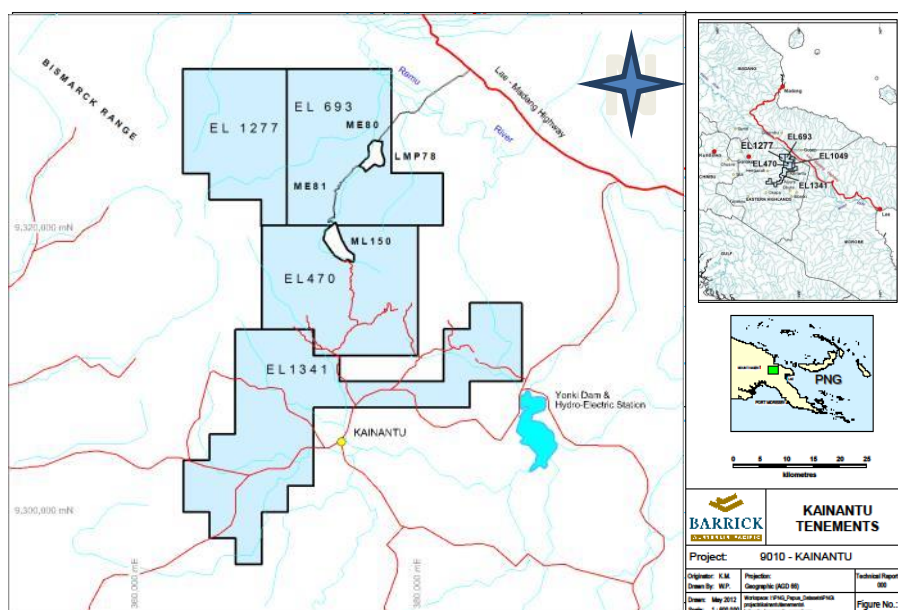
On August 21, 2014, Otterburn, K92 Holdings and the shareholders of K92 Holdings entered into a Share Exchange Agreement, pursuant to which Otterburn agreed to acquire all of the issued and outstanding shares of K92 Holdings, from the shareholders of K92 Holdings on the basis of one common share of Otterburn for each outstanding common share of K92 Holdings, for an aggregate of 49,126,666 Otterburn common shares. Subsequently the transaction was restructured, and Otterburn and Cada International Ltd. (a wholly owned subsidiary of Otterburn) entered into a merger agreement with K92 Holdings on April 15, 2016, pursuant to which K92 Holdings agreed to merge with

Cada International Ltd. to form an amalgamated subsidiary of Otterburn, and whereby Otterburn agreed to acquire all of the outstanding shares of K92 Holdings, in exchange for common shares of Otterburn on the basis of one post-consolidation common share of Otterburn for each common share of K92 Holdings, for an aggregate of 49,126,666 Otterburn common shares.

K92 Mining Inc. (formerly Otterburn) is a company incorporated under the laws of British Columbia, Canada; the common shares of which are publicly listed on the TSX Venture Exchange.

K92ML is the registered holder of the following tenements in PNG, as issued by the applicable government authorities in accordance with the PNG Mining Act 1992 (the "Mining Act"):

1. Mining Lease 150 ("ML150"), effective until June 14, 2024;
2. Mining Easements 80 and 81 ("ME80" and "ME81"), each effective until June 14, 2024;
3. Licence for Mining Purposes 78 ("LMP 78"), effective until June 14, 2024;
4. Exploration Licence 470 ("EL470"), effective until February 05, 2017;
5. Exploration Licence 693 ("EL693"), effective until February 05, 2017;
6. Exploration Licence 1341 ("EL1341"), effective until June 20, 2018. ; and
7. Exploration Licence 1277 ("EL1277") which expired on May 20, 2009. The PNG Minister for Mining rejected K92ML's application for renewal on December 5, 2011. K92ML initiated legal action to compel the Minister for Mining to overturn the decision, but the court instructed the parties to instead try to reach an out-of-court settlement. Negotiations in that regard have to date been unsuccessful; and if not settled will revert to the courts for a decision.



Kainantu Project Location.

Source: Barrick 2014

1.2 GEOLOGY AND MINERALIZATION,

The Kainantu property is located within the New Guinea Thrust Belt, close to its northern contact with the Finisterre Terrane. The property area is underlain by metamorphosed sedimentary rocks of the Early Miocene Bena Bena Formation, unconformably overlain by Miocene age sedimentary and intermediate volcanic rocks of the Omaura and Yaveufa Formations. These formations were intruded in the mid-Miocene by the Akuna Intrusive Complex, which comprised multiple phases of mafic to felsic magma. Late Miocene age Elandora Porphyry dykes formed small high level crowded feldspar porphyry dykes and diatreme breccias.

Mineralization on the property includes gold, silver and copper occurring in epithermal Au telluride veins and Au Cu Ag sulphide veins of Intrusion Related Gold Copper (“IRGC”) affinity and also less explored porphyry Cu Au systems; and alluvial gold. The Irumafimpa-Kora vein deposit is the most advanced project at Kainantu with current defined resources and past modern mining activity in the Irumafimpa area. The deposit occurs in the centre of a large mineralized system approximately 5 km x 5 km in area that has been partly delineated by drilling and comprises several individual zones of IRGC and porphyry style mineralization. The current resources occupy a broad northwest trending mineralized zone more than 2.5 km long and up to 60m wide in which individual veins vary from less than one metre wide that pinch and swell over short distances (Au telluride lodes) to more continuous veins up to several metres wide (Au Cu Ag sulphide lodes).

The Kora veins average 3.1m true width; which is the entire extent of the known veins before cut-off grades are applied. The Mill veins at Irumafimpa average 1.2m true width, which is the entire extent of the known veins before cut-off grades are applied, and also the minimum width used during resource estimation.

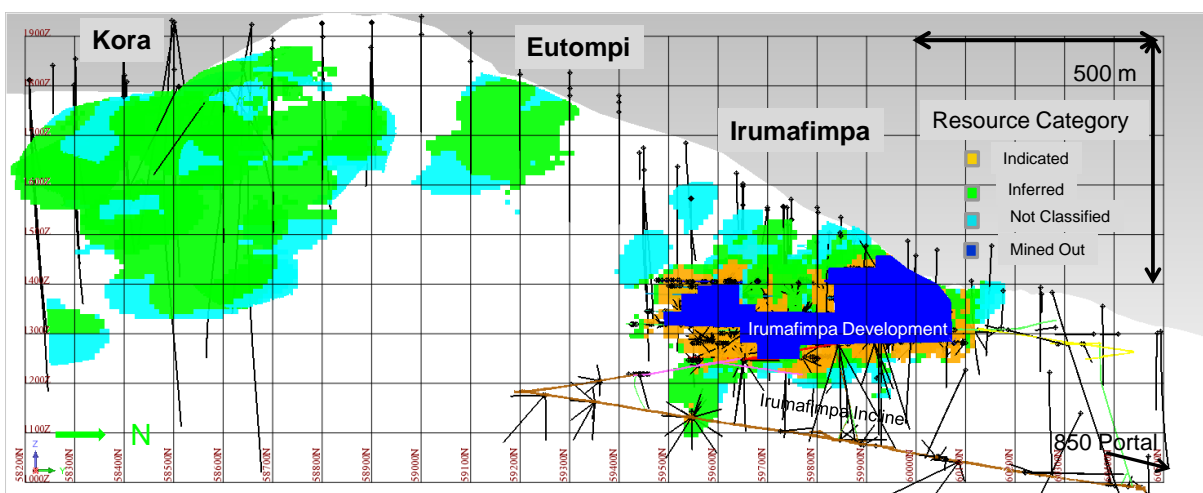
Other less advanced prospects on the property include epithermal Au veins similar to Irumafimpa, IRGC veins similar to Kora, porphyry Cu Au systems, skarn Cu, Pb and Zn mineralization and alluvial gold.

1.3 RESOURCE ESTIMATE

A resource estimate was completed for the Irumafimpa-Kora vein systems based on the historical surface and underground drilling conducted by previous owners, Barrick and HPL. Face channel and grade control samples collected during previous mining operations were also used but have only a local influence.

Comparison of grade control face sampling and drilling in the same mineralized zones shows a significant bias towards lower average grades in drilling compared with the average grade of the face samples. For all veins the highest recorded values for gold (outliers) occurred in drillhole samples and grade capping was therefore used. Face samples are however concentrated in the higher grade mining areas, so were included in resource estimation.

Results are presented in the table below and should be read in conjunction with the notes following.



ML150 long section with blocks coloured by resource category.

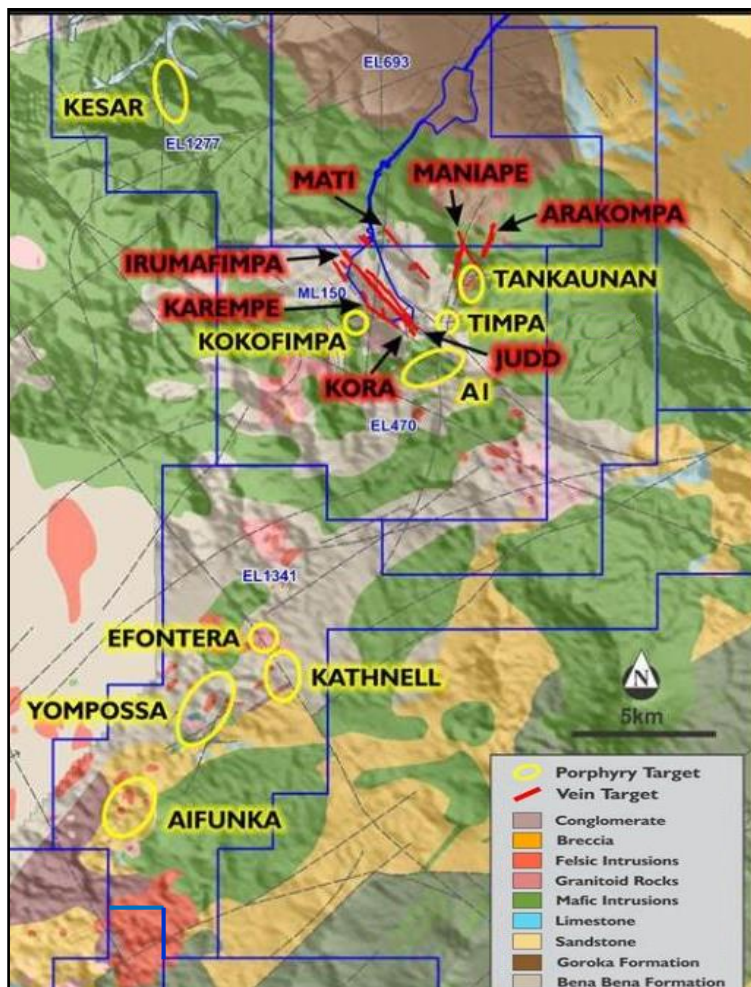
Mineral Resource by Deposit and Category										
Deposit	Resource Category	Tonnes	Gold		Silver		Copper		Gold Equivalent	
		Mt	g/t	MOz	g/t	MOz	%	Mlb	g/t	MOz
Irumafimpa	Indicated	0.56	12.8	0.23	9	0.16	0.28	37	13.4	0.24
	Inferred	0.53	10.9	0.19	9	0.16	0.27	74	11.5	0.20
Kora/Eutompi	Inferred	4.36	7.3	1.02	35	4.9	2.23	215	11.2	1.57
Total Indicated		0.56	12.8	0.23	9	0.16	0.3	4	13.4	0.24
Total Inferred		4.89	7.7	1.21	32	5.06	2.0	218	11.2	1.76

*M in Table is millions. Reported tonnage and grade figures are rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. Gold equivalents are calculated as AuEq = Au g/t + Cu%*1.52+ Ag g/t*0.0141.*

1. The current sample exploration database was supplied by Barrick in MS Access format.
2. Estimation undertaken in Surpac™, using ordinary kriging (“OK”) in unfolded space.
3. The estimation block size was 10m in Y and 10m in Z with width estimated in unfolded space as a variable. Grade was interpolated by domain using OK estimation with parameters based on directional variography by domain. Thickness of the vein was also estimated by OK estimation.
4. Results validated against drill data and Inverse Distance Squared, Nearest Neighbour, Gram M Accumulation estimates and Ordinary Krige uncapped estimates.
5. Minimum mining width of 1.2 m horizontal. Grade was diluted to account for minimum width.
6. This mineral resource estimate is based on 78,935 metres of drilling from 767 holes, and 18,312 metres of assayed intervals across all lodes. A single vein composite was used for each drill intercept on each lode – cut-off for selection was 3 m-gms Au Equivalent. There are a total of 2,003 vein composites across 19 veins, including 349 face composites.
7. A mined out area representing the extent of current mining projected across all lodes were removed from the final model as the exact location of individual stopes is not clear.
8. Top caps were applied to the composites for each vein. Grade caps were selected to restrict the influence of outliers where drilling was sparse, and varied by vein.
9. A minimum of 2 samples and maximum of 12 samples were used for each block. Search distances varied by lode and reflect the variogram ranges of 100-200 m, maximum projection beyond last drill-hole is 50 m.
10. The volume for each vein was defined by a wireframe in 3D space and is used to constrain the resource blocks.
11. Lower cut-off grades for reporting were a combination of thickness and grade reflecting mining methods, metallurgical recovery, and royalties:
 - a. Narrow Vein - Shrink Stopes - 1.2 m – 3 m thick and ≥ 6 g/t AuEq
 - b. Wide Vein – Mechanised Stopes - > 3 m thick and ≥ 5 g/t AuEq
12. Resource categories are based on estimation confidence and number of informing samples as a guide. Blocks shown in the Long Section have been coloured by resource category. Turquoise blocks are unclassified blocks with only one sample supporting them and are not included in the resource estimate.
13. Vein blocks in the Irumafimpa deposit have been assigned a density of 2.9 t/m³ and vein blocks in the Kora deposit have been assigned a density of 2.8 t/m³.

1.4 EXPLORATION TARGETS

The Kainantu project is located in a recognized copper-gold province, as evidenced by the underlying geology and presence of nearby major projects operated by global majors Barrick, Newcrest and Harmony. There remain a significant number of major untested and early stage targets. Within ML150 are the Kora lodes which are strongly mineralized at the limit of drilling and open and in all directions, as well as the Judd, Karempa and other unnamed mineralized lodes parallel to defined resources which have economically attractive grade in surface and/or drill samples from very limited work to date.



Kainantu geology and known vein and porphyry deposits and prospects.

(Source: Barrick, 2014)

1.5 PREVIOUS MINING AND PROCESSING

During the mining operation at Irumafimpa between 2006 and 2009, mining was predominantly shrink stoping with some bench stoping (longhole). The method applied was based on the geological structure and varying vein widths. Multiple independent reviews have shown that previous operators had considerable difficulty with dilution issues during mining which has been mainly attributed to the geological complexity of the veins and a poor understanding of grade distribution within the veins.

The processing plant built to treat the Irumafimpa lodes was demonstrated in the previous operating phase between 2006 and 2008 (HPL and Barrick) to be generally well suited to the mineralization in that deposit.

The underground mining operation and process facility were not operated between January 2009 and September 2016. K92ML commenced rehabilitation of the underground workings in March 2016 and refurbishment of the treatment plant in April 2016.

In order to comply with the terms of the renewal of ML150, K92ML was required to refurbish the mine and mill by December 31, 2016. This was effectively accomplished in September 2016.

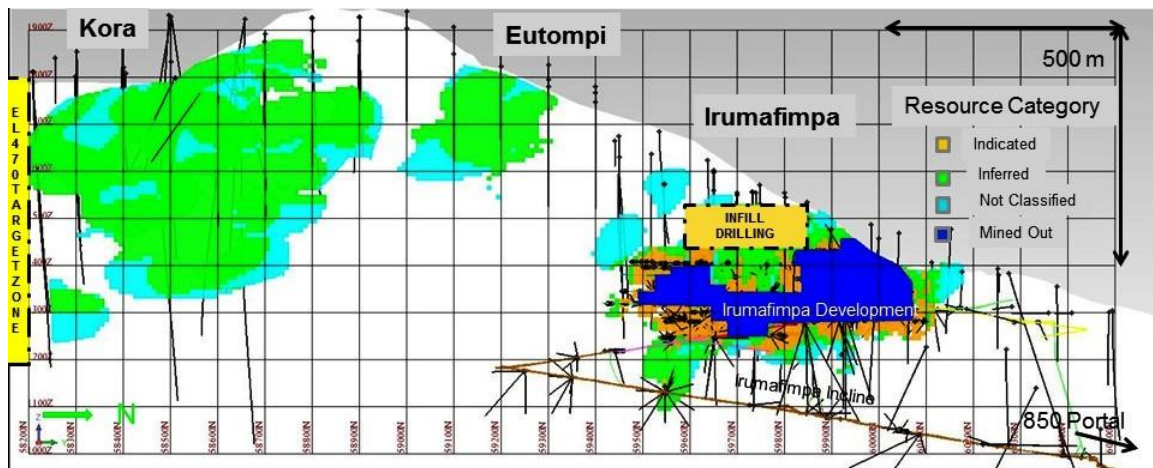
An additional requirement is that operations and production from the Kora deposit must commence on or before 30 June 2018.

1.6 EXPLORATION

Further investigation is required to understand the geological complexity of the veins at Kainantu and the controls on high grade shoots. K92ML has commenced close spaced drilling from existing underground workings to confirm indicated and inferred resources at Irumafimpa and to test the Judd vein.

Significant opportunity remains for resource extension within the immediate mine environment, including:

- The Irumafimpa-Kora vein system is open at depth, in the central areas beneath the top of the mountain (Eutompi) and to the South (Kora) beyond the ML150 boundary.
- Blocks shown in the Longitudinal Section below have been coloured by resource category. Turquoise blocks are blocks with only one sample supporting them and are not included in the resource estimate. These unclassified areas are extensive and represent obvious targets for immediate drillhole targeting with significant upside to possible production and mine life. AMDAD has estimated there are approximately 1Mt of unclassified material above 4.5 g/t AuEq. However the width of some of these veins may not be sufficient for economic mining.



Kainantu Exploration Targets 2016

1.7 SCOPING STUDIES

The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

It should also be noted that the mine plan and scoping studies prepared for the Irumafimpa and Kora deposits are not based on Ore Reserves. The estimates of tonnes and grade reported and scheduled in both the Irumafimpa and Kora Scoping Studies do not constitute an Ore Reserve because:-

- Most of the resource estimate on which the tonnes and grade are based on are at too low a level of confidence to allow conversion to Ore Reserves.
- There is insufficient geotechnical information for the Kora deposit to be confident in development and extraction design parameters and costs and the mine plan can only be considered conceptual.
- Limited metallurgical testwork has been completed for the copper-gold mineralization at Kora and further work will be required to confirm the processing cost and recovery assumptions.

Non-mining economic and processing parameters assumed and referred to in the studies are conceptual. They were applied for the purpose of identifying the part of the Resource that notionally may be economic, in order to prepare conceptual extraction designs. Schedules are based on conceptual development and stoping quantities and not practical designs. Cashflow schedules are based on these assumed parameters. They should be treated with caution, and they should not be interpreted as a measure of the value of the deposit.

1.7.1 Irumafimpa

The preliminary economic assessment for Irumafimpa is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Additionally geotechnical assessment is required to confirm the feasibility of stope designs. There is no certainty that the preliminary economic assessment will be realized.

Key estimates from the Irumafimpa Mine Plan prepared by AMDAD are:

- Over the 3 years of the mine plan treatment of 0.49Mt tonnes at 8.4 g/t Au, 5.8 g/t Ag, 0.11%Cu would generate a net operating cashflow of USD \$56 million.
- Over the 8 years of the Mine Life treatment of 1.40Mt tonnes at 8.2 g/t Au, 5.8 g/t Ag, 0.19%Cu would generate a net operating cashflow of USD \$153 million

Note the cashflows stated above for Irumafimpa are operating cashflows only. They do not include any allowance for capital costs.

1.7.2 Kora

The preliminary economic assessment for Kora is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Additionally geotechnical assessment is required to confirm the feasibility of stope designs. There is no certainty that the preliminary economic assessment will be realized.

Key estimates from the Kora Mine Scoping Study prepared by AMDAD are:

- Over a 9 year operating life the plant would treat 3.2 Million tonnes averaging 7.1 g/t Au, 25 g/t Ag and 1.7% Cu (9.3 g/t Au Eq*).
- This would generate an estimated positive cash flow of US\$537 million using current metal prices if 15m levels are used in mining. If 25m mining levels are used then net cashflows are estimated as US\$558 million. This cashflow includes conceptual allowances for capital.
- Production of an estimated average of 108,000 Au Eq* ozs per annum over an 8 year period from Year 2 through to Year 9.
- An estimated Pre-tax NPV of US\$415 Million for 25m mining levels; or US\$397 Million for 15m levels; using current metal prices, exchange rate and a 5% discount rate;
- An estimated After-tax NPV of US\$329 Million for 25m mining levels; or US\$316 Million for 15m levels; using current metal prices, exchange rate and a 5% discount rate;
- Initial Capital Cost is estimated to be US\$13.8 Million, including the US\$3.3 million for the plant upgrade identified in the Mincore Scoping Study, but excluding the proposed Kora exploration inclines and diamond drilling. Sustaining Capital Cost is estimated to be a further US\$64 million spent over the life of the Kora mining for 25m mining levels or US\$83 for 15m mining levels.
- Operating Cost per tonne is estimated to be US\$125/tonne for 25m mining levels or US\$126/tonne for 15m mining levels.

- Excluding Initial Capital Expenditure of US\$14M, Cash Cost is estimated to be US\$547/oz Au Eq (inclusive of a 2.5% NSR) and AISC of US\$619/oz Au Eq for 25m mining levels; or US\$549/oz Au Eq (inclusive of a 2.5% NSR) and AISC of US\$644/oz Au Eq for 15m mining levels.

Current Metal Prices used were: Au – US\$1,300/oz; Ag – US\$18/oz; Cu – US\$4,800/tonne.

*Au Eq – calculated on above Metal Prices.

1.7.3 Treatment Plant Upgrade

Key conclusions from the study by Mincore on requirements for upgrading the treatment plant to 400,000tpa are:

- There is sufficient crushing and milling (comminution) power to grind 50tph to P80 of 106 µm.
- Additional flotation capacity is required to achieve acceptable residence times for each cell. There is sufficient space to install additional cells if future testwork identifies a requirement for longer residence time.
- The existing concentrate thickener and filter is adequate for 400,000tpa Kora feed averaging 1.7% copper.
- The existing tailings line is adequate but a full pump upgrade will be required.
- Construction time for the plant upgrade was estimated as 10 months.

1.8 RECOMMENDATIONS

1.8.1 Exploration

- Drilling should concentrate on infill drilling of current resources and extensions to veins within ML 150.

1.8.2 Mine

- The estimated costs used in producing the preliminary mine plans and scoping studies for mining of the Irumafimpa and Kora gold deposits need further refinement using actual costs from Irumafimpa once operations reach a steady state.
- Geotechnical studies of the mine workings need to be advanced to determine ground conditions and support requirements for development within waste and the mineralised veins.
- The position and condition of existing development and stope workings at Irumafimpa needs to be confirmed.
- Stope stability analysis is required to guide the selection of level interval (15m or 18m) and stope strike lengths suitable for the next stage of Kora mine design.
- Groundwater conditions need to be investigated.
- More detailed ventilation planning is required including analysis of ventilation options including VentSim modelling of airways to determine airflows, pressures, air power and fan specifications. Vent rise paths will need geotechnical investigations.
- The feasibility of raiseboring holes from surface greater than 500m long has to be investigated considering the implications, timing, and costs involved
- Development profiles for the Kora incline and lateral access development require further analysis in relation to materials handling requirements. More analysis to reduce initial waste development is recommended.

- The source and cost of any surface waste rock sources should be investigated and the various cement backfill options for Kora should be reviewed.

1.8.3 Treatment Plant

- Further metallurgical testwork is required prior to process design on the expanded treatment plant.
- Operating and capital cost estimates for the expanded plant need to be updated.

2 INTRODUCTION

2.1 ISSUER

This report is an independent technical report dated 02 March 2017 of the geology, exploration, mineral resource estimates, and mining scoping studies for the Irumafimpa and Kora gold deposits at the Kainantu project. The Kainantu property covers a total area of 405 sq.km and is located in the Eastern Highlands Province of Papua New Guinea, approximately 180 km west-northwest of Lae. (Figure 1)

In October 2016 Mr John Lewins, Chief Operating Officer of K92 Mining Inc. (“K92”), requested Nolidan Mineral Consultants (“Nolidan”), Australian Mine Design and Development (“AMDAD”), and Mincore Pty Ltd (“Mincore”) to prepare a report in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) incorporating the results of recently completed mineral resource estimates and mine scoping studies of the Irumafimpa and Kora gold deposits and plant upgrade studies.

Nolidan was engaged by K92 to update the mineral resource estimates for the Irumafimpa and Kora deposits which were previously reported in May 2015. AMDAD was initially engaged in 2016 by K92ML to compile a 3 Year Mine Plan for mining of the Irumafimpa deposit. Later in 2016 AMDAD was engaged to undertake a Scoping Study for the development of the Kora deposit. In conjunction with the Kora Scoping Study Mincore was engaged to carry out a detailed study on the potential expansion of the existing processing plant to treat 400,000tpa of ore primarily from the Kora deposit.

As part of the Irumafimpa and Kora studies AMDAD prepared conceptual cashflows to provide guidance in relation to the economic viability of those mine plans. Those cashflows are the basis of the Preliminary Economic Assessment presented in this Technical Report.

K92 intends that this report be used as an Independent Technical Report as required under Part 4 “Obligation to File a Technical Report” of NI 43-101 to support publicly disclosed information.

2.2 TERMS OF REFERENCE AND PURPOSE

At K92’s request, the scope of the report included the following:

- Site verification and review of project.
- Preparation of an Independent Technical Report prepared in accordance with NI 43-101.
- Update of the previously issued mineral resource estimate for the Irumafimpa and Kora deposits
- Description of mining and milling infrastructure at Kainantu.
- Summarize the results of the mining plan for the Irumafimpa deposit
- Summarize the results of the preliminary economic assessment (“scoping study”) of the Kora deposit
- Summarize the studies on upgrading the capacity of the process plant.

2.3 INFORMATION USED

This report is based on historical technical data provided by K92 to Nolidan. K92 provided open access to all the records necessary, in the opinion of Nolidan, to enable a proper assessment of the project and resource estimates. K92 has warranted in writing to Nolidan that full disclosure has been made of all material information and that, to the best of the K92’s knowledge and understanding, such information is complete, accurate and true. The report also summarises information provided in previous recent NI 43-101 reports:

Independent Technical Report and Resource Estimate, Kainantu Project, Papua New Guinea, dated 06 March 2015.

Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea, dated 01 May 2015.

Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea, dated 15 April 2016.

Additional relevant material was acquired independently by Nolidan from a variety of sources. This material was used to expand on the information provided by K92 and, where appropriate, confirm or provide alternative assumptions to those made by K92.

With respect to Items 6, and 9 through 13 of this report, the author has relied in part on historical information including exploration reports, technical papers, sample descriptions, assay results, computer data, maps and drill logs generated by previous operators and associated third party consultants. Historical documents and data sources used during the preparation of this report are listed in Item 27: References.

2.4 SITE VISIT BY QUALIFIED PERSONS

Mr. Anthony Woodward of Nolidan has visited the Kainantu Gold Mine twice. A two day visit from 12th November to 13th November 2014 when the project was on care and maintenance included a review of drill core and exploration data from the Kainantu project. The three day site visit from 22nd November to 25th November 2016 included a visit to the rehabilitated underground workings, current underground diamond drilling sites at 1247mRL and 950mRL, inspection of the treatment plant, and discussions with company site management. Underground development activities were occurring on 1205, 1220 and 1235 Levels of the mine.

Mr. Chris Desoe of AMDAD visited the Kainantu site from 8th June 2016 to 14th June 2016. At that time the project was in the initial stages of restarting, focussing on rehabilitation of the underground access and establishment of power and ventilation. Mr Desoe examined the surface facilities and various areas of the existing underground workings, and held discussions with the mining operations and planning personnel.

3 RELIANCE ON OTHER EXPERTS

The author has relied on reports, opinions or statements of legal or other experts who are not Qualified Persons for information concerning legal, environmental, political or other issues and factors relevant to this report.

4 PROPERTY DESCRIPTION AND LOCATION

The Kainantu property covers a total area of 405 sq.km and is located in the Eastern Highlands Province of Papua New Guinea, approximately 180 km west-northwest of Lae (Figure 1). The project is located at the approximate centre of the Project, at 6°06'25" S Latitude and 145°53'27" E Longitude.

The property comprises four exploration licences, EL470, EL693, EL1277 and EL1341, one mining licence, ML150, two mining easements, ME80 and ME81, and one licence for mining purposes, LMP78. Tenements are owned 100% by K92 Mining Limited ("K92ML") but there is an understanding in-place for a 5% share to be divested to the local landowners. Further information on this understanding is detailed in Section 4.3.1. Memorandum of Understanding (MOU). To the extent known by Nolidan, there are no option agreements or joint venture terms in place for the property. A tenement map is shown in Figure 1 and tenement details are summarised in Table 1.

The Project as described herein is 100% owned by K92 Mining Limited ("K92ML"); a company incorporated in Papua New Guinea, which is 100% owned by K92 Holdings (PNG) Limited ("K92PNG"), a 100% owned subsidiary of K92 Holdings International Limited ("K92 Holdings").

On August 21, 2014, Otterburn, K92 Holdings and the K92 Holdings shareholders entered into a Share Exchange Agreement, pursuant to which Otterburn agreed to acquire all of the issued and outstanding shares of K92 Holdings, from K92 Holdings shareholders, in consideration for issuing shares in the capital of Otterburn. However, after further consideration by the parties, it was determined that effecting a tri-party merger under BVI law was more appropriate in order to effect Otterburn's acquisition of K92 Holdings. Accordingly, Otterburn entered into an agreement with K92 Holdings, pursuant to which K92 Holdings will merge with a newly created British Virgin Islands subsidiary of Otterburn, and whereby the Otterburn will acquire all of the outstanding shares of K92 Holdings, in exchange for shares of Otterburn.

K92 (formerly Otterburn) is a company incorporated under the laws of British Columbia, Canada; the common shares of which are publicly listed on the TSX Venture Exchange.

Nolidan has not undertaken any title search or due diligence on the tenement titles or tenement conditions and the tenement's status has not been independently verified by Nolidan.

K92ML is the registered holder of the following tenements in PNG (MRA, 2016), as issued by the applicable government authorities in accordance with the PNG Mining Act 1992 (the "Mining Act"):

1. Mining Lease 150 ("ML150"), effective until June 14, 2024;
2. Mining Easements 80 and 81 ("ME80" and "ME81"), each effective until June 14, 2024;
3. Licence for Mining Purposes 78 ("LMP 78"), effective until June 14, 2024;
4. Exploration Licence 470 ("EL470"), effective until February 05, 2017;
5. Exploration Licence 693 ("EL693"), effective until February 05, 2017;
6. Exploration Licence 1341 ("EL1341"), effective until June 20, 2018. ;
7. Exploration Licence 1277 ("EL1277") which expired on May 20, 2009. The PNG Minister for Mining rejected K92ML's application for renewal on December 5, 2011. K92ML initiated legal action to compel the Minister for Mining to overturn the decision, but the court instructed the parties to instead try to reach an out-of-court settlement. Negotiations in that regard have to date been unsuccessful; and if not settled will revert to the courts for a decision.

The renewal of ML150, ME80, ME81, and LMP78 occurred immediately prior to the acquisition of K92ML by K92PNG.

K92PNG acquired K92ML from Barrick (Niugini) Limited ("Barrick") pursuant to an agreement dated June 11, 2014 (the "K92ML Purchase Agreement") (which closed March 6, 2015), for the sum of US\$2,000,000. Under the terms of that agreement K92PNG is obligated to make additional payments of up to US\$60,000,000 as follows:

- (i) US\$20,000,000 upon K92PNG determining 1,000,000 ounces of gold equivalent (based on in-situ and mined product classified as measured mineral resource, indicated mineral resource, probable ore reserve or proven ore reserve); and
- (ii) US\$5,000,000 upon upon K92PNG determining each additional 250,000 ounces of gold equivalent (on the same bases as stated above) up to an aggregate of 3,000,000 ounces.

The obligation to pay additional payments will cease on March 6, 2025.

The PNG National Government has expressed its desire to recommence mining on ML150 as soon as possible to deliver benefits to the local community, Provincial Government and Nation (Barrick 2014).

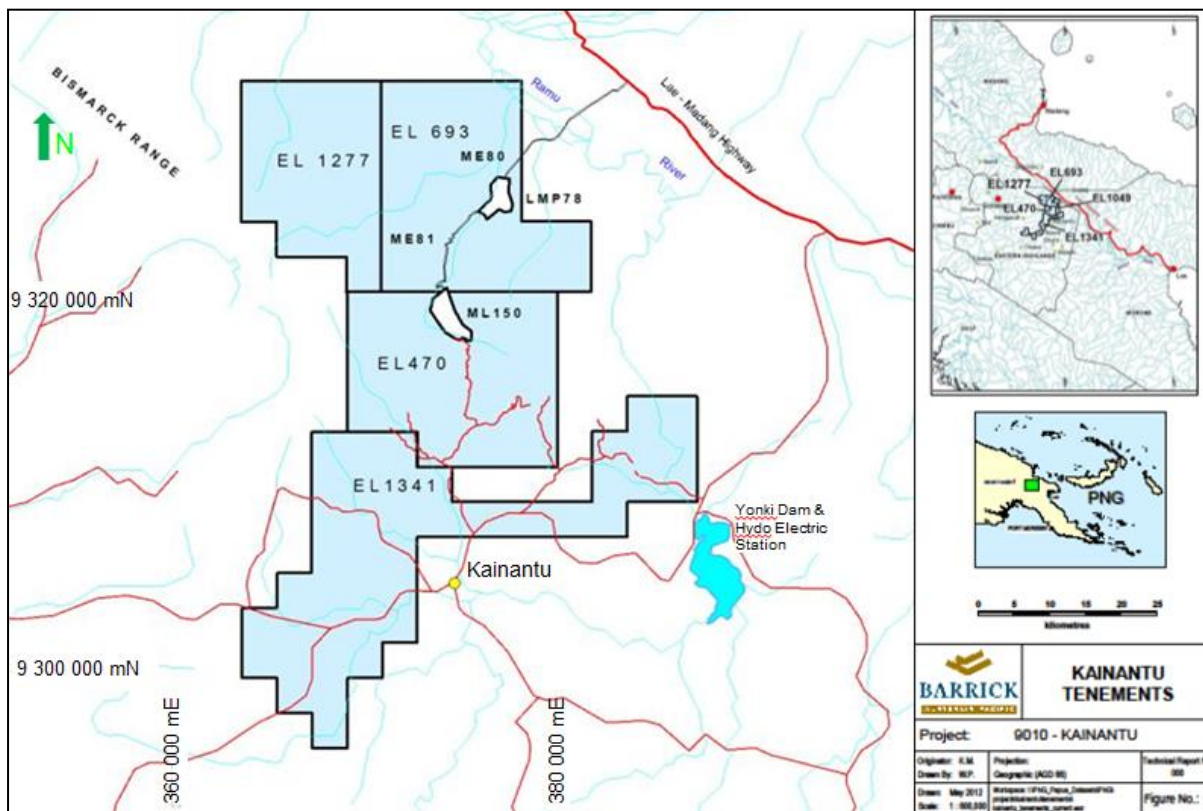


Figure 1. Kainantu Project Location and Tenements.

Source: Barrick 2014

Table 1. Project Tenure Details.

Tenement No.	Grant Date	Expiry Date	Renewal or Appln. Date	Area (km ²)	Rent (2016) Kina	Owners [#]
EL470	5/2/1982	4/2/2017	Current	95.0	13,160	K92ML
EL693	5/2/1986	4/2/2017	Current	95.0	13,160	K92ML
EL1277	30/5/2001	29/5/2009*	TBA*	68.3	9,400	K92ML
EL1341	21/6/2004	20/6/2018	Current	146.8	20,210	K92ML
ML150	4/6/2002	14/6/2024	Current	2.9	3,456	K92ML– 95% Landowners – 5%**
ME80***	14/6/2002	14/6/2024	Current	0.29	N/A	K92ML
ME81***	14/6/2002	14/6/2024	Current	0.35	N/A	K92ML
LMP78***	14/6/2002	14/6/2024	Current	2.1	2,512	K92ML

* Last approved renewal was to 29/05/2009. Application for renewal of tenure to 29/05/2011 was rejected by the Minister for Mining on 05/11/2011. K92ML undertook legal action to compel the Minister to overturn the decision and renew the lease. This was not successful and K92ML continues to negotiate settlement terms and the date for renewal.

** Ownership of ML150 currently 100% K92ML. 5% pledged under commercial terms to Landowners in the 2003 Memorandum of Understanding, and ratified by the 2014 K92ML Purchase Agreement.

*** ME80, ME81 and LMP78 are linked to the current ML150.

4.1 TENURE

4.1.1 Exploration Licence

An exploration licence may be granted for a term not exceeding two years, which may be extended under Section 28 of the Mining Act 1992 and Regulation. An exploration licence includes all land in the State, within the bounds of the exploration licence, including all water lying over that land.

An exploration licence authorizes the holder, in accordance with any conditions to which it may be subject, to:

- a) Enter and occupy the land which comprises the exploration licence for the purpose of carrying out exploration for minerals on that land; and
- b) Subject to Section 162, extract, remove and dispose of such quantity of rock and earth, soil or minerals as may be permitted by the approved programme; and
- c) Take and divert water situated on or flowing through such land and use it for any purpose necessary for his exploration activities subject to and in accordance with the provisions of the Water Resources Act (Chapter 205); and
- d) Do all other things necessary or expedient for the undertaking of exploration on the land.

The holder of an exploration licence is entitled to the exclusive occupancy for exploration purposes of the land in respect of which the exploration licence was granted.

Subject to Subsection (2), the Minister shall, on the application under Section 24 of the holder of an exploration licence, extend the term of the exploration licence for periods each of up to two years, where the Board advises the Minister that the holder has:

- a) Complied with the conditions of the exploration licence during the previous term of the exploration licence; and
- b) Paid compensation as required by this Act; and
- c) Submitted a programme for the proposed extended term which the Board recommends for approval under Section 26.

Where he considers that it is in the best interests of the State to do so, the Minister may refuse to extend the term of an exploration licence.

Where the Board is unable to give the advice required under Subsection (1) to the Minister, the Minister may, after receiving a recommendation from the Board, extend the term of the exploration licence for such period or periods of up to two years as he may determine, and include such further conditions of the exploration licence as he may consider necessary.

In considering whether the holder of an exploration licence has paid compensation as required by this Act, the Board shall rely on the advice of the Chief Warden

4.1.2 Mining Lease

A mining lease (ML) may be granted for a term not exceeding 20 years, which may be extended under Section 46 of the Mining Act 1992 and Regulation. A mining lease must be not more than 60 km² in area, and be in a rectangular or polygonal shape.

A mining lease authorizes the holder, in accordance with the Mining (Safety) Act (Chapter 195A) and any conditions to which the mining lease is subject, to; -

- a) enter and occupy the land over which the mining lease was granted for the purpose of mining the minerals on that land and carry on such operations and undertake such works as may be necessary or expedient for that purpose; and
- b) construct a treatment plant on that land and treat any mineral derived from mining operations, whether on that land or elsewhere, and construct any other facilities required for treatment including waste dumps and tailings dams; and
- c) take and remove rock, earth, soil and minerals from the land, with or without treatment; and

- d) take and divert water situated on or flowing through such land and use it for any purpose necessary for his mining or treatment operations subject to and in accordance with the Water Resources Act (Chapter 205); and
- e) do all other things necessary or expedient for the undertaking of mining or treatment operations on that land.

Subject to the Act, the holder of a mining lease -

- a) is entitled to the exclusive occupancy for mining and mining purposes of the land in respect of which the mining lease was granted; and
- b) owns all minerals lawfully mined from that land.

4.1.3 Mining Lease No 150 Renewal Conditions

Mining Lease No. 150 was renewed on 23 January 2015 for a period of 10 years to 13 June 2024. Conditions of the lease renewal are summarised below:

1. The lessee must comply with the Kainantu Mine Project Proposals for Development Tenure Extension Application 2014 dated 10 June 2014.
2. The mine must comply with the Mining Safety Act.
3. The Lessee must comply with all relevant legislation.
4. The change in control of K92ML must occur within 3 months of ML renewal.
5. The mine and mill must be completely refurbished by 31 December 2016 (this variation to the original condition 5 of the lease renewal was approved by the Mining Minister on December 07, 2015).
6. Operations and production from the Kora deposit must commence on or before 30 June 2018.
7. Develop a detailed rehabilitation and Mine Closure Plan at least 5 years prior to the planned closure of the mine or the expiration of the Mine lease or any extended Mining Lease, whichever occurs first.
8. Any public statement in relation to the Mining Lease and Kainantu Gold Project must also disclose any relevant conditions that form part of the extension of the Mining Lease.

4.1.4 Expenditure Commitments

The tenement package has current annual rents of PGK 85,868 and annual minimum expenditure commitments of PGK 1,435,000 under approved work programs for the granted tenements.

4.1.5 Reporting Requirements

Pursuant to the Mining Act (1992), license holders are required to provide reports to the Mineral Resources Authority ("MRA") as follows:

Mining Licenses

- Monthly Mineral Return – Submitted every calendar month from date of grant of lease, detailing production of minerals (if any), including quantity and value of ore mined/treated and the quantity and value of minerals recovered.
- Monthly Royalty Return - Submitted every calendar month from date of grant of lease, detailing minerals won that are shipped/exported, prices and exchange rates at time of sale, expenditure, and net revenue from which royalty is calculated and paid to landowner groups.
- Annual report – as for Exploration License.

Exploration Licenses

- Bi-annual prospecting report – submitted every 6 months from date of expiry, on cancellation and on surrendering EL. Summarises all works undertaken on or in connection with EL since the previous report.
- Bi-annual expenditure report - submitted every 6 months from date of expiry, on cancellation and on surrendering EL. Summarises all expenditure connected with acquisition and interpretation of exploration data on the lease.
- Annual report – submitted every 12 months from date of grant of lease. Provides detailed information on all work on, or in connection with the license. Includes aims of works, procedures applied and conclusions reached. All relevant data must be included.

4.2 ROYALTIES

The Mining Act 1992 (Act) provides that all minerals at or below the surface of any land (i.e. gold, silver, copper and other minerals) are the property of the State. K92ML, pursuant to the Mining Lease from the State, owns what is mined from the orebody.

The tenements are subject to royalties and interests in favour of the Government of Papua New Guinea in accordance with the Mining Act 1992 (Act). The holder of a mining lease or a special mining lease under the Act is required to pay a royalty to the State equal to 2% of either:

- the Free on Board (FOB) value of the minerals, if they are exported without smelting or refining in Papua New Guinea; or
- the Net Smelter Return from the minerals, if they are smelted or refined in Papua New Guinea.

No other royalty agreements exist over the tenement package.

While not strictly a royalty cost, the PNG government imposes a second cost on mining projects, that of the MRA Levy. This levy is 0.25% of mine revenue (there are no deductions allowed for concentrate transport, smelting and refining).

4.2.1 States Right to Acquire 30% Interest In Mining Projects

Under the laws and upon grant of a mining licence (ML) or a special mining licence (SML) the State may elect at its discretion to take, at sunk cost, up to a 30% participating interest in any major mineral development in PNG. Upon exercise of that option, the State will fund its share of capital and ongoing costs and the mine developer will be repaid its share of sunk costs.

In respect of ML150, the State waived its right to acquire a 30% interest in the existing mining licence when they were first granted and has no similar rights under the ML renewal process. However, the State retains the option in respect of the Exploration Licences should any be converted into a Mining Licence or Special Mining Licence.

4.3 CARE AND MAINTENANCE

In January 2008, Barrick sought to place the mine into care and maintenance. The basis of the care and maintenance application was that the mining operation was not economic at the market conditions existing at that time. Barrick submitted that it would undertake significant exploration on ML150 and surrounding tenements to prove up sufficient resources to enable mining operations to resume.

Barrick received approval to have the mine in care and maintenance via the Variation to the Approved Purposes for Mining Lease No. 150 dated 13 February 2009.

Barrick received an extension to its care and maintenance until February 2013, when the Mining Advisory Council determined that extension of care and maintenance was appropriate provided a Mine Closure Plan was submitted.

Mining Lease No. 150 was renewed on 23 January 2015 for a period of 10 years to 13 June 2024.

Conditions of the lease renewal are discussed in section 4.1.3

Since 01 May 2015 various consultants have been engaged by K92ML to review aspects of the mine and mill refurbishment. Rehabilitation by K92ML of the Irumafimpa mine, process plant and associated infrastructure commenced in late March 2016. Remedial work on the 800 Portal and Incline, the main mine access for the Irumafimpa mine, was completed in June 2016 with the upper working levels of the mine accessible and ventilation re-established. Refurbishment of the Kainantu Processing Plant was completed in September 2016 and the first batch of underground ore from Irumafimpa treated in October 2016.

4.4 OTHER SIGNIFICANT FACTORS AND RISKS

Environmental permitting, tailings disposal, mine closure plans, and landowner compensation agreements are discussed in Section 20: “Environmental Studies, Permitting, and Social or Community Impact” of this report.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PHYSIOGRAPHY

The Property lies within an area of mostly rugged topography, with transecting rivers forming lower lying areas. Elevations range from 400m to 1600m above sea level. Vegetation is mostly primary rainforest with areas of shifting agriculture in valley floors.

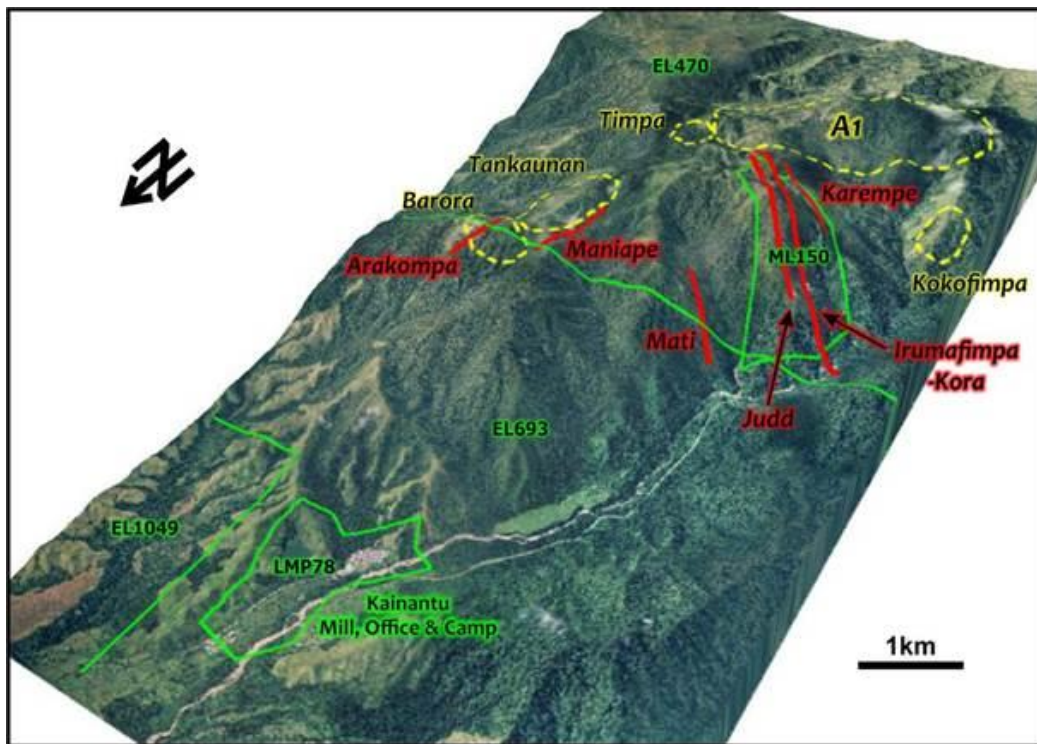


Figure 2. Oblique View of Northern Part of Property, Showing Relief and Location of Main Infrastructure.

Source: Barrick 2012

5.2 ACCESS

The property area is accessed by a two hour drive along the sealed Lae-Madang Highway from Lae. Lae is the capital city of the Morobe Province and second largest city in PNG. It is serviced by daily flights from Port Moresby and other PNG centres and also hosts the largest cargo port in PNG.

The property is serviced by a 10 km long formed access road from the Lae-Madang Highway, commencing at Gusap Airstrip to the Kumian Process Plant and Office facility. The access road crosses one single lane bridge at the Ramu River. From the process plant site, a formed haul road travels 6.5 km to the 800 Lower Portal of the mine. The haul road crosses three major single lane bridges.

Access and haul roads span 6m width and are constructed within two Mining Easements (ME's 80 and 81) commencing at the Ramu Bridge. The haul road rises 391m in elevation over its total length. These roads are graded and reformed generally twice a year in low traffic conditions, and have not deteriorated significantly in high rainfall seasons.

5.3 CLIMATE

The climate across the Property is variable due to topography. Hot temperatures and wet conditions characterize the climate at Kainantu. Daytime temperatures reach 30°C dropping to night time lows of 20°C. A pronounced wet season occurs between November and April, although rainfall is common throughout the year. Rainfall averages 235 mm/month during the November to April wet season, and 137 mm/month during the dry season. Annual rainfall averages approximately 2000 mm. Project operation/exploration is subject to the weather; reduced visibility when cloudy prevents operation of helicopters and heavy rainfall or earthquakes can trigger landslides.

5.4 LOCAL RESOURCES

The Property site offices are located 140 km from Lae, 21 km from Kainantu township and 56 km from Goroka (Table 2). Goroka is the Capital of Eastern Highlands Province and contains Local and Provincial Level Government Offices.

Table 2. Local Resources to Property

Local Resources	Lae (Morobe Province)	Goroka	Kainantu
Population:	~100,700	~18,500	~6,700
Elevation:	10m	1600m	1570m
Distance from Lae:	-	285km	170km
Distance to Property Site Offices	140	56	21
Airport:	Runway Length 2440m. 1 Runway;	Runway max 1646m. 2 x runways.	In use
Commercial air travel:	+ 11 flights daily	3 flights daily. 1 hr flight from Port Moresby.	No
Facilities:	Many	Schools, hospital, police station, district and provincial court, tertiary education, fuel stations, banks	School, hospital, police station, district court, fuel stations, banks. Local Level Government Offices.

5.4.1 Yonki Dam and Ramu Hydro Electric Power Station

Yonki Dam provides water for the Ramu Hydro Power Station and the Yonki Toe of Dam Power Station operated by PNG Power Ltd. The Dam commissioned in 1991 on the upper Ramu River, has a 335M m³ capacity, a 60m high earth fill dam wall with 680m long crest.

Mining Projects including Hidden Valley created a need for additional power output. The Yonki Toe of Dam Project was commissioned in 2013 to help meet that requirement.

Currently the Ramu 1 Hydro Power station is supplying 54 MW from three generators on to the Ramu Grid while the Yonki Toe of Dam supplies 14MW. They are supplemented by 4MW from the Pauanda Hydro Power station, 10MW from the Baiune Hydro Power station at Bulolo in Morobe Province and

a combined thermal generation capacity of 20MW from the diesel power stations in Lae, Madang and the Highlands centres, giving a total generation capacity of 102MW into the Ramu Grid (PNG Power website, 2014).

The grid serves Lae, Madang & Gusap in the Mamose Region, and Wabag, Mendi, Mt Hagen, Kundiawa, Goroka, Kainantu & Yonki in the Highlands.

5.4.2 Gusap Airstrip

The Gusap Airstrip is a fully licenced, international grass strip located in the Ramu Valley and maintained jointly by the project and Ramu Agricultural Industries mainly for use in emergencies and for charter flights.

5.5 INFRASTRUCTURE

The Kainantu mine is located within ML150 and the main Kainantu exploration camp and processing plant are located within LMP78 which is located within EL693. The Property includes all mine infrastructure, exploration camps, exploration data and diamond drill core.

The property is well supported by regional infrastructure, and contains all the necessary site infrastructure for mining operations

Underground mining at Kainantu operated from 2004 to 2008 and was based on mining of the Irumafimpa gold deposit. The majority of the mining infrastructure from that period remains in place.

The Kainantu processing plant is located approximately 7 km from the opening of the 800 portal which accesses the Irumafimpa Mine. The plant was on care and maintenance between December 2008 and September 2016. Simple processing technology was used and following crushing, screening and grinding, sulphide bearing material was separated from non-mineralized host rock by flotation and a gold-rich flotation concentrate sold. Further details of site infrastructure can be found in Section 13 Mineral Processing and Metallurgical Testing and Section 18 Project Infrastructure of this report.

6 HISTORY

Gold was discovered in the area in 1928 in the Kainantu alluvial gold areas, however modern exploration did not commence until the early 1980's. After the discovery of Irumafimpa, Highlands Pacific Limited ('HPL') focused on high grade Au telluride mineralization with little to minor work conducted on the porphyry Cu Au targets. HPL commenced mining operations on the Irumafimpa deposit in 2005.

Barrick purchased the tenement package from HPL in late 2007 and concentrated on increasing resources at Irumafimpa-Kora and discovering economic porphyry Cu-Au mineralization. There has been a significant amount of exploration on the property by various owners, which is summarised in Table 3. The operation was on care and maintenance between January 2009 and August 2016.

Table 3. Kainantu exploration statistics, Barrick and historical.

Tenement	Drillholes	Drill Metres	Drill Samples	Stream Sediments	Rock Chips/Trench	Soils Samples	Pan Concentrate	Unknown
Barrick								
ML150	30	11497	10522		4/-	8		
EL470	11	6072	6039		2077/12	926		
EL693	0	0			26/65			
EL1277	0	0			141/-			
EL1341	1	530	491	2	939/-	404		

Historical								
ML150	641	43110	26456	25	185/719	549	12	4
EL470	19	3084	1216	947	903/111	1196	486	5
EL693	46	11694	887	294	340/452	470	201	
EL1277	0	0		367	159/211	624	178	
EL1341	39	3365	2113	1026	2627/168	2802	62	890

6.1 PREVIOUS OWNERSHIP

EL470 was granted to Renison Goldfield Consolidated (PNG) (“RGC”) on 5th July 1982 as PA470. The area of EL693 was granted to RGC as PA462 and held in joint venture between RGC and Kafenu Mining until 1986, when a renewal application was rejected. The area was granted to RGC on 29 December 1986 as EL693.

RGC entered a Joint Venture over the EL’s 470 & 693 with Highlands Gold Resources Limited (“HGL”) in 1989, with HGL as the Operator. In 1994 RGC withdrew from the joint venture and the tenures became the sole property of HGL. When HGL was restructured in 1996, the new company Highlands Pacific Resources Limited (“HPL”) inherited the properties.

The properties were joint ventured between HPL and Greater Pacific Gold NL (“GPG”) from 1996 to 1998 with GPG as the Operator. This agreement was succeeded by a joint venture between HPL and Nippon Metals and Mining Company (“Nippon”) commencing in 1999, with HPL as the Operator and Nippon as the Funder. Nippon withdrew from the joint venture in late 2000.

In the following years, HPL systematically increased the size of its tenement package with applications granted for tenements in 2001 (EL11277), 2002 (ML150, LMP78, ME80, and ME81), 2004 (EL1341), 2005 (EL1399) and 2006 (EL1400). Barrick purchased the Kainantu tenement package from HPL in December 2007 through its 100% owned subsidiary Placer Dome Oceania Limited. This entity’s name was subsequently changed to Barrick Kainantu Limited (now “K92 Mining Limited”) which was the most recent holder of the Kainantu package tenements.

At the time of the purchase by Barrick, the package included seven exploration licences; EL470, EL693, EL1049, EL1277, EL1341, EL1399 and EL1400; one mining licence, ML150; two mining easements, ME80 and ME81; and one licence for mining purposes, LMP78. During its term of operations, Barrick surrendered the EL’s 1399, 1400 and 1049; and added two exploration license applications; ELA1898 and ELA1899. These two applications were later dropped in late 2013.

In November 2011, an application for renewal of EL1277 was rejected by the PNG Minister for Mining. Barrick commenced Court action to dispute this decision in Court. No settlement has been reached out of Court, and the status of EL1277 remains subject to negotiation.

The current total area of the tenement package is approximately 405 km².

6.2 HISTORICAL EXPLORATION 1928-2012

The Historical Exploration up to 2007 described in this section is summarised from Smith (2008).

Ned Rowlands, an Australian prospector, first discovered gold in the Kainantu area in 1928 on a small creek draining into Abinakenu Creek. From 1928 to 1940, approximately 102 kg of gold was reportedly won as alluvial gold. Production ceased during WWII and did not resume in the Kainantu area until 1947. Between 1947 and 1972 alluvial gold production from the Kainantu area totalled 772.8 kg fine gold and 58.9 kg of silver.

Between 1948 and 1952, copper was discovered at Yonki Creek. In 1955, prospectors worked this small lode, which contained the secondary copper minerals malachite and covellite. Approximately 8 tonnes of handpicked ore grading 8% copper was shipped to Australia for processing.

The southern end of the Irumafimpa lodes was discovered some time prior to 1967. In 1967, Ken Reihder and Ray Frazer started working Prospect Claim 6 for copper and gold. The workings, known as the Kora mine, produced about 1,000 tonnes of gold and copper ore between 1967 and 1970. The ore processed through a five-head stamp mill is recorded as averaging three ounces recovered gold to the tonne.

Between 1969 and 1972, most reconnaissance work concentrated on the Yonki copper gold lode, which lies south of Abinakenu Creek. Two samples were collected near Yar Tree Hill from auriferous vein quartz reef.

In early 1982, general reconnaissance was carried out in the area by stream sediment and rock chip sampling. The work confirmed the presence of gold in alluvium and rocks over a wide area. Further work was recommended.

In 1984, further reconnaissance revealed that alluvial gold is present in virtually all of the creeks draining a NNW trending ridge between Abinakenu and Asupuia village. Later in 1984 and 1985, various programs were carried out to sample the ridge south and east of the Asupuia – Abinakenu ridge. Later, in 1989, Highlands Gold carried out further sampling east of Mt Kanuna. One party attempted to walk up the main ridge between Asupuia and Abinakenu, only to be turned back by hostile landowners.

Prospecting Authority (PA) 693 was initially granted to RGC (PNG) Pty Ltd on 24th December 1986 and renewed for a further two-year period on 29th December 1988. In July 1989, Highlands Gold Resources N.L. (HGL) entered into a joint venture agreement with RGC to earn a 50% interest in EL693. Expenditure commitments were fulfilled and HGL assumed its share of the ownership in 1994.

Highlands Gold actively explored the Kainantu properties from 1989 to 1994. Their initial work consisted of mapping, sampling and trenching. The work delineated several high grade gold targets including Irumafimpa, Maniape and Arakompa. Exploration was focused on Irumafimpa where six diamond drillholes were drilled (for a total of 1,402m) during the last quarter of 1992. These drillholes returned some very encouraging gold results. To follow up on these, further extensive trenching, mapping and sampling was conducted. During the last half of 1993 a geophysics program comprising magnetics, CSAMT and IP was implemented, and a further 15 diamond drillholes (for a total of 3597.3m) were completed.

In 1996, Highlands Gold was taken over by Placer. In June of that year, Placer floated the exploration assets of Highlands Gold off into a new company called Highlands Pacific. The Kainantu tenements became part of the core assets of Highlands Pacific. That same year Highlands Pacific joint ventured the property to a junior exploration company, Greater Pacific Ltd. and this company became operator and manager of the project. Greater Pacific however struggled to make any exploration progress on the property, due to landowner difficulties and funding shortfalls. By the end of 1998 it became obvious that Greater Pacific would be unable to meet their joint venture obligations. At that time Highlands Pacific staff reviewed all of the previous exploration conducted within the Kainantu district. This review indicated a very high potential for discovery of a significant tonnage of high-grade gold mineralization within the Irumafimpa, Maniape and Arakompa vein systems. A follow up work program, to be managed by HPL, was proposed.

The joint venture with Greater Pacific was terminated early in 1999, and subsequently a joint venture with Nippon Metals and Mining Company was ratified. Under the terms of this agreement, Nippon was to sole fund the initial stages of exploration whilst HPL manage the exploration programs. In 1999 the Nippon-Highlands joint venture drilled 14 holes in the Irumafimpa area with reasonable success. The following year the venture drilled another 12 holes to further define the Irumafimpa resource. Nippon withdrew from the joint venture in late 2000 and Highlands Pacific subsequently regained 100% of the project.

Local people started mining zones of the Irumafimpa zone in 1992 after the discovery of the outcrop by Highlands Gold. Surface mining at all of the three mineralized structures continues today, and provides a major source of income for the local people.

Modern development of the Irumafimpa deposit commenced in 2004, and the mine has struggled to achieve planned mined grades, through a combination of complexity of geology and unplanned dilution. The net effect of not achieving planned head grades was a shortfall in metal production resulting of purchases of spot gold to enable the company to meet its hedging requirements. Continued shortfalls in metal production pushed Highlands Pacific to consider a sale of the assets, which was acquired by Barrick Gold in December 2007.

Barrick conducted Exploration from 2008 to August 2012. In addition to resource evaluation of the Kora deposit their priority was discovery of a large porphyry system. Land access issues were the main challenge to implementing exploration activities. Access to the high priority A1 project was only available for the 6 months before Exploration was halted by the decision to divest the project

Irumafimpa-Kora is an advanced property and current Resources are described in Section 14 of this report. Figure 10 shows the location of the prospects described below in relation to property boundaries.

6.3 ML150 (IRUMAFIMPA, KORA, JUDD AND KAREMPE)

6.3.1 Kora and Irumafimpa

A representative long section is shown in Figure 3. A total of 24 diamond holes were drilled by Barrick at Kora , including a single hole at the nearby Karemppe vein system (Figure 4). Drilling confirmed the continuity of the Kora Lode and confirmed that the overall system has a vertical extent to >800m. Significant intercepts are summarised in Table 4 and Figure 5 shows the consistency of grade intersected at Kora.

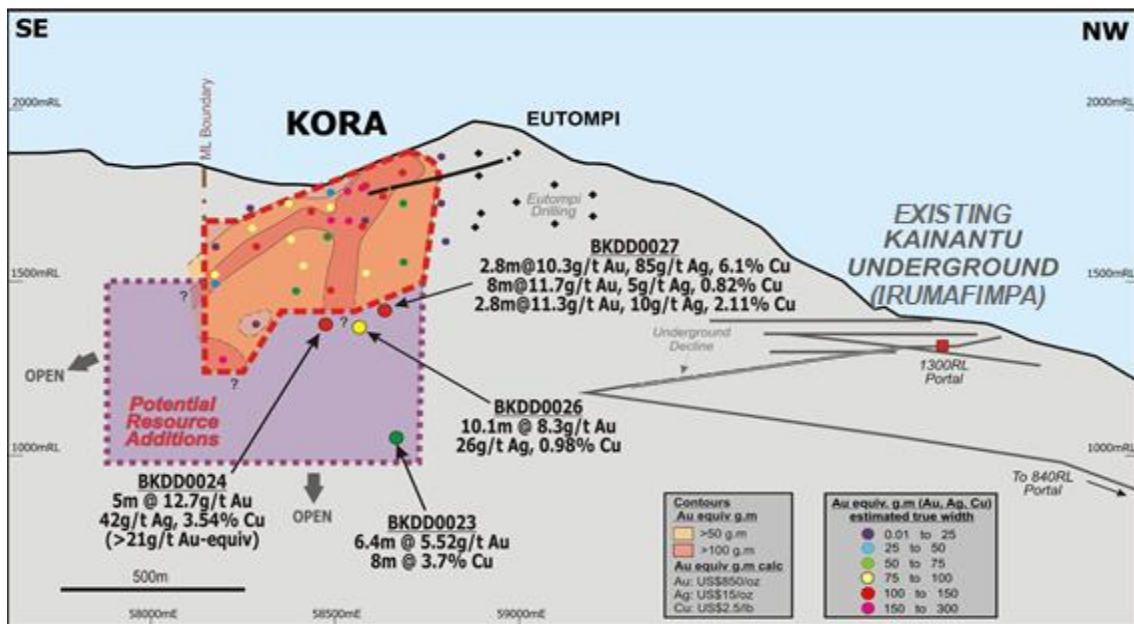


Figure 3. Kora long section showing potential depth extents of mineralization.

(Source Barrick 2014)

Prospect location in relation to property boundaries is shown in Figure 10

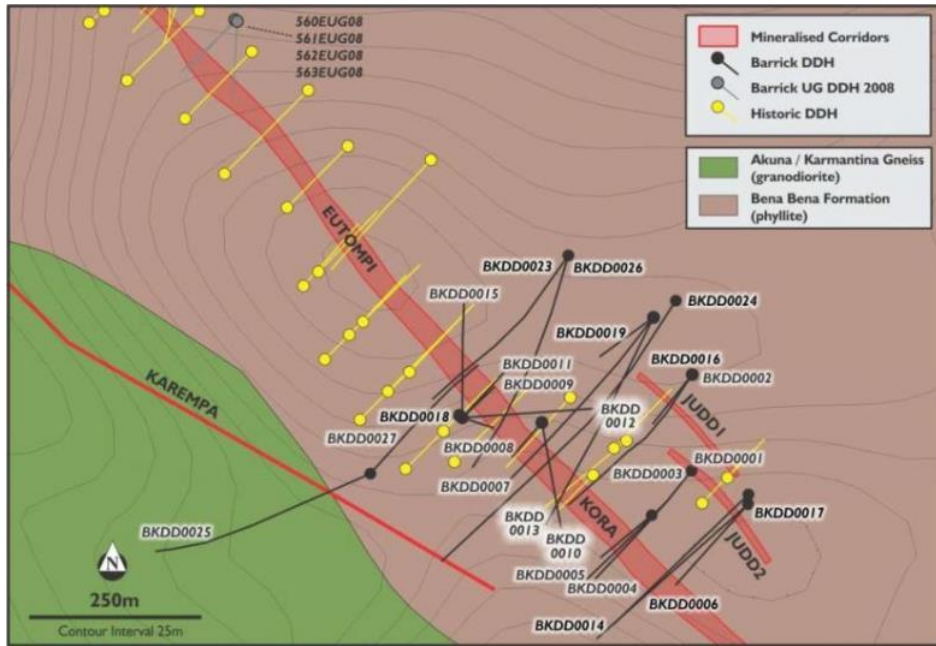


Figure 4. Local geology and Barrick drillholes location plan at Kora and Karempa.

(Source Barrick 2014)

Prospect location in relation to property boundaries is shown in Figure 10

Table 4. Significant intercepts, Barrick drilling (> 1 g/t Au) at Kora.

Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Cu (%)	Metal Accumulation. Factor (gm)
BKDD0001	279	282	3	5.16	8.37	15.48
BKDD0001	299	303	4	6.3	8.04	25.20
BKDD0002	113.3	116.3	3	347.73	0.21	1043.19
BKDD0005	138.1	146	7.9	20.14	6.74	159.11
BKDD0005	156	159	3	8.33	7.96	24.99
BKDD0005	173	182.7	9.7	4.64	0.53	45.01
BKDD0006	575.2	581	5.8	6.76	7.94	39.21
BKDD0007	515.15	522.51	7.36	22.78	2.22	167.66
BKDD0008	87.5	89.5	2	53.36	4.8	106.72
BKDD0008	123.38	130	6.62	9.57	0.44	63.35
BKDD0009	218.87	221.36	2.49	207.09	3.04	515.65
BKDD0009	225.6	231.4	5.8	25.05	2.25	145.29
BKDD0010	104.8	107	2.2	101.7	15.07	223.74
BKDD0011	38	47	9	19.17	1.08	172.53
BKDD0013	488	492	4	228.91	0.45	915.64
BKDD0015	62.4	73	10.6	184.78	1.85	1958.67
BKDD0023	945	951.4	6.4	5.55	0.46	35.52
BKDD0024	619	624	5	12.94	3.54	64.70
BKDD0026	582.9	593	10.1	8.21	0.97	82.92
BKDD0027	472	480	8	11.97	0.82	95.76

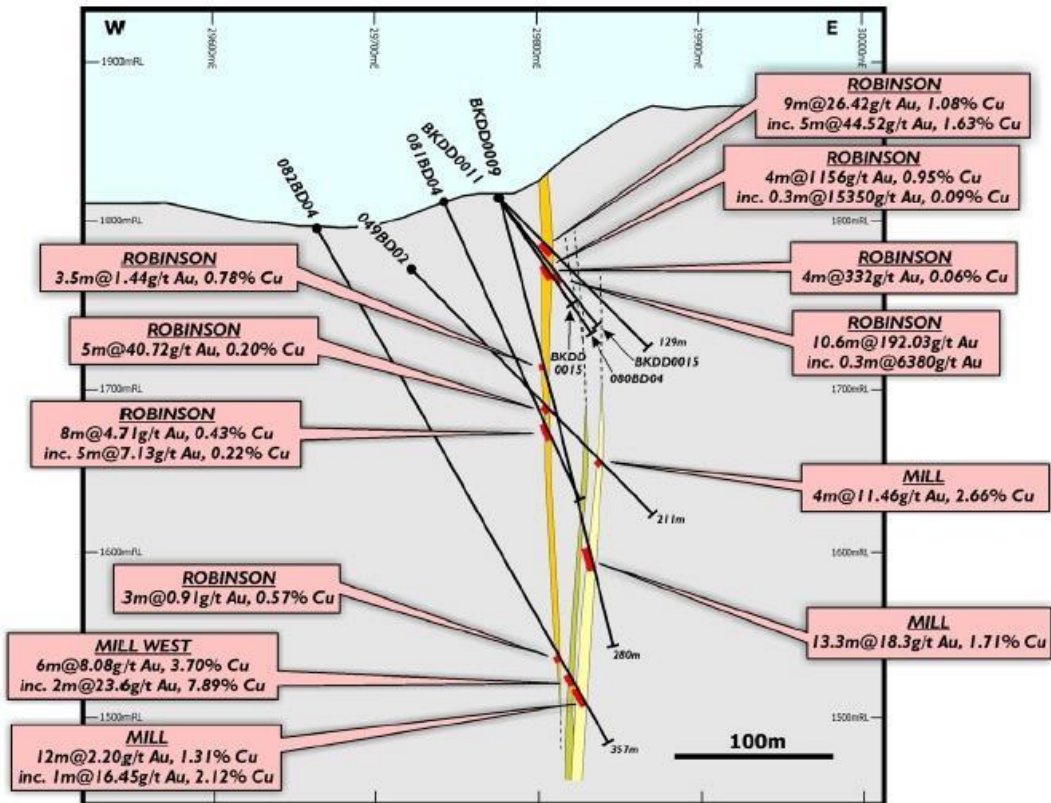


Figure 5. Cross section 58600mN at Irumafimpa showing consistency of high grade, particularly within the Robinson lode.

Yellow colouring indicates the Mill lode and orange colouring the Robinson lode Prospect location in relation to property boundaries is shown in Figure 10

(Source Barrick 2014)

A review of >100g/t Au and >10% Cu intersections showed greater continuity of high grade at Kora when compared to Irumafimpa (Figure 6; Figure 7).

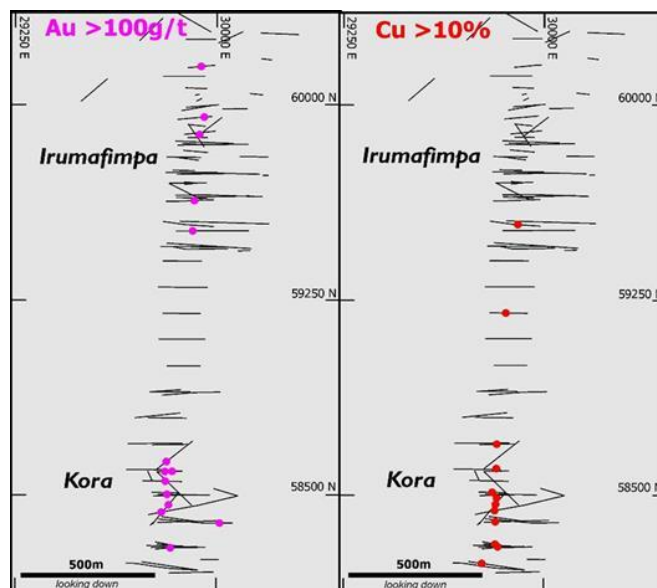


Figure 6. Surface drilling traces showing surface projections of >100g/t Au and >10% Cu.

(Source Barrick 2014)

Prospect location in relation to property boundaries is shown in Figure 10

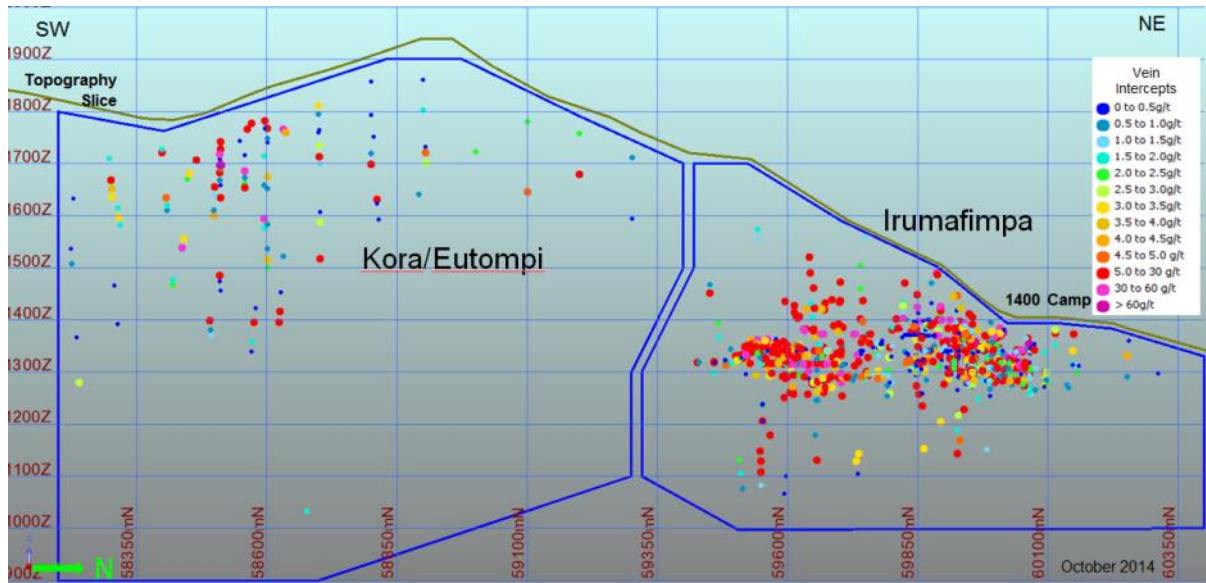


Figure 7: Long section view of Kainantu Resource Areas with Vein Composites colour coded for Au Eq

In addition, veins are wider and likely more continuous than those at Irumafimpa. Mineralization is open in all directions. There is also strong potential below the Eutompi area and high grade mineralization to the southeast where structures hosting Kora lodes were identified by Barrick mapping 800m beyond the ML boundary.

Potential also exists to define additional vein hosted resources within the ML at Judd and Karempe.

6.3.2 Judd

Judd, a narrow intermediate vein system located 200m east of and parallel to Kora was partially tested by Barrick holes designed to test the Kora lode at depth. This drilling on the Judd lode returned several highly encouraging intersections of the Judd lode including 1m @ 4.1 g/t Au, 9m @ 8.8 g/t Au and 1.1% Cu and 3m @ 278 g/t Au (Figure 8). Barrick considered that holes designed to specifically target the Judd lode would have the potential to yield high grade resources within close proximity to the immediate mine environment.

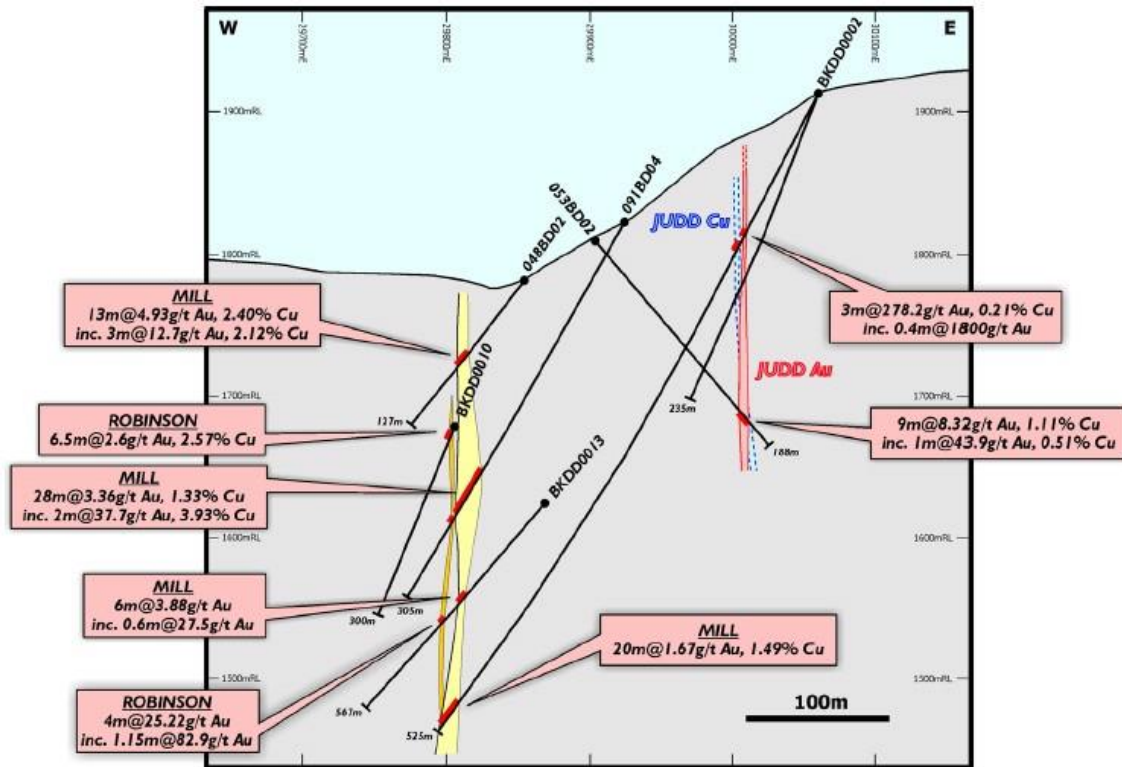


Figure 8. Cross section showing narrow mineralization intersected at Judd lode.

Pink colouring shows the Judd lode, yellow colouring the Mill lode and orange colouring the Robinson lode.

(Source Barrick 2014)

Prospect location in relation to property boundaries is shown in Figure 10

6.4 HISTORICAL EXPLORATION REVIEWS

Barrick engaged independent consultants Corbett (2009) and Tosdale (2012) to carry out exploration targeting reviews for the Kainantu project. Their findings are included below as they represent independent assessment of the potential of the Kainantu property. Barrick also conducted several internal reviews of the exploration prospectivity. Key findings are summarised below.

6.4.1 Corbett (2009)

Corbett provided review and recommendations for existing exploration targets and highlighted that the early stage potential and that many areas of interest had received little follow up:

Irumafimpa-Kora - The Irumafimpa structure hosts low sulphidation quartz-sulphide Au + Cu mineralization typical of that which might form marginal to porphyry Cu-Au intrusions. Continued data analysis should seek to identify any link structures, which may form steep plunging shoots under conditions of strike-slip deformation. A possible porphyry Cu-Au at the fluid upflow is also recognised as a target, below.

Kesar - While it is stressed exploration at Kesar Creek remains in the very early stage, the project is rated with a low priority. The current programme of geological mapping and sampling should continue to map out the Kesar Creek prospect which might be accessed at the end of this program.

Kokofimpa - Although in the early stages of investigation, Kokofimpa displays many aspects of hydrothermal alteration and mineralization commonly associated with porphyry Cu-Au systems and so warrants continued investigation. Further work recommended.

Other targets listed in order of declining merit:

- The Bilimoia target lies SW of the original Timpa Cu-Au breccia in the vicinity of a Barrick Mo in soil anomaly and represents the SE strike extension of the Kora vein. It is targeted as a possible intrusion-related upflow for the Kora-Irumafimpa low sulphidation deep epithermal Cu-Au vein mineralization.
- Kora Deep occurs as the deeper portion of the Kora-Irumafimpa
- Barora, which represents an intense magnetic high and site of mixed anomalous geochemistry, has long been targeted for possible blind porphyry Cu-Au mineralization
- The Mesoan vein system, which is parallel to and NE of Irumafimpa, is evidenced at the surface by artisan workings and so warrants follow up geological mapping and sampling when access has been gained.
- The Kompene diatreme is rimmed by anomalous Au, Cu, Ag, Zn, Pb and Mo geochemistry as a theoretical site for carbonate-base metal style Au mineralization
- The ridge NNE of Maniape which contains anomalous Au, Cu, Pb and As geochemistry has probably not been prospected and warrants follow up.
- There are strong As anomalies in the Mainape-Arakompa-Kampene area which require verification, as much of this area was prospected in the 1989 program. If these soil anomalies are valid, further follow up is required.

6.4.2 Tosdale (2012)

The summary and recommendations were offered by Tosdale (2012) regarding future exploration programs are included below:

Different levels of separate magmatic-hydrothermal systems underlie the Tankuanan, Timpa, A1 (Moly Hill), and Breccia Hill prospects. Significant exploration on Tankuanan has failed to identify a potentially economic porphyry Cu system, and further exploration expenditure does not appear warranted. The only exception would be a program to test for higher grade that might be accessible under the potentially inclined late mineral pebble breccia and late mineral sericitically altered porphyry that outcrops on the west side of the Tankuanan property.

In contrast, the lack of systematic exploration on the Breccia Hill and A1 (Moly Hill) prospect coupled with geologic evidence suggest that these prospects could contain mineralized systems. What is also unknown is the deposit types or the potential depth beneath the current surface. At least at the Timpa prospect, the presence of a hydrothermal system is evident, as the geologic and geochemical data confirms that it represents a separate system from the nearby Tankuanan porphyry prospect. However, at the current outcrop levels, the mineralized breccia may represent a level of a porphyry Cu system that lies above the level of significant Cu and Au mineralization.

6.5 HISTORICAL ESTIMATES

All mineral resources reported in this section are provided for informational purposes only.

6.5.1 Historical Estimates Irumafimpa-Kora

Historical estimates for the Irumafimpa and Kora deposits have previously been prepared before K92 entered into an agreement to acquire an interest in the property that contains the deposit.

Early HPL resources reported in accordance with JORC 2004 were prepared by independent consultants Hackchester Pty Ltd (2005) and Mining Associates Pty Ltd (2006). Numerous historical estimates and financial models were prepared by Barrick for Irumafimpa-Kora. K92 is not treating the historical estimates as current mineral resources or mineral reserves. These historic resources are not

reported here as they are superseded by the current Mineral Resource estimate contained in Section 14 of this report.

The current resource statement presented in Section 14 in this document supersedes all previous resource figures.

6.5.2 Historical Estimates – Arakompa and Maniape

Historical estimates have also been reported for the Arakompa and Maniape deposits and are shown in Table 11.

Table 5 Historical estimates reported for the Arakompa, Maniape and Aifunka deposits

Deposit	Category	Historical Resource *			
		cut-off g/tAu	t	Au g/t	Au Oz
Maniape ⁽¹⁾	Unclassified	1	7,990,000	2.2	557,000
Arakompa ⁽²⁾	Unclassified	5	2,753,000	9.0	798,000
Aifunka North ⁽³⁾	Unclassified	-	851,000	3.7	102,000
Aifunka South ⁽³⁾	Unclassified	-	214,000	1.8	12,000
Total			11,808,000	5.6	1470,000

(1) Barrick (2008). Method unclear. Little Ag, no Cu. Based on 16 drillholes.

(2) HPL (2002). Method: Polygonal narrow vein model. Little Ag, no Cu. Based on 18 drillholes. No work since.

(3) HGL (1992). Method. Polygonal model. Highlands Gold Annual Report 1992 (number 964).

* A qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves. There was no classification assigned so no comparison can be made to resource categories under JORC Code 2012 or CIM Definition Standards. Nolidan has quoted the historical estimates for information and targeting purposes only.

K92 is not treating these historical estimates as current mineral resources or mineral reserves. Before these historical estimates can be determined if all or any can be classified as current, a Qualified Person for K92 must review of the existing database, QAQC, with appropriate data verification procedures, and the geological model. Because the historical estimates were unclassified, it is likely that additional confirmatory and infill drilling would be required. No exploration is planned on these prospects in the current 12 month plan as the Company will be focusing its efforts on the mine and mill refurbishment, close spaced drilling at Irumafimpa and drilling of the Kora extension.

6.6 HISTORIC PRODUCTION

6.6.1 Irumafimpa-Kora

Smith and Thomas (2008) visited the Irumafimpa Mine site to analyse the causes of the poor reconciliation from mineral reserve to grade control and again from grade control to final mill reconciled production. Due to the difficulty of obtaining comprehensive data from site Smith and Thomas (2008) report that it was not possible to produce a full mine reconciliation to the Barrick standard, however they note; site staff did make available a number of comparative tables that provide an adequate proxy for mine reconciliation. Table 6 presents a stope by stope comparison of mineral reserve estimates against grade control estimates for stopes being mined or planned in November 2008. It is evident that grade control (GC) has identified significantly less tonnes, grade and metal than was reported in the ore reserve (OR), as shown by the GC:OR ratios.

Table 6. Irumafimpa - Stope by stope comparison of Reserve estimate against Grade Control estimate (November 2008)

RESERVE*		GRADE CONTROL - STOPE ENVELOPE	
Vein	Block Model - Sept 2006	Vein	RECOVERABLE STOPE ORE

A	COMPLETED STOPES	Width	Grade	Block Width	Grade	Tonnes	Ounces	Width	Grade	Designed Mining Width	Grade	Tonnes	Ounces
1	17L-17ShrM5	1.1	34.3	1.3	26.3	3,236	2,736	0.6	19.2	1.0	12.2	1,940	763
2	19L-13ShrM4	1.5	18.2	1.7	16.1	4,499	2,332	0.6	13.3	1.0	8.0	1,653	423
B	ACTIVE STOPES												
1	17L-14ShrM5	1.3	11.5	1.5	10.0	3,983	1,284	1.2	10.1	1.4	8.7	1,726	481
2	17L-15/16ShrM4	1.4	26.4	1.6	13.2	4,381	1,859	1.4	13.2	1.6	6.6	4,381	930
3	19L-14/15/16ShrPu	1.5	17.0	1.7	15.0	13,124	6,312	1.5	8.2	1.7	7.2	8,663	2,003
5	19L-11CAFm4	4.7	19.6	5.2	17.7	13,324	7,575	1.1	17.2	1.1	16.4	1,137	599
6	19L-26/27LHM6	1.1	15.1	1.3	12.7	6,529	2,672	1.5	10.0	2.4	6.2	7,748	1,554
7	20L-17/19ShrM1	1.1	20.3	1.3	17.1	9,980	5,483	1.0	10.5	1.2	8.7	6,749	1,882
	Sub-Total	2.2	18.4	2.5	15.3	51,321	25,186	1.3	10.3	1.7	7.6	30,404	7,448
C	TO BE COMMENCED												
1	22L-20/21ShrM5	0.7	8.9	1.0	6.2	5,164	1,032	1.5	9.9	1.7	8.7	7,646	2,131
2	22L-22ShrM5	1.7	26.7	1.9	23.9	4,966	3,822	1.4	8.8	1.6	7.6	2,441	600
3	21L-17/18ShrM3	1.1	8.1	1.3	6.8	3,251	710	1.3	7.0	1.5	6.0	5,309	1,028
4	19L-12CAFm4	1.9	25.0	2.1	22.5	5,303	3,842	1.1	17.2	1.3	13.2	1,791	761
5	19L-18/19ShrM1	1.1	20.7	1.3	17.4	6,522	3,658	1.3	9.1	1.5	7.9	5,369	1,356
6	18L-12CAFpu	0.5	35.0	1.0	17.8	2,582	1,479	1.4	11.1	1.6	9.7	1,263	393
7	18L-14/15ShrPu	1.4	23.1	1.6	20.3	8,474	5,530	1.4	10.3	1.6	9.0	5,080	1,475
8	18L-16ShrPu	1.5	22.2	1.7	19.6	4,506	2,845	1.5	7.6	1.7	6.7	4,835	1,041
9	19L-24/25LHM6	1.0	52.4	1.5	34.7	7,648	8,531	2.1	12.8	2.3	11.7	9,049	3,390
	Sub-Total	1.2	26.0	1.5	20.2	48,414	31,450	1.5	10.1	1.7	8.9	42,783	12,176

*RESERVE based on September 2006 Block model

GC : OR	tonnes	grade	ounces
active and completed stopes	59%	50%	30%
upcoming stopes	88%	44%	39%

Table 7 presents mill production for the life of the Irumafimpa mine. On a qualitative basis a negative reconciliation on grade from grade control to mill production is evident. The grade control grades in Table 6 are of the order of 8 to 9 g/t Au whereas the back calculated mill head grade for 2008 was 5 g/t Au.

Table 7. Historic mill production for Irumafimpa

Year	Mill tonnes	Head grade Au g/t	Contained Oz Au
2006*	104,272	8.00	26,819
2007*	141,452	7.00	31,835
2008**(6 months)	61,532	5.02	9,939
LOM Total	307,256	6.94	68,593
* From Highlands Pacific annual reports			
** Barrick Ownership (mining and processing ceased in January 2009)			

6.6.2 Other sites

Illegal mining is an important activity for the provision of local income. It is understood the illegal mining is restricted to the oxidised upper portions of mineralized prospects where gold is easily obtainable in its native form. The sites and extent of illegal mining have not been examined in this report.

6.7 HISTORICAL PERFORMANCE AND RECONCILIATION REVIEWS

The operations at Irumafimpa-Kora were suspended in January 2009. A general timeline of the operations is shown in Table 22. Nolidan notes that there were several historical reviews into the poor performance of operations with recommendations for improvements including:

- A full technical review by SRK in 2006.
- Mining Associates (2006)
- Clark (2007) conducted a Technical Review of Mine Geological Systems in July 2007.
- A review by Gauthier and Pridmore (2007) of Barrick which included review of geology and resource issues in October 2007

- A mine reconciliation review by Smith and Thomas (2008) of Barrick.

Table 8. Summary operations timeline for the Project

From	To	Irumafimpa Operations History (ML150)
January 2004		Highlands Pacific DFS approved by Mineral Resources Authority
2005	October 2007	Kainantu Gold Mine operated as Highlands Kainantu Limited (HKL)
November 2007		Barrick purchased the Kainantu project.
January 2008	June 2008	Barrick suspended mining operations from January to June 2008 in order to improve safety in line with Barrick standards. Technical aspects of operation also reviewed and implementation of some changes commenced
July 2008	January 2009	Mining restarted in July 2008 and was halted permanently in January 2009.
January 2009	December 2009	Exploration of epithermal and sulphide veins continued on the ML until June 2009, and then halted due to review of exploration priorities.
January 2010	current	Project on Care and Maintenance, limited exploration on EL's

Nolidan noted in its report that the Kainantu operations experienced significant problems with reconciling resource estimates of the Irumafimpa lodes with head grades. Mine geologists found it difficult to identify continuous mineralized structures and consequently stope development between levels was frequently on splays off the main veins resulting in mining of waste when the vein splays died out. Irumafimpa stope mapping and sampling plans show significant grade variability along strike in the shrink stopes and skilled geological support will need to be maintained.

Selection of treatment plant feed from development headings will require more assay control and less reliance on visual assessment as it appears that development did not always mine to the limits of the mineralized structures.

A thorough understanding of the controls on gold mineralization and the gold distribution within the mineralized structures will help control mine dilution. Attention to detail in grade control sampling will be a necessity.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Kainantu property is located within the New Guinea Thrust Belt, close to its northern contact with the Finisterre Terrane (Figure 9). The contact is marked by the northwest trending Ramu-Markham Fault, a major suture zone that marks the northern margin of the Australian Craton. The New Guinea Thrust Belt records an early Miocene or older ductile, tight folding event that was followed by middle Miocene intrusions. Late Miocene regional scale low-angle thrust faulting followed, associated with the collision of the Finisterre Terrane. The belt is characterised by a number of north-northeast trending fault zones that commonly host major ore deposits.

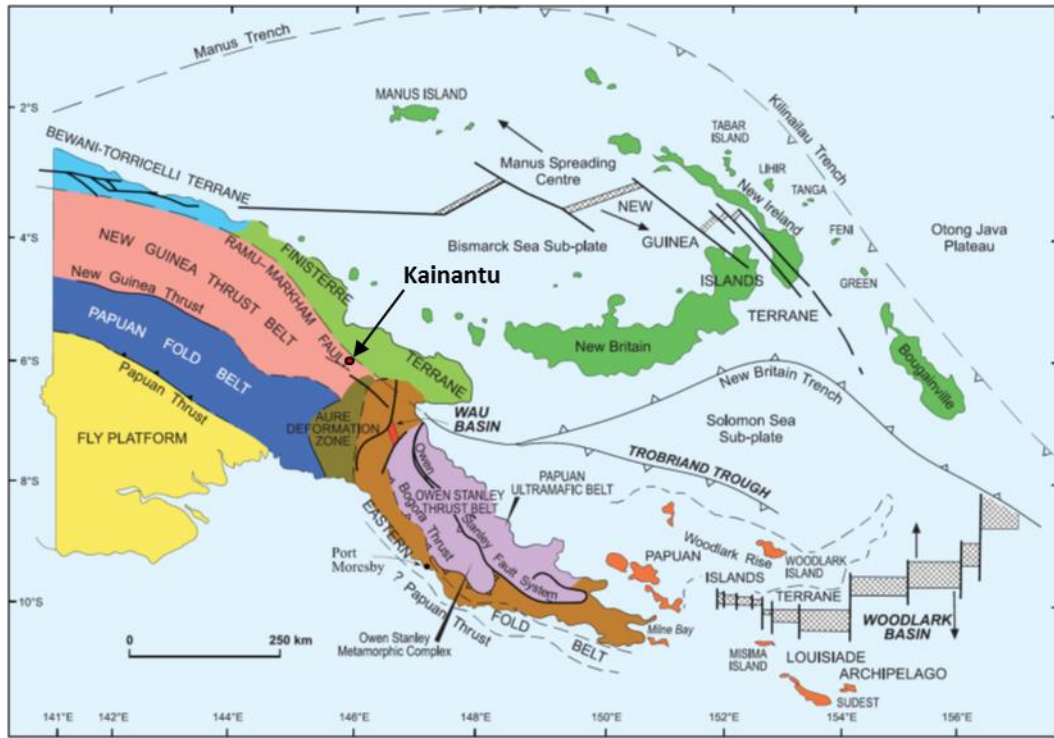


Figure 9. Tectonic Framework of Papua New Guinea, Showing Location of Kainantu Property.

Source: Williamson & Hancock (2005).

7.2 PROPERTY GEOLOGY

The Kainantu area is underlain by rocks of the Early Miocene Bena Bena Formation, comprising pelite, psammite, conglomerate and marl beds metamorphosed to greenschist to amphibolite grade. These are unconformably overlain by Miocene age Omaura Formation consisting of volcano-lithic sandstones and siltstones and numerous fossiliferous limestone lenses. The overlying Yaveufa Formation consists of basaltic and andesitic flows, agglomerates, volcanoclastic sandstone and limestone (Tingey and Grainger, 1976). The mid-Miocene Akuna Intrusive Complex consists of multiple phases ranging from olivine gabbros, dolerites, hornblende gabbros and biotite diorites to granodiorites. Late Miocene age Elandora Porphyry dykes form small high level crowded feldspar porphyry dykes and diatreme breccias associated with mineralization (Table 9). A north-northeast trending transfer structure transects the area, with associated mineralization, alteration and porphyry complexes aligned along it. Local deformation history as documented in the Irumafimpa-Kora mine area is shown in Table 10.

Table 9. Summary of main regional rock units identified within Kainantu area.

Age	Rock Units
Recent Quaternary	Kainantu Formation – basal fluvial conglomerate, sandstone and mudstone overlain by well bedded tephra.
~~~ Unconformity ~~~	
Late Miocene	Elandora Porphyry – intermediate dykes sills and stocks.
Early Miocene	Akuna Intrusive Complex – range in composition from olivine gabbros through to granodiorites.
Early Miocene – Mid Miocene	Yaveufa Formation - basaltic and andesitic agglomerates, lithic tuffs, volcanoclastic sandstone and limestone.
Late Oligocene – Late Miocene	Omaura Formation – thin bedded to laminated calcareous siltstone and mudstone.
~~~ Unconformity ~~~	
Early Mesozoic	Bena Bena Formation - pelite, psammite, conglomerate and marl metamorphosed to schist and phyllite.

Table 10. Local deformation history for the Kainantu area.

Source (Blenkinsop, 2005)

Deformation history		
Event	Structures	Interpretations
D4	Chinook	Joint: open due to in situ stress orientation
D3	Faults with gouge	N-S shortening: faults along S1
	Mill lode style mineralization	Extension on Mill Lode: Reactivation of S1
D2	Crenulations: L_1^2 lineation, S2	NNE shortening
D1b	Shear zone network	Localisation into zones of intense deformation
D1aq	Main cleavage - S1 L1 lineation = L_1^0	N-NE shortening

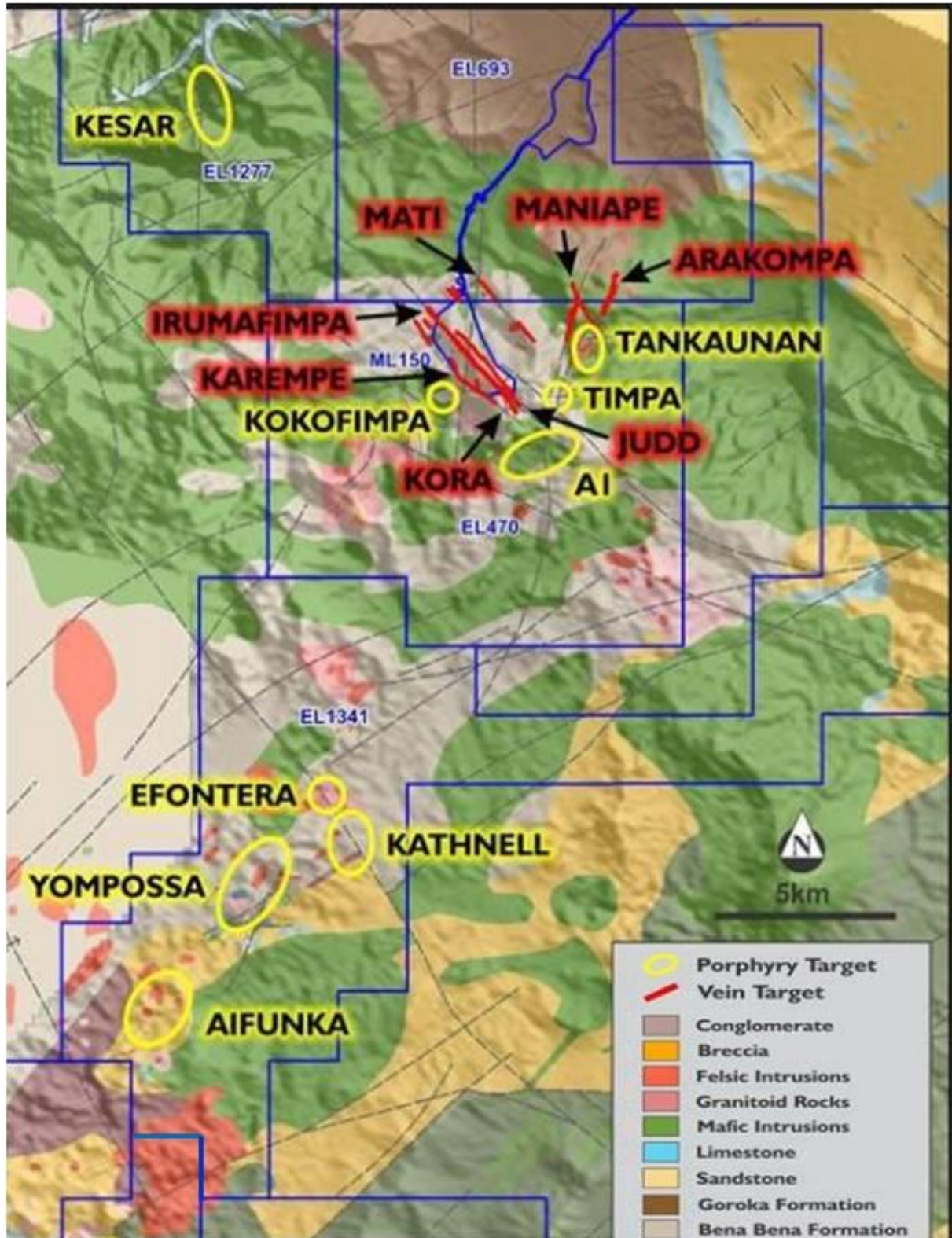


Figure 10. Kainantu property geology and known vein and porphyry deposits and prospects.

The prospects are summarised in Table 11.(Source: Barrick, 2014)

7.3 MINERALIZATION OVERVIEW

The descriptions in this section have been sourced from the summary provided in Barrick (2014).

Mineralization on the property includes gold, silver and copper occurring in epithermal Au telluride veins and Au Cu Ag sulphide veins of Intrusion Related Gold Copper (“IRGC”) affinity and also less explored porphyry Cu Au systems; and alluvial gold.

The Irumafimpa-Kora vein deposit is the most advanced project at Kainantu with current defined resources and past modern mining activity in the Irumafimpa area. The deposit occurs in the centre of a large mineralized system approximately 5km x 5km in area that has been partly delineated by drilling and comprises several individual zones of IRGC and porphyry style mineralization. Peripherally exploration activities have identified further areas of vein and porphyry-style mineralization.

Other less advanced prospects on the property include epithermal Au veins similar to Irumafimpa, IRGC veins similar to Kora, porphyry Cu Au systems, skarn Cu, Pb and Zn mineralization and alluvial gold. A summary of the mineralization style, host rocks and dimensions and continuity for the Irumafimpa-Kora vein deposit and the other vein and porphyry prospects in the Kainantu Project is shown in Table 11 and described further below.

The location of the deposits and prospects in relation to the property boundaries is shown in Figure 10.

Table 11. Summary of mineralization, host rocks, dimensions and continuity for main Kainantu deposits and prospects

Deposit / Prospect	Mineralization	Host Rocks	Dimensions	Continuity
Irumafimpa-Kora (including Eutompi)	Vein Low sulphidation Au-Cu (described in Section 7.4) (Resources reported in Section 14)	Quartz veins in chlorite-sericite schist.	>2.5 km strike x 60 m wide System is open along strike and at depth	Drilling shows strike and depth continuity at a gross scale. Gold mineralization is discontinuous.
Judd	Vein Low sulphidation Au-Cu (Barrick drilling returned 3m @ 278g/t Au)	Quartz veins in chlorite-sericite schist.	2.5km strike x 1-4m wide Vein system as defined by surface mapping and sampling and sporadic drilling. Mineralization open along strike and to depth	Surface continuity along strike unknown due to poor outcrop exposure
Karempa	Vein Epithermal Au (rock chip average grades of 6.7 g/t Au, 16.8 g/t Au, 45.2 g/t Au and 50.8 g/t Au;)	Quartz veins in granodiorite and chlorite-sericite schist.	3km strike and 1-2m wide vein as defined by surface mapping and sampling. Mineralization open along strike and to depth	Discontinuous vein outcrops and no drilling
Arakompa	Vein Epithermal Au	Quartz veins in Akuna Diorite	3km strike and 1-2m wide vein system NNE trending No deep drilling.	Surface continuity along strike unknown due to poor outcrop exposure
Maniapa	Vein Epithermal Au	Bena Bena Metamorphics, Akuna Diorite,	Strike length 1km Near surface zone of mineralization of 700m strike x 34m wide x 125m depth defined by surface sampling and diamond drilling	Continuity of near surface mineralization confirmed by drilling
Mati / Mesoan	Vein Epithermal Au (Rock chips average of 28g/t Au and a maximum of 131g/t Au)	Bena Bena Metamorphics, Akuna Diorite,	1 km strike mineralized zone defined No drilling	Surface continuity along strike unknown due to poor outcrop exposure No drilling

Deposit / Prospect	Mineralization	Host Rocks	Dimensions	Continuity
Kesar (reconnaissance stage)	Vein and Porphyry Au and Cu Vein rock chip grades up to 30g/t Au. Porphyry copper grades up to 0.5% Cu. Quartz-sulphide veins with pyrite ± chalcopyrite ± galena ± sphalerite ± molybdenite ± covellite also identified	Quartz veins. Dacitic porphyry dykes with potassic alteration contain Cu mineralization.	Undefined	Undefined
A1 (reconnaissance stage)	High-sulphidation and porphyry Cu-Au Brecciated vuggy silica-pyrite-enargite mineralization and anomalous molybdenum in soils Historic float sample of massive enargite-pyrite returned 16.6% Cu and 12g/t Au.	Bena Bena Metamorphics, Akuna Diorite, Feldspar porphyry and breccias	Undefined	Undefined
Kokofimpa	Porphyry Cu-Au	Akuna Intrusive Complex and Elandora porphyry intrusions within the Bena Bena Metamorphics	3 km x 3 km Defined porphyry system with multiple magmatic phases with minimal drilling in center of prospect.	Undefined
Tankaunan	Porphyry Cu-Au	Akuna Intrusive Complex and mid-late Miocene Elandora Porphyry intrusions within Bena Bena Metamorphics	Extent of systems needs to be defined by first pass 400x400m drilling.	Undefined
Timpa	Porphyry potential postulated Cu-Au-As in Soils Advanced argillic alteration Quartz Breccia (monomict, quartz cemented, with shallow quartz infill textures; soil sampling shows the breccia is anomalous in Au, As, Bi, Sb, W)	Bena Bena Metamorphics and breccia	Quartz breccia is 500 m by 100 m. Other mineralization Undefined	Undefined
Aifunka	Skarn (Porphyry-related) Cu and Au Au (Barda reefs)	Mineralization is hosted in calc-silicate bands spatially associated with the brecciated porphyry dyke contacts. Underlain by the Omaura Sediments and Akuna Intrusive Complex with Elandora Porphyry.	Undefined	Undefined
Yompossa	Porphyry Cu-Au (60m @ 0.3% Cu and 0.1g/t Au from 105m in BHP01)	Underlain by Bena Bena Formation and Omaura Formation. Contains feldspar porphyry intrusions interpreted to be associated with Elandora Porphyry	Anomaly is 500m x 600m and is open to the NE. Potential for mineralization below historic drilling.	Undefined
Kathnel	Base metal epithermal veins (Pb-Zn-Cu-Au)	-	Undefined	Undefined
Efontera	Porphyry Cu-Au	-	Undefined	Undefined

7.4 IRUMAFIMPA-KORA VEIN SYSTEM

The Irumafimpa-Kora vein system (comprising the Kora, Eutompi and Irumafimpa prospects) is interpreted to contain two stages of mineralization (Corbett, 2009). The earliest is a sulphide-rich Cu-dominant stage. This is overprinted by a quartz-rich Au-dominant crustiform quartz vein to breccia system with high grade gold associated with tellurides (e.g. Calaverite AuTe). The alteration and mineralization paragenesis recognised in the Irumafimpa-Kora vein system is summarised below in Table 12.

Table 12. Mineralization and alteration paragenesis in the Irumafimpa-Kora vein system.

Stage	Name	Description
Stage 1.	Silicification and fuchsite alteration	silica, fuchsite
Stage 2.	Sulphide-rich Cu-dominant	quartz, pyrite, chalcopyrite, bornite
Stage 3.	Quartz-rich Au-dominant	quartz, gold tellurides (calaverite and kostivite), native gold
Stage 4.	Quartz Cu	quartz, pyrite, chalcopyrite, bornite

Stage 1 is the earliest period of alteration and is characterised by silicification and fuchsite alteration of phyllitic wall rock.

Stage 2 mineralization comprises coarse-grained idiomorphic quartz and pyrite (typically euhedral) veins with base metals. Volumetrically this early mineralization appears to be the most abundant mineralization. At Kora the mineralization comprises massive pyrite veins to pyritic breccias, grading to pyrite-chalcopyrite-bornite veins characterised by elevated Zn, Pb, Sn, W, Bi, and Sb. High copper grades (average 2.2 % Cu) occur at Kora. There appears to be a lateral zonation northward to lower copper grades at Irumafimpa.

Stage 3 mineralization is the dominant gold-bearing stage and is characterised by crustiform, vughy and colloform quartz veins, quartz breccias, and xenomorphic pyrite. Most of the gold occurs as the gold tellurides calaverite and kostivite, which are concentrated at vein margins. Significant native gold has been locally observed and is probably a result of oxidation of tellurides at Irumafimpa, and as primary native gold at Kora.

Stage 4 is manifested as local brecciation and deposition of low temperature quartz along with minor copper mineralization.

At Irumafimpa, the abundant essentially barren mineralization (quartz and sulphide) is highly visible and voluminous whereas gold mineralization is more cryptic and occupies a minor volume within the earlier mineralization stages (Figure 11).

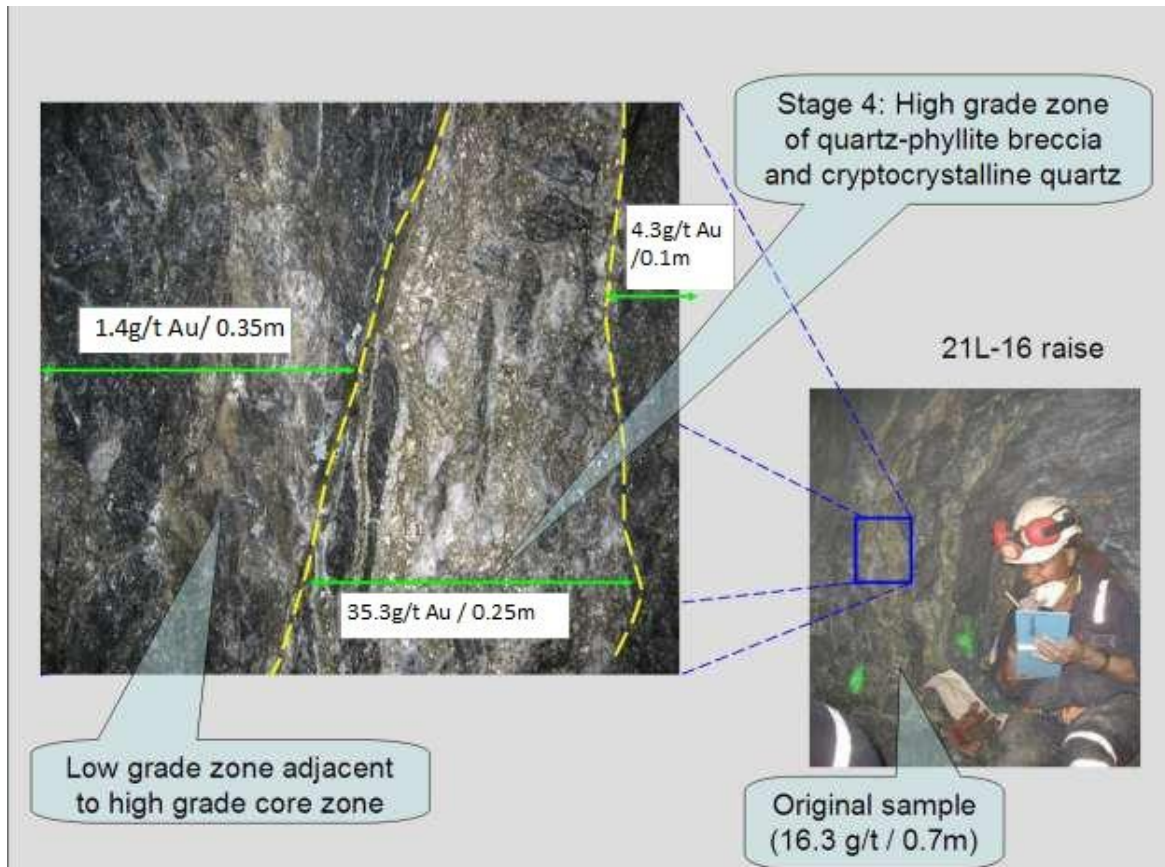


Figure 11. Diagram illustrating grade distribution within an original 0.7m sample.

(Source: Smith and Thomas, 2008)

7.4.1 Host rocks

Dominant host rock is highly sheared and deformed Bena Bena Formation low grade metamorphics intruded by Elandora porphyry at the Northern end of the Vein system.

7.4.2 Controls

The structural history of the Irumafimpa-Kora area has been documented by Blenkinsop (2005). The Irumafimpa-Kora vein system follows the main NW shear zones of the contiguous Irumafimpa and Kora structures. Veins are breccia veins with abundant clasts of both altered wall rock and earlier stages of vein mineralization. Vein formation was multistage, with at least four identifiable episodes of alteration and mineralization (Table 12).

At Kora both the sulphide-rich Cu-dominant and quartz-rich Au-dominant mineralization occur along the same NW trending sub-vertical structure. This is likely a long lived structure, which was reactivated at several different stages. The quartz-rich Au-dominant mineralization shows variations in dip (from sub-vertical to locally -60° dip) and strike, which define larger high grade shoots.

Late stage faults with gouge postdate the mineralization (Table 10). These usually occur on the vein margins but can cause local disruption of the veins.

7.4.3 Dimensions and Continuity

The current resources occupy a broad northwest trending mineralized zone more than 2.5 km long and up to 60m wide in which individual veins vary from less than one metre wide that pinch and swell over short distances (Au telluride lodes) to more continuous veins up to several metres wide (Au Cu Ag sulphide lodes).

Historical exploration has identified and subdivided several shoots within the lodes, defining the Kora, Eutompi and Irumafimpa Prospects. The vertical extent in outcrop is also significant, with Kora identified for at least 200m vertical extent (1750-1950m RL) and Irumafimpa outcropping at 1300m RL.

At Kora, drilling has confirmed that the overall system has a vertical extent greater than 800m. Mineralization is open in all directions. Wider mineralized zones (up to 6m) contain multiple high grade veins which may be splays. The Kora veins average 3.1m true width; which is the entire extent of the known veins before cut-off grades are applied. The Kora veins range from 1.6m (Kora No. 3 vein) up to 4.2m true width (Kora No. 1 vein). The Mill veins at Irumafimpa average 1.2m true width, which is the minimum width used during resource estimation.

Eutompi is the area of mineralized lode between Kora and Irumafimpa, extending from around 58,900mN to 59,400mN. Limited drilling has been conducted in this region and only at high levels. Drill density is insufficient to generate a constrained resource. The drilling indicates this area may be more structurally complex than at other locations, but has confirmed that the intermediate and low sulphidation styles of mineralization continue throughout. Results include 25m @ 2.0 g/t Au, 4.2% Cu, 88 g/t Ag (including 1m @ 22.6 g/t Au, 17% Cu, 1000 g/t Ag) in hole 107BD06 and 2.3m @ 13.39 g/t Au (108BD06).

7.5 OTHER VEIN SYSTEMS

7.5.1 Judd

A narrow intermediate and low sulphidation vein system located 200m east of and parallel to Kora which was partially tested by Barrick holes drilled to test the Kora lode at depth. This sporadic drill testing on the Judd lode returned a maximum intersection of 3m @ 278g/t Au. Surface mapping and sampling has indicated a mineralized strike length of over 2.5 km. Judd is located 200m east of Kora on ML150. Holes designed to specifically target the Judd lode have the potential to yield resources within close proximity to the immediate mine environment and have been allocated a high priority by K92ML.

7.5.2 Karempa

Karempa is a high grade vein system of over 3km strike extent (Figure 12, Figure 13) immediately west of Irumafimpa-Kora with only one drillhole testing the system to date. Epithermal boiling textures, strike continuity, an associated VTEM anomaly and high grade surface results (e.g. 156g/t returned from colloform banded epithermal quartz veins) define this target. Rock chip characterization sampling at four locations along the length of the vein system indicate a 1m to 2m width, and returned average grades of 6.7 g/t Au, 16.8 g/t Au, 45.2 g/t Au and 50.8 g/t Au.

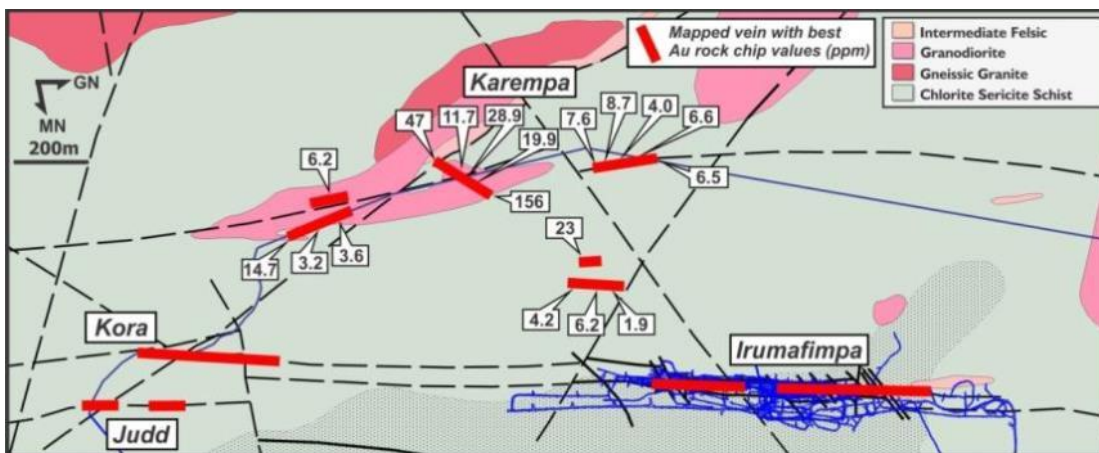


Figure 12. Karempa location plan showing mapped veins and rock chip results.

(Source Barrick 2014)

Prospect location in relation to property boundaries is shown in Figure 10

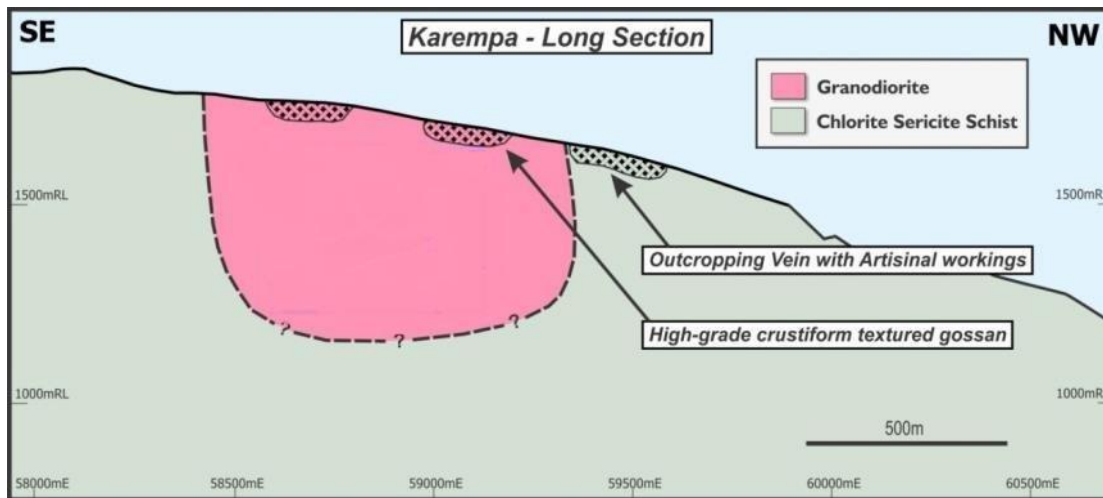


Figure 13. Karempa long section showing strike extent of known surface footprint.

(Source Barrick 2014)

Prospect location in relation to property boundaries is shown in Figure 10

Vein systems other than the Judd and Karempa veins are described in more detail in Section 7.5 of the “Independent Technical Report, Resource Estimate and Summary of Mine Facilities, Kainantu Project, Papua New Guinea” by Nolidan Mineral Consultants, Author Anthony Woodward, April 15, 2016 which is filed on SEDAR.

7.6 PORPHYRY SYSTEMS

Prospects containing porphyry mineralization and high-sulphidation mineralization at Kainantu occur within an eight kilometre zone surrounding the Irumafimpa-Kora vein system and stretching to the east, south and west of the veins (Figure 10). Many of the porphyry targets that have been delineated in the Kainantu project area are early stage (reconnaissance) and have not been drill tested.

These prospects have not shown economic mineralization to-date and are not considered high priority targets as the current focus of exploration will remain on vein mineralization. They are summarised in Table 11 and described further in Section 7.6 of the “Independent Technical Report, Resource Estimate and Summary of Mine Facilities, Kainantu Project, Papua New Guinea” by Nolidan Mineral Consultants, Author Anthony Woodward, April 15, 2016 which is filed on SEDAR.

8 DEPOSIT TYPES

Gold-copper deposits within the SW Pacific Magmatic Arcs have been classified into three main groups by Corbett and Leach (e.g. Corbett and Leach, 1997):

- Porphyry-related (including gold skarn).
- High sulphidation gold-copper.
- Low sulphidation (including sediment-hosted replacement).

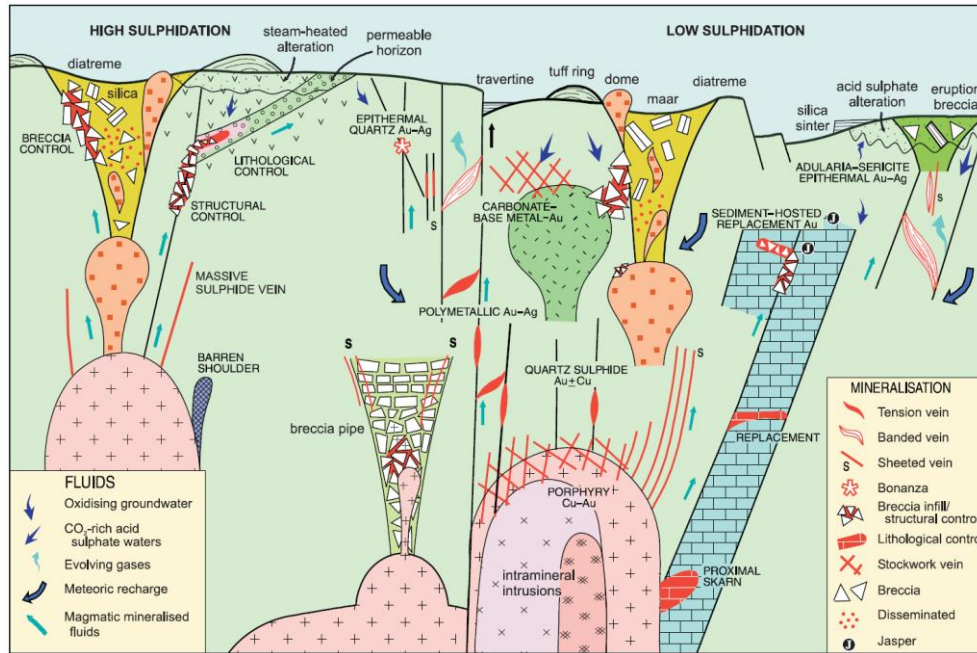


Figure 14. Conceptual model for porphyry and related low and high sulphidation mineralization.

Source: Corbett (1997)

Telescoping may overprint the varying styles of low sulphidation gold mineralization upon each other or upon the source porphyry intrusion.

Hydrothermal porphyry-related activity in the Kainantu area may have been protracted and associated with more than one intrusive phase (17 Ma to younger than 7 Ma). According to Corbett (2009), while the accepted wisdom is that porphyry Cu-Au mineralization in the Kainantu region is related to Elandora style porphyry intrusions, the coincidence of prograde alteration (Kokofimpa area; K-feldspar alteration) with Akuna-style diorite intrusions suggests alteration and mineralization may have been initiated at an earlier Akuna age and continued to an association with Elandora intrusions. The presence of Elandora clasts within advanced argillic altered breccias, is consistent with a protracted history of activity. The (17-13 Ma) extended age of Akuna intrusions provides for batholithic intrusions to be overprinted by the mineralized phase recognised herein and distinguished from the younger (9-7 Ma) Elandora-style intrusions. Corbett (2009) recommends limited age dating is conducted once field relationships are established.

These exploration models as used by HKL and Barrick emphasized the epithermal and porphyry geological setting, which is broadly correct, at least spatially. But these models were later refined by Espi and others (2006) who recognized that the high grade quartz-Au-telluride veins with common percent Cu grades and significant W and Bi (e.g. Irumafimpa and Kora) were likely a significant separate event not directly connected to a porphyry Cu-Au source. The term “intrusion-related lodes” was introduced to describe this mineralisation style. The consistent Au-Te association is interpreted to indicate an alkalic intrusion source at depth. Felsic dykes observed adjacent to some of the mineralised veins could be derived from such a source and may serve as a useful exploration guide.

9 EXPLORATION

K92ML has not commenced surface exploration on ML150 at the Irumafimpa gold mine.

Historic exploration on ML150 (Irumafimpa, Kora, Judd, and Karempe) is reported in Section 6.1 of this report. Further exploration information at other prospects at Kainantu is described in the “Independent Technical Report, Resource Estimate and Summary of Mine Facilities, Kainantu Project,

Papua New Guinea” by Nolidan Mineral Consultants, Author Anthony Woodward, April 15, 2016 which is filed on SEDAR.

10 DRILLING

In September 2016 two diamond drill rigs commenced work underground at the Irumafimpa gold mine. One rig was focused on drilling out the Irumafimpa deposit for grade control and mine planning purposes on a 15m by 15m pattern from 1235mRL and 1247mRL and another drill rig was targeting the Judd vein system from 950mRL.

No results have been made available to Nolidan at the time of compiling this report.

Historic drilling data on ML150 (Irumafimpa, Kora, Judd, and Karempe) is reported in Section 6.1 of this report. Historic drilling data on the Arakompa and Maniape prospects is reported in Section 6 of the “Independent Technical Report, Resource Estimate and Summary of Mine Facilities, Kainantu Project, Papua New Guinea” by Nolidan Mineral Consultants, Author Anthony Woodward, April 15, 2016 which is filed on SEDAR.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE PREPARATION

11.1.1 Drill core (HPL)

Procedures for all HPL exploration sampling were not sighted by Nolidan. According to Logan (2006), the following procedures were followed by HPL for the sampling of drill core at least from 2004 onwards:

- A line was drawn down the drill core.
- Competent drill core was halved using a diamond saw.
- Less competent core was wrapped in packaging tape prior to cutting with a diamond saw.
- Pieces of broken core were halved whenever possible, if not possible random but representative pieces were sent for assay.
- Clay zones were halved using a knife when cutting by saw was not possible.
- Intervals of poor core recovery were sampled from core block to core block, because it is usually impossible to determine exactly where the core loss was.

11.1.2 Mine Grade Control (HPL)

Written procedures for HPL grade control sampling were not sighted by Nolidan. The following comments were taken from comments in internal Barrick documents (Gauthier and Pridmore, 2007; Smith and Thomas, 2008).

- Grade control sampling was a standard channel sample with all crosscuts, active development and stope faces sampled.
- Every 3m cut on the development drives were sampled and mapped.
- Faces are not generally washed down prior to mapping and sampling. Mud and dust on mining faces increase the risk of contaminating samples and make accurate mapping difficult.
- Sample lines are frequently marked up by the sampler not the geologist. This means that samples are not readily related to geology.
- The location and extent of the gold bearing veins within the mineralized structures is not well understood by the majority of the geologists. As a consequence of this, the measurement of the gold-bearing vein widths is inaccurate.
- The samplers chip into their open hand, as opposed to directly into a sample bag. This is a serious contamination issue. The mine is humid and in places wet so that some of the sample material usually sticks on the sampler’s gloves after each sample.

11.1.3 Drill core (Barrick)

All drill core was logged, photographed (wet and dry), then cut and sampled at Barrick's Kumian core yard. Logging data entry was completed using an in-house developed version of the Acquire software. After logging, core was half-cut using diamond saws, and continuously sampled into numbered calico sample bags. The samples were submitted to the sample preparation facility of Intertek Laboratory Services in Lae (PNG). Sample preparation involved drying the samples at 105°C, crushing in a jaw crusher with 95% of the sample passing <2mm, riffle splitting and pulverising to 95% passing <75µm.

11.1.4 Drill core (K92ML)

Procedures for K92ML drillcore sampling have been sighted by Nolidan and are similar to those used previously by HPL and Barrick.

11.2 SAMPLE SECURITY

No written sample security procedures were sighted by Nolidan

11.3 SAMPLE ANALYSES

The following descriptions of analytical techniques used by HPL are taken directly from SRK (2006):

Drillhole and channel sample data used in the resource estimate has been analysed using a combination of fire assay and aqua regia techniques at a number of separate laboratories over the course of the project. Gold in tellurides can prove problematic to analyse using fire assay techniques as the tellurium content can lead to losses of precious metal during cupellation which subsequently results in a low bias in the results. In order to address this issue the sample is therefore oxidised either through the use of an oxidising flux, roasting the sample or a combination of both in order to oxidise the tellurium. These techniques are reported to have been used for all samples at Kainantu.

Between 1992 and 2002 the exploration data was analysed at the laboratory of Astrolabe Propriety Limited in Madang, Papua New Guinea. Gold was determined by Fire Assay with AA finish. The majority of the assays were undertaken using a 50g charge although some were assayed using two separate 25g charges the values of which were then combined.

Between 2002 and 2005 the exploration data was analysed by the Australian Laboratory Services (ALS) in Townsville, Australia. Gold was determined by Fire Assay with AA finish using a 50 g charge.

Since January 2006 (up to closure in 2008) all samples collected on the mine have been analysed by the mine laboratory at Kumian, Papua New Guinea. Gold is determined using aqua regia with AA finish 50 g charge as opposed to the Fire Assay approach utilised at the exploration stage. While aqua regia is an accepted technique for gold assaying care must be taken as the matrix of the sample can adversely affect digestion leading to understated concentrations. In particular, care should be taken with, for example, high silica (quartz) content. In these circumstances aqua regia techniques may understate the gold content relative to a Fire Assay.

During Barrick exploration at Kainantu analytical pulps were shipped to Intertek Laboratory Services in Jakarta (Indonesia) for analysis. Au was analysed by 50g fire assay (FA50) with AAS finish (gravimetric finish for samples with Au > 5 ppm). Multi-element analysis was done by multi acid digestion (HCl/HNO₃/HClO₄/HF) ICP (IC50) for 33 elements including Ag and Cu. Samples with > 0.5% Cu were re-analysed with AAS finish (GA50). Later samples were also assayed for Mn and S.

11.3.1 Laboratory Independence and Certification

The analytical laboratories of ALS in Townsville, Astrolabe in Madang and Intertek in Lae and Jakarta are all accredited by the National Association of Testing Authorities (NATA). Nolidan has not sighted any certification regarding the onsite Kumian Laboratory.

11.4 QUALITY ASSURANCE AND QUALITY CONTROL

Quality Assurance (“QA”) concerns the establishment of measurement systems and procedures to provide adequate confidence that quality is adhered to. Quality Control (“QC”) is one aspect of QA and refers to the use of control checks of the measurements to ensure the systems are working as planned.

The QC terms commonly used to discuss geochemical data are:

- Precision: how close the assay result is to that of a repeat or duplicate of the same sample, i.e. the reproducibility of assay results.
- Accuracy: how close the assay result is to the expected result (of a certified standard).
- Bias: the amount by which the analysis varies from the correct result.

Original reports regarding QAQC procedures and results during HPL and Barrick sampling programmes were not available to Nolidan for the preparation of this report.

Barrick is an established publicly traded Canadian mining company with multiple international mining, development and exploration operations. As such, it is reasonable to assume that for the Kainantu project Barrick used industry standard QAQC procedures as per the QAQC procedures they employ at their other projects (see section 11.4.1.5).

However, summaries of QAQC procedures and results occur in several different reports on the HPL drill samples and are compiled below.

11.5 QC PROGRAMS

QA/QC procedures usually involve the following types of QC samples being taken or inserted into the sampling stream by the personnel collecting the samples.

- Certified Reference Materials (“CRM”, or “standards”): low, medium and high grade added at a planned rate of about one every 20 samples or 5%. CRM assess accuracy.
- Field Duplicate Samples: one in every 20 samples is split and submitted as a field duplicate. Both samples are inserted into the sampling stream and prepared and assayed like any other sample. Field duplicates are used to monitor sample batches for poor sample management (bias), contamination and tampering and laboratory precision. Field duplicates also provide some measure of sample homogeneity.
- Field Blank: Samples of a “blank”, known to contain low level of economically interesting metals are inserted into the sample stream. Field blanks are usually inserted at a planned rate of one every 20 samples. Blanks assess contamination.
- Referee Laboratory duplicates (“check assays”): Sample pulps are sent for duplicate assay to another laboratory. Results are then plotted against the original laboratory results to check for anomalous results, contamination or equipment failure or calibration trends (bias).

Analysing laboratories also carry out their own internal QA/QC procedures involving the insertion of CRM, blanks and assay repeats.

QC programs are subdivided by company and time period. Descriptions of QAQC programmes up to 2006 are taken from SRK (2006).

11.5.1 1992-2002 Exploration

Between 1992 and 2002, exploration data was analysed by Astrolabe. QA/QC procedures include the routine repeat analysis of 15% of the data together with the re-assaying at an external laboratory of all samples returning greater than 5 g/t Au. No standards were utilised. It is reported that no significant problems were detected. Figure 23 presents a scatter plot (sourced from the HPL DFS report)

comparing the results of the Astrolabe internal repeat assays. Although the scale of the axes results in poor resolution at low values the overall result indicates a good level of precision.

11.5.2 2001-2005 Exploration

Between 2002 and 2005, HPL exploration samples were analysed by ALS in Townsville. QA/QC procedures included the use of standards (every 10 to 20 samples), repeats and check assays at other laboratories including all samples greater than 5 g/t Au. It was reported that no significant problems were detected.

Figures 24 to 25 present scatter plots (sourced from the DFS report) of external check analyses (Genalysis) versus ALS and for ALS internal repeats analyses respectively. Although the scale of the axes results in poor resolution at low values the overall result indicates a good level of precision and no discernible bias. Figure 26 presents an example CRM control chart (again sourced from the DFS report) which indicates (for this particular CRM) that deviations from the CRM value were typically less than 5%. This is considered an acceptable level of accuracy.

11.5.3 2006-2008 Mine Sampling

From 2006 to mine closure in 2008, mine samples were analysed at the on-site laboratory ("Kumian"). From the information supplied to Nolidan, it is not clear if this refers to only grade control samples, or all samples (including underground exploration drilling). It appears that no field QC samples were inserted in grade control assay batches, and the only QAQC undertaken was by the laboratory itself. A Barrick review of the mine operations in 2008 referenced the inclusion by mine geologists of 'blind' CRM into assay batches.

QA/QC procedures at the Kumian Laboratory included the use of a blank, a standard, two repeats and two barren flushes for every 20 samples analysed. According to SRK (2006), check analyses for each batch check were undertaken at ALS (using aqua regia) and Intertek laboratory (using Fire Assay). However, according to a Barrick internal review in 2007, there were no check assays undertaken on grade control data. Barrick's review also indicated that check assay results were not being routinely recorded, and that written QA/QC procedures were not finalised.

Check analyses showed a low bias to Kumian results compared with ALS and Intertek. A low bias was also present in CRM control charts (both laboratory and mine CRM) for Kumian, in the order of 5-10%. Reasons for the low bias were apparently not fully examined, although one cause suggested by Barrick was incomplete digest using aqua regia. Aqua regia techniques often understate gold content relative to fire assay and Nolidan suggests that the installation of fire assay facilities at the Kumian laboratory should be investigated.

Repeat analyses showed a good level of precision.

11.5.4 2004-2006 Exploration

Exploration drilling at Eutompi and Kora from 2004-2006 was managed by Ross Logan and Associates. QA/QC included insertion of two gold CRMs and limestone blanks, but no mention is made of field duplicates. Insertion rates for QC samples are not specified. According to the report on drilling, these procedures were standard for HPL at the time.

Samples were analysed by ALS (2004 Kora drilling) and by Intertek Laboratories in Lae (other drilling). Results for CRMs plotted within acceptable limits for both laboratories, although some drift over time was noted for Intertek. Field blanks did not show any issues with contamination.

11.5.5 After 2008 Barrick Exploration

QA/QC procedures have not been sighted by Nolidan for Barrick exploration drilling since they acquired the property in 2008. However Barrick reports that routine quality control is conducted at various stages throughout the sample preparation and analytical stages of drillcore sampling including reference standards, replicate and duplicate sampling and blanks as detailed in the assay flowchart. QA/QC

checks are done whilst importing each assay file and on a monthly basis. The levels of variability and accuracy at which actions are initiated are site specific but as a guide:

- Batches that have two standards in a row outside the two standard deviation limit are actioned; Any standards outside three standard deviations are actioned; and
- Batches with blanks greater than 10X expected value are actioned

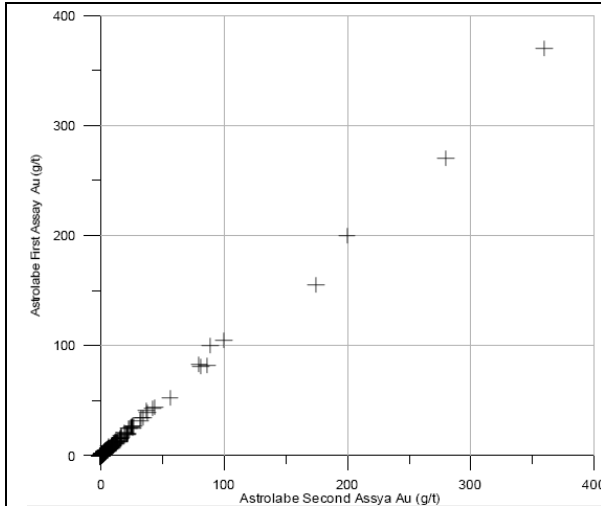


Figure 15. Scatter of of Repeat Data, Astrolabe Laboratory 1992-2002

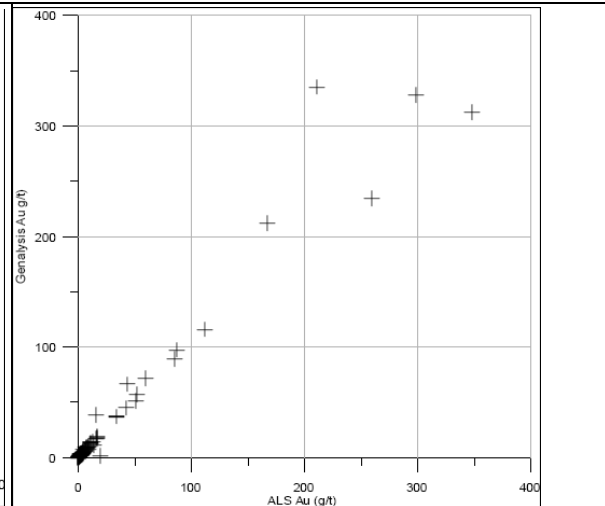


Figure 16. Scatterplot of ALS vs Genalysis Results, 2002-2005.

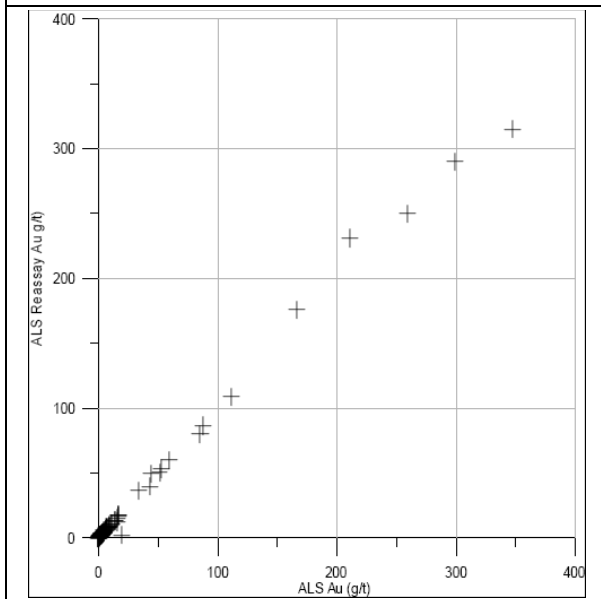


Figure 17. Scatterplot of ALS Repeat Assays, 2002-2005.

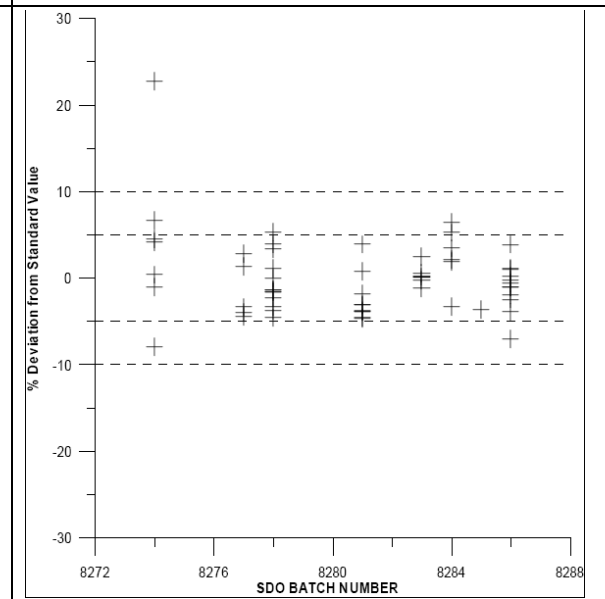


Figure 18. Example CRM Control Chart (2002-2005)

11.6 ADEQUACY OPINION

No independent review of the drillhole sampling was done by Nolidan. Although it appears that this work was done to an industry acceptable standard, there is always a risk involved with geological interpretations and grade continuity. Geological logs were compared to selected drill core laid out specifically for the task of validating the geological logs.

Generally, the results of the QA/QC program implemented are considered satisfactory for an advanced stage property. It is Nolidan's opinion that the sample preparation, security and analytical procedures were adequate and follow accepted industry standards.

The classification of the current resource was restricted to Indicated and Inferred due to the drill spacing at Kora and limited confidence in underground sampling information from Irumafimpa.

It was concluded that Kainantu's database is reliable and falls within the norms of reasonable variation and is suitable for disclosing resources.

12 DATA VERIFICATION

12.1 DATA VERIFICATION PROCEDURES

This report was prepared on the basis of information compiled by Highlands and Barrick as supplied to Nolidan by Barrick and a two day visit to the Kainantu gold mine including a review of the Kainantu drill core and drill sections at the Exploration office. Discussions were held with Barrick's Exploration Manager and Mine Manager while on site.

12.1.1 Drillhole Database

All exploration data sourced by Barrick, including historic and Barrick data, is entered into an acQuire database located in Perth. This includes surface sample location and assay data, surveyed collar and downhole survey data, geological logs and assay data. Validation of the data entry is at the cell level and is controlled by predetermined validation tables. A number of checks are incorporated using SQL scripts to ensure the integrity of the data.

The drillhole database integrity was reviewed for internal inconsistencies, duplicate sample numbers and assay reference numbers. No significant errors were detected.

12 holes had duplication of survey results, results were the same except for the database field SURVTYPE duplicated records were logged as both CAMERA and FEFLEX. Nolidan removed the camera records from the database.

12.1.2 Face Samples

Comparison of grade control face sampling and drilling in the same mineralized zones shows a significant bias towards lower average grades in drilling compared with the average grade of the face samples. For all veins the highest recorded values for gold (outliers) occurred in drillhole samples and grade capping was therefore used. Face samples are however concentrated in the higher grade mining areas, so were included in resource estimation.

Recoveries in diamond drilling were recorded as being typically less than 80% in mineralized zones (SRK, 2006), which may explain the assay bias in terms of gold loss in non-recovered material. However, there were also a number of problems noted with underground channel sampling by Smith and Thomas (2008), including potential bias introduced by over-sampling of softer material.

12.1.3 Site Visits

Mr Anthony Woodward visited Kainantu Gold Mine from 12th November to 13th November 2014. The project was on care and maintenance. In the course of the site visit, Mr Woodward viewed mineralized vein systems in drill core, and examined the drill core processing and storage facilities (Figure 19). He also viewed photographs of mineralization in underground development headings and in drill core.



Figure 19: Core Logging facilities



Figure 20: Almonte Automatic core saw and brick saw

Barrick ceased mining and processing in January 2009. Site buildings and camp facilities were in a good functioning order and appear constantly maintained. Underground mobile equipment has been parked and exposed to the elements since mining ceased. Discussions were held with Barrick's Exploration Manager and Mine Manager while on site.

Mr. Woodward again visited the Kainantu site from 22nd November to 25th November, 2016. The three day site visit included a visit to the rehabilitated underground workings, current underground diamond drilling sites at 1247mRL and 950mRL, inspection of the treatment plant, and discussions with company site management. Development activities were occurring on 1205, 1220 and 1235 Levels.

Chris Desoe (AMDAD) visited Kainantu site from 7th June to 14th June 2016. The project was in the initial stages of restarting, focussing on rehabilitation of the underground access and establishment of power and ventilation. Mr Desoe examined the surface facilities and various areas of the existing underground workings.

12.1.4 Independent Samples

No independent samples were collected. A review of drill core and mineralized intercepts was undertaken in the core yard (Figure 19). Examples of lodes and styles of mineralization in core were inspected. Drill logs were compared with drill core. Figure 21 shows localized shear brecciation with pyrite- chalcopyrite mineralization and minor carbonate and quartz. Red haematite stains can be seen around sub-angular quartz clasts. Figure 22 shows dominant fine grained foliated phyllite with crustiform quartz-pyrite and trace chalcopyrite veins within intervals of semi-massive pyrite and chalcopyrite in fine grained quartz.



Figure 21: BKDD0027 461.6 to 462.6 m 16.3 g/t Au, 131 g.t Ag and 6.48 g/t Cu



Figure 22: BKDD0026 583.25 to 584.2 m 0.49 g/t Au, 19.4 g/t Ag and 3.4% Cu

12.2 LIMITATIONS

No surface outcrops, drill pads or hole collars were inspected during the site visits and no surface outcrops were inspected. No independent samples were collected for analysis during the site visits. In November 2014 an underground inspection was not possible as road access to the mine at the time of the site visit was temporarily blocked by landslides. In November 2016 visits were made to the rehabilitated underground workings including current underground diamond drilling sites at 1247mRL and 950mRL, and an inspection of the treatment plant.

Very limited mine production data has been located which limits the ability to gain an understanding of reconciliation problems. No face mapping has been found although some photographs of sampled development headings were located. Some stope mapping/sampling sheets from shrink stopes at Irumafimpa were located during the recent site visit. Smith and Thompson (2008) provide the only record of production data.

Nolidan has not been able to fully review all aspects of the project, including:

- Sampling procedures and QA/QC
- Drill collar locations accuracy and reliability
- Drilling procedures
- On-site laboratory assay procedures and performance

Descriptions of existing operations, performance and exploration prospects were obtained from existing documentation which is extensive. Not all documentation was able to be thoroughly reviewed. Industry standard procedures appear to have been used.

12.3 VERIFICATION OPINION

Significant data is available from the previous operators (HPL and Barrick), and is included in the database supplied by Barrick. Based on the data verification performed, it is Nolidan's opinion that the data available and reviewed is adequate for the purposes used in this technical report for resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section refers to both historical information derived prior to commencement of Irumafimpa-Kora operations and subsequent reviews of operations and metallurgical performance which were used as the basis for the planning of the refurbishment of the mill by K92ML. The following descriptions are summarised from Barrick (2014).

13.1 MINERALIZATION CHARACTERISTICS

The main Irumafimpa-Kora lode of the Kainantu Project is sulphide-rich Cu-dominant mineralization overprinted by a quartz-rich Au-dominant crustiform quartz vein to breccia system with high gold associated with tellurides (Calaverite AuTe).

There is currently no geometallurgical model for Irumafimpa or Kora. Assessment of the previous mining operation shows that the inability to inform the plant metallurgists of impending feed characteristics often resulted in dramatic consequences and inefficiencies in the operation of the plant.

13.2 NATURE OF TESTING AND RESULTS

13.2.1 Samples 2000

Initial metallurgical testwork on Kainantu diamond drill core samples was conducted by Metcon Laboratories (Sydney) in 2000. Only a limited amount of testwork was conducted, which included gravity recovery and flotation testing. Leach and Carbon-In-Leach of the whole ore and the flotation concentrate was also conducted.

13.2.2 Irumafimpa Samples - March 2002

Two samples were provided by the Highlands Pacific Group for metallurgical testing. The sample used for testwork is cited in the HRL report as being from the Mill Vein. The quartz lode was originally classified as the Mill Lode, though it was later reclassified as probably being the Puma lode.

The sample tested at HRL was taken from a quartz lode that intersected the main adit drive at 29,934mE 60,060mN (local Irumafimpa Grid). The quartz lode was approximately 1.0 m true width. The sample was recovered from a blast across the full width of the lode, and as such the lode sampled at this point would represent close to a full mining width.

The sample sent to AMDEL for comminution testing was taken from the same location as the sample used for metallurgical testwork at HRL, and would have consisted largely of quartz.

Data from these tests were used for project feasibility studies and plant design.

13.2.3 Kora Testwork 2009

In 2009, test work was completed by AMMTEC on two composite samples from Kora. Composite 1 was described as "High Au Intervals" and Composite 2 was described as "High Cu Intervals". The test work was divided into two stages, the first to determine the grind size and the second to optimise float and gravity recovery at that grind size.

The conclusions were:

- Composite 1 – The test work indicates a recovery of 91.9% of the gold, via gravity (66%) and copper mineral flotation (25.8%) with a concentrate gold content of 200–300 g/t. On the same sample, the copper recovery into the float concentrate is 91.3% with a copper concentrate grade of 20-30% copper. The flotation mass recovery is in the region of 30%.
- Composite 2 - The test work indicates a recovery of 90.3% of the gold, via gravity (61.6%) and copper mineral flotation (28.7%) with a flotation concentrate gold content of 6-7 g/t. On the same sample, the copper recovery into the float concentrate is 90.8% with a copper concentrate grade of 20-25% copper. The flotation mass recovery is in the region of 12%.
- Pyrite Flotation – The gold recovery from the pyrite flotation is relatively low with Composite 1 recovering 2-6% gold and Composite 2 about 5% recovery. The economics of installing a dedicated pyrite flotation plant would have to be closely evaluated before including these recoveries in the overall recovery.

No additional metallurgical work has been undertaken since the testwork was completed by Ammtec in May 2009.

Table 13: Kora recoveries adjusted for saleable concentrate.

Method	Element	Composite 1 High Au Interval	Composite 2 High Cu Interval
Gravity Recovery	Au	66.04%	61.62%
Copper Mineral Flotation Recovery	Au	25.86%	28.71%
	Cu	91.29%	90.80%
Overall Recovery	Au	91.90%	90.33%
	Cu	91.29%	90.80%

13.3 ORIGINAL PROCESS SELECTION AND DESIGN

Test work was conducted for a number of process options including combinations of flotation and leaching. The final process selection was based on bulk flotation to a saleable high gold content concentrate.

The original plant design, engineering and construction were undertaken by Ausenco in 2005. The plant design criteria were based on test work, owner’s information, engineers experience and industry practise. The basic design was:

- Primary jaw crusher;
- Double deck screen with recycle crushers;
- Ball mill with cyclone;
- Flash flotation in the milling circuit;
- Rougher and cleaner flotation;
- Concentrate filtering;
- Tailings disposal dam.

There was initial consideration to install a gravity recovery plant, but this was subsequently removed from the design. The test work conducted identified a suitable depressant to produce an acceptable level of fluorine in the concentrate. Mass and solution balances were developed for 170,000 dry tonnes per year. Equipment selection and sizing followed accepted industry practice and the plant was constructed to a sound quality for a minimum 10 year mine life.

13.4 RECOVERY ASSUMPTIONS

In operation, gold recovery varied considerably since commissioning the plant. It was not possible to consistently realize the recoveries that were achieved with laboratory test work on the ore.

Test work was conducted on site during October-November 2006 by JK Tech. Based on recommendations from this work, operations improved.

Data between January 2007 and November 2007 were reviewed by Barrick to establish a reasonable estimate going forward. During this period, 125,341 tonnes of ore were treated to produce 8,178 tonnes of concentrate, equating to a mass pull of 6.5%. It was noted that mass pull in October and November was approximately 4.5%, which is believed to be due to the addition of lime as a pH modifier to suppress pyrite flotation and increase concentrate grade. However, for the purposes of the study it is assumed that this may not be a sustainable practice, and the average mass pull over the whole time period was used.

The average gold recovery over the same time period was 85% into a copper-gold sulphide concentrate. It should be noted that HPL was able to achieve weekly recoveries of up to 95% on a regular basis.

13.5 REPRESENTIVITY

To the extent known, it is understood the test samples were representative of the various types and styles of mineralization and the mineral deposit as a whole. Added to this is the fact that this was an operational plant processing material directly from the mine.

13.6 FACTORS AFFECTING POTENTIAL ECONOMIC EXTRACTION

Previous operation of the process plant on ore from the Irumafimpa resource provides confidence in the ability to operate and the base assumptions for economic evaluation of future operations – throughput, gold recovery and concentrate grade. The identified issues from testing and early operations (high fluorine in concentrate and low concentrate gold grade) were successfully mitigated through the use of specific gangue depressant and general pyrite depression with lime addition.

The previous operations were able to achieve concentrate sales at satisfactory terms to traditional markets for copper sulphide concentrates and there is every likelihood that a new operation would be able to do the same.

14 MINERAL RESOURCE ESTIMATE

The mineral resource estimate reported in this report uses the same resource block model generated in November 2014 and reported in the NI 43-101 reports by Nolidan dated 01 May 2015 and 15 April 2016. Rock density values used for this current resource estimate have been revised to reflect new information and gold equivalents have been adjusted to reflect current metal values.

In November 2014 after a review of previous resource estimates (see section 6.5 Historical Estimate) Nolidan recommended to Otterburn (K92) that the current resource estimate should be quoted:

- a) Using a standard Ordinary Kriging estimation approach. Grade caps should be selected to restrict the influence of outliers where drilling was sparse.
- b) Cut-offs should be based on a combination of thickness and grade reflecting potential mining methods. Lower cut-off grades of 5g/t AuEq for wide veins (> 3m width) and 6g/t AuEq for veins between 1.2m and 3m width were suggested.
- c) Resources should not be reported at confidence levels above Indicated due to the current drill spacing at Kora and limited confidence in underground sampling information from Irumafimpa.

Following these recommendations Nolidan completed a resource estimate for the Irumafimpa-Kora vein systems based on the historical surface and underground drilling conducted by previous owners, Barrick and HPL. Face channel and grade control samples collected during previous mining operations were also used but have only a local influence.

Comparison of grade control face sampling and drilling in the same mineralized zones shows a significant bias towards lower average grades in drilling compared with the average grade of the face samples. For all veins the highest recorded values for gold (outliers) occurred in drillhole samples and

grade capping was therefore used. Face samples are however concentrated in the higher grade mining areas, so were included in resource estimation.

Nolidan considered that estimation in unfolded 2D space for grade and thickness across narrow veins with allowance for minimum mining widths and unfolding was most applicable to the Kainantu vein system. Industry standard methods were used to conduct the estimate using GEOVIA Surpac™ software. The method utilises estimation in unfolded space. A detailed description is presented in Section 14.7.1 Methodology, and similar methods are widely used in resource estimation (Glacken et al 2014). Vein thickness and grades for Au, Ag, and Cu were estimated in unfolded 2D space before being translated back into a true 3D block model. The model has to incorporate a level of conceptual interpretation (implicit modelling) as the veins are very narrow. Traditional cross section interpretation (explicit modelling) is near impossible due to changes in drill-hole orientation with difficulty in maintaining a true separation of the vein hanging wall and footwall.

14.1 APPROACH

Nolidan considers that there is no appreciable difference in mineralization across the veins, which are narrow (less than 1m in places) and no mining selectivity across the vein is possible. Thus a two dimensional estimate of grade and thickness was considered to be a better method to apply at the Irumafimpa-Kora deposit. In principle the true thickness and grade (and geostatistics) of a vein domain are estimated in unfolded space, i.e. on a 2D grid. This vertical plane is sub-parallel to the vein direction, and grades and thicknesses are absolutely tied to informing samples/composites. The process of "unfolding" and "refolding" results in some smoothing of vein contacts, which may result in minor apparent spatial departures of the vein wireframes from some composite centroids.

14.2 SUPPLIED DATA

Nolidan was supplied with a drillhole database named BARexpldata.mdb. Table 14 shows a summary of the database structure.

Table 14. Master Database Structure

Table Name	Description	Record Count
Collar	Collar information associated with drill type and location	2,611
Survey	Downhole azimuth, dip and depth	8,882
Assay	Assay intervals with associated gold , copper, silver and other results	47,686
Alteration	Logged alteration intervals and descriptions	8,451
Drill	Logging information per hole	85
Veins	Logged intervals with type and degree of veining	6,668
zone_code	Mineralized intercepts used for previous mineral resource estimates	2,679
Structure	Logged intervals of structural geology	7,064

Within the Irumafimpa-Kora resource area, the types of holes available were diamond drilled from the surface (DD), diamond drilled from underground workings (DDUG) and face samples (FS).

A new table was created (named "intercepts") to store vein intercepts in, which were initially copied from the zone code intercept table. MS Access queries were run to ensure mineralization was not excluded adjacent to defined intercepts and un-necessary waste samples were not included. There are examples of vein intercepts with material below cut-off being included, however these tags are required to constrain vein geometry and ensure vein continuity.

There were some mineralized intervals that were not used for resource estimation. These intercepts were given a "UN-" prefix in the "intercepts" table, and present targets for development.

Table 15: Mineralized samples outside vein tags.

Hole_id	depth_from	depth_to	Au_ppm	Ag_ppm	Cu_%
002BD92	87	91	20.24	2.25	0.057
002BD92	201	207	4.391	12	0.093
002BD92	213	214	2.5	4	0.53
003BD92	43.65	44.4	280	26	0.041
004BD92	132.5	133.1	25.3	2.5	0.115
004BD92	156.95	159	4.611	1.378	0.315
011BD94	84	86.3	1.4991	42.3522	1.959
017UG02	6	11	2.952	1.84	0.456
017UG02	23	25	8.835	6.8	0.684
018BD94	240.5	241.5	1.5	65.4	1.05
056BD02	73.5	75	2.28	41	7.76
057BD02	26	28	2.53	11.7	0.232
072BD03	136	138	11.975	11.95	0.235
090BD04	122	124	1.785	106.35	3.212
114UG06	29.5	34.6	3.523	-	-
538AUG08	22.5	25.5	19.6437	-	-

The MS Access database was connected directly to GEOVIA Surpac™ for data display, vein compositing, wire-framing, unfolding, estimation refolding storing in a 3D block model.

The following files were also supplied by K92 Holdings:

- Topography wireframe (Surpac™ DTM) derived from airborne laser (LIDAR) survey
- Surveyed mine workings (declines, inclines, stopes etc. as Surpac™ lines) for Irumafimpa underground development
- Original geological interpretation of veins and faults (Surpac™)

A local mine grid (denoted IG99) oriented roughly parallel to the strike of mineralization was set up by HPL. This grid was used for resource estimation and is based on a 2D rotation from Australian Map Grid (AMG66) coordinates used in exploration. Transformation parameters from AMG66 to IG99 are:

Rotation: 45.4° east, X shift: -9258890.5 m, Y shift: -34421.2 m, Z shift: 0 m

Existing vein intercepts table from the previous vein interpretation was used as a starting point.

14.3 DIMENSIONS

Database extents (Table 16, Figure 23) are for the Irumafimpa-Kora resource area. These coordinates are in mine grid. The database fields used for mine grid are “KAINANTU_IG_X” and “KAINANTU_IG_Y”.

Table 16: Database Extents

Database	Min (m)	Max (m)	Extents (m)
Northing	58207.074	61398.587	3191.513
Easting	29334.635	30413.42	1078.785
RL	730.727	1940	1209.273
Hole Depth	0.3	971.9	971.6

The Irumafimpa-Eutompi-Kora vein system is a 3 km long, 300m wide, northwest trending continuous lode structure with veins across three distinct mineralizing events. As modelled, veins at Kora are between 58100mN and 58950mN, and veins at Irumafimpa are between 59400mN and 61000mN. Between the Irumafimpa and Kora vein systems is the Eutompi area (Figure 31), only one vein (E4) lies in this area and overlaps the Kora area from 58600 mN to 58950 mN.

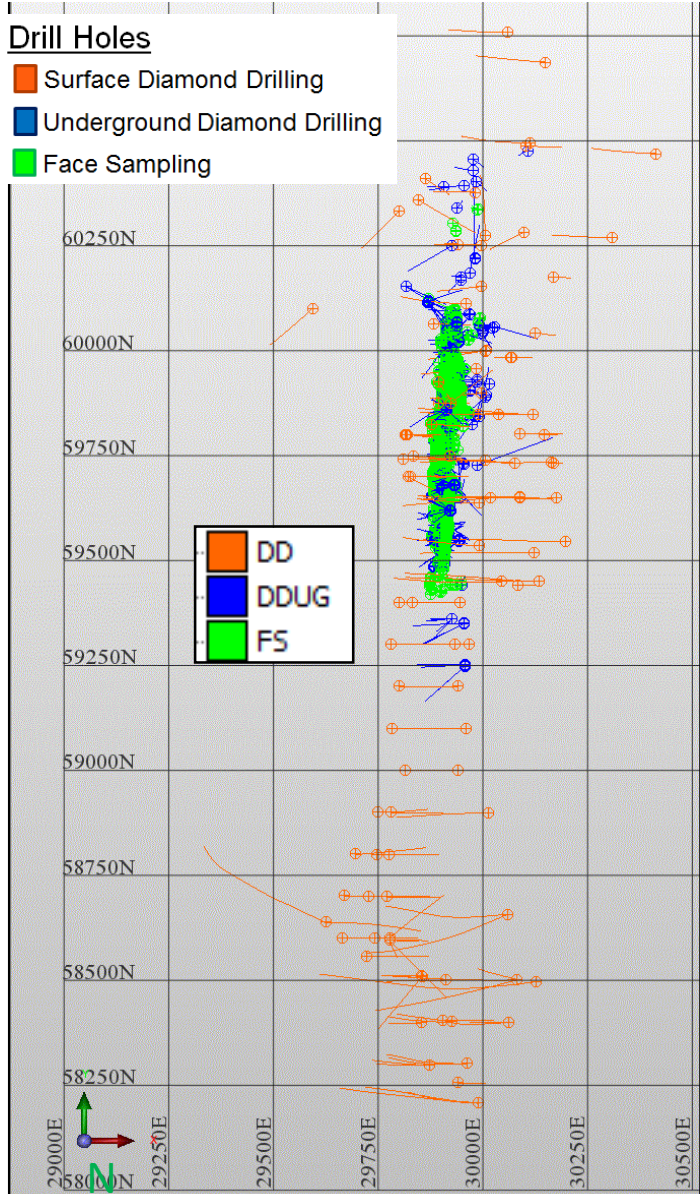


Figure 23: Plan view of the Irumafimpa-Kora Resource drilling, coloured by drillhole type.

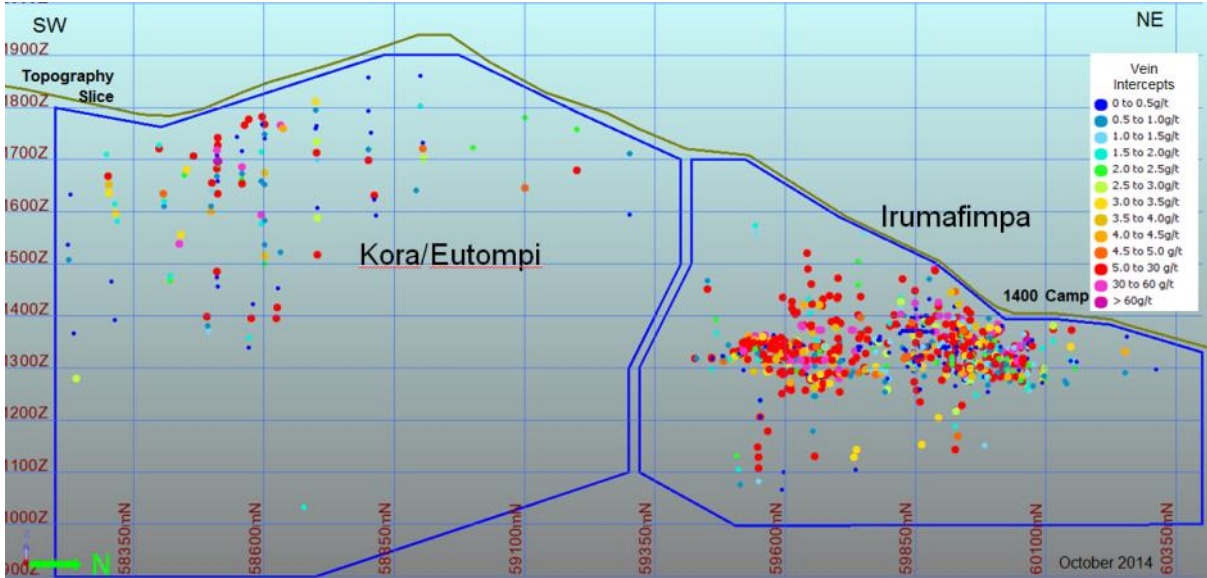


Figure 24: Long section view of Kainantu Resource Areas with Vein Composites colour coded for AuEq

14.4 GEOLOGIC INTERPRETATION

The Kora deposit consists of a series of sub-parallel, north-south striking veins. From west to east these veins are called K3, K2, K5, K1 and, E4. Further to the east are the J4, J3, J2 and J1 veins. The two figures below (Figure 25; Figure 26) show a typical arrangement of the veins at Kora in plan view and cross section.

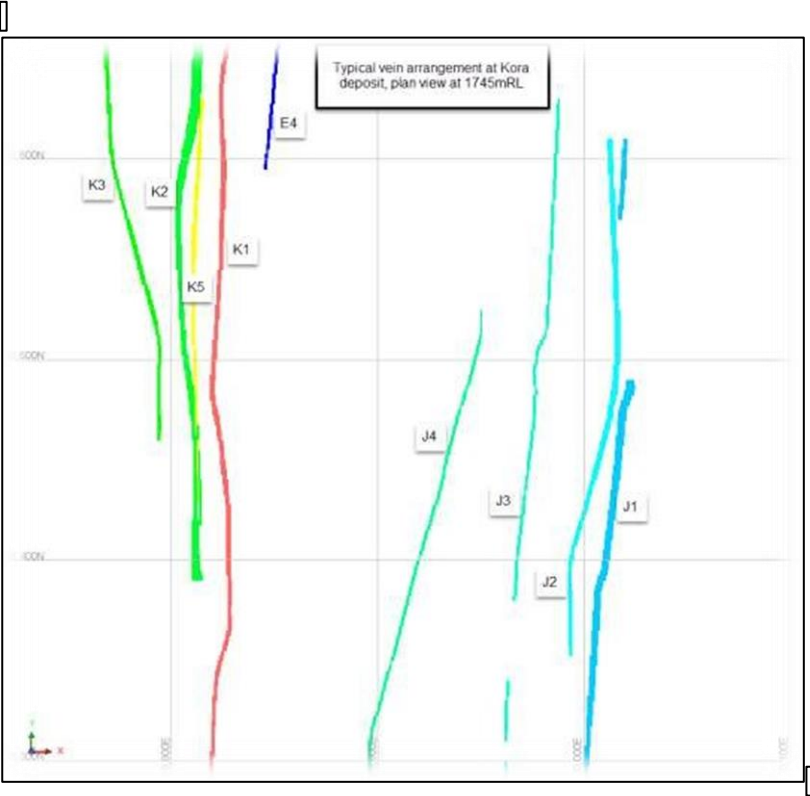


Figure 25: Vein arrangement at Kora, plan view at 1745mRL

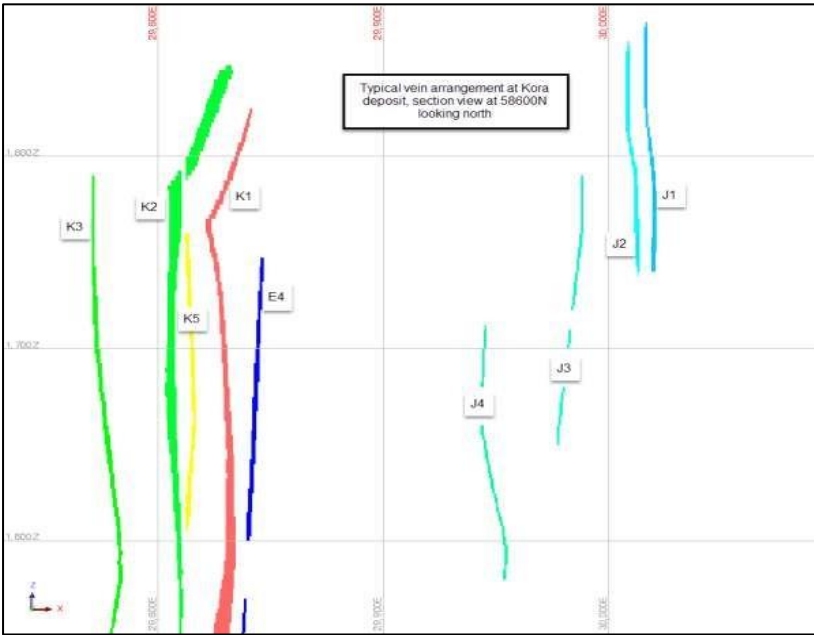


Figure 26: Vein arrangement at Kora, section view at 58600N, looking north

A 3D wireframe model and block model was constructed using a series of procedures within Surpac™. Existing vein intercepts table from the previous vein interpretation was used as a starting point for modelling. Veins were identified as drillhole intercepts greater than 3 g/t AuEq, however assays less than this were incorporated between intercepts to maintain continuity. Printed level plans from site were incorporated into interpretations.

Gold equivalent values were generated in the database using the following formula:

$$AuEq = Au \text{ g/t} + Cu\% * 1.52 + Ag \text{ g/t} * 0.0141$$

This gold equivalent formula is based on current metal prices. Section 14.13 of this report contains a more detailed explanation.

14.5 DATA PREPARATION AND STATISTICAL ANALYSIS

Prior to a statistical analysis, grade domaining is normally required to delineate homogeneous areas of grade data. At Irumafimpa-Kora individual veins are assumed to represent sufficiently homogenous mineralization, although geochemistry of different veins does vary from Kora to Irumafimpa. Statistical analysis does not take into account spatial relationships of the data.

The purpose of statistical analysis is to define the main characteristics of the underlying grade distribution to assist with geological and grade modelling work. This process is important as the statistics of the individual sample populations can influence how grade data is treated and application of grade estimation techniques. For example highly skewed data may require special grade capping and indicator semivariogram analysis.

Statistical analysis of the grade data was principally carried out using the Surpac™ Software package. Surpac™ was used to export composite drillhole data as a comma separated file (CSV) for importation into Supervisor™. More detailed spatial analysis (semi-variograms) was conducted within Supervisor. The Supervisor package is an internationally recognised geological and mining software toolbox which incorporates geostatistical tools that can be used at all stages of the mining process from initial feasibility studies through to production control.

14.5.1 Drillhole Spacing

Drillhole data spacing is variable within each domain.

At Kora, from surface to about 300-500m below surface, there is an average spacing between drillhole intercepts at Irumafimpa-Kora of about 50-70m. Vein intersections below this depth are sparser.

Irumafimpa is much more densely sampled because of underground development. Spacing between vein intercepts is on the order of 20-50 m.

14.5.2 Domains & Stationarity

A domain is a three-dimensional volume that delineates the spatial limits of a single grade population, has a single orientation of grade continuity, and is geologically homogeneous. Statistical and geostatistical parameters are applicable throughout the volume (i.e. the principles of stationarity apply). Typical controls that can be used as boundaries to domains include structural features, weathering, mineralization halos and lithology.

Due to tight geological domaining, stationarity concerns are minimised as each domain contains only one population of grade data.

Kora and Irumafimpa veins have the same strike and dip, and appear to line up on the same structural trend. To determine if the veins could be considered the same domains, and so be reinterpreted and possibly joined into a single large vein system, existing vein composites were extracted and the vein chemistries were inspected.

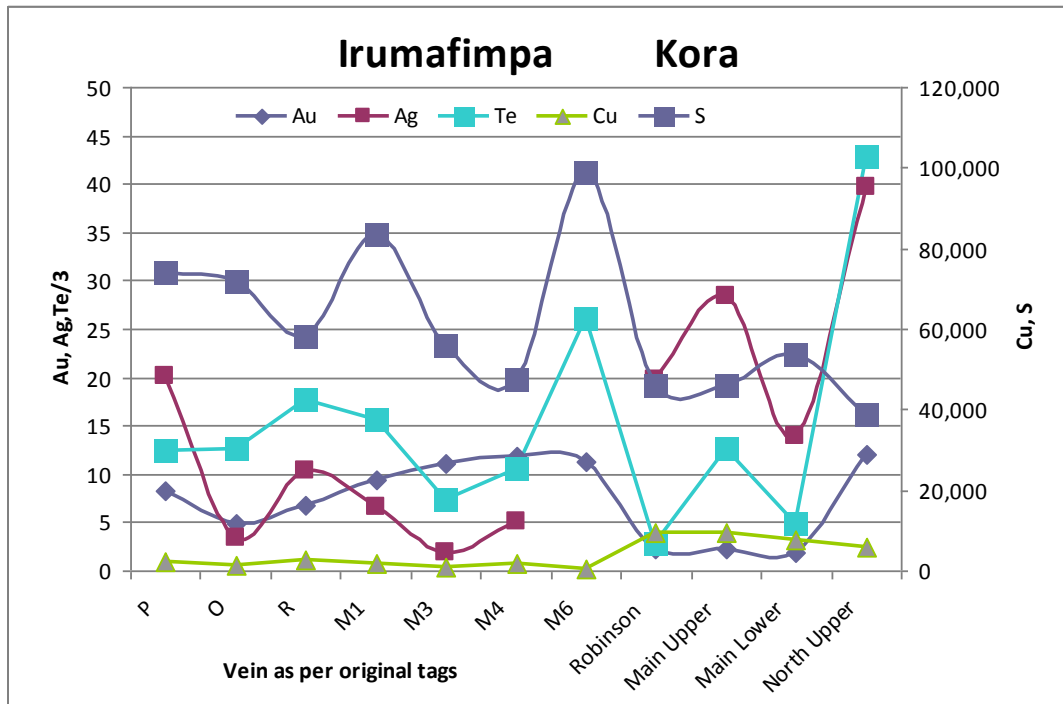


Figure 27: Comparison of Vein Chemistries

Kora veins were found to have relatively higher copper and silver grades and lower tellurium and sulphur grades than Irumafimpa veins, suggesting that they are part of a different phase of mineralization. In addition to this, grades at Eutompi were too low to allow interpretation of any vein mineralization from Kora to Irumafimpa.

Kora and Irumafimpa veins remain separate domains in this resource. In addition Nolidan believes that there has been insufficient drilling to confirm or disprove whether the “IJ” (Irumafimpa Judd) and “J” (Judd) veins are continuous between prospects.

14.5.3 Compositing

The two-dimensional technique used to estimate resources at Irumafimpa-Kora uses a single down-hole (or along channel) composite sample extracted from the drillhole database for each intercept within the vein. True thickness was calculated using the overall dip and dip direction of the vein. It is assumed that the grade of the vein at each location is the grade of the intercept thus reducing concerns of volume variance and negating the need for constant length samples. Scatter plots showed no correlation between grade and thickness, thus grade and thickness are treated as independent samples.

14.5.4 Basic Statistics

Summary statistics for gold, silver and copper in vein intercept composites by vein are presented in Table 17. Informing sample grades range from a minimum of 0.7 g/t Au for Judd 2 (“J2”) to a maximum of 45.6 g/t Au for Judd 1 (“J1”).

Table 17: Univariate uncapped statistics for gold, silver and copper by vein

Vein	No. Composites	Mean Grade			Maximum Grade			CoV		
		Au g/t	Ag g/t	Cu %	Au g/t	Ag g/t	Cu %	Au g/t	Ag g/t	Cu %
E4	21	4.4	26.9	2.0	19.5	110.4	6.3	1.23	1.08	0.83
J1	8	45.6	29.5	1.0	347.7	102.9	3.6	2.68	1.11	1.12
J2	7	0.7	25.9	1.6	1.6	52.1	3.8	0.64	0.74	0.73
J3	7	5.4	15.2	1.0	28.4	32.6	3.9	1.88	0.61	1.34
K1	32	9.8	32.2	2.5	93.1	145.3	7.4	1.95	1.11	0.68
K2	34	12.3	33.8	1.6	178.1	254.9	7.5	2.80	1.50	1.20
K3	13	1.7	21.6	1.2	6.8	74.5	3.1	1.26	0.96	0.65
K5	11	2.6	47.8	1.6	11.0	224.1	3.6	1.15	1.44	0.90
M1	321	10.7	6.0	0.3	155.0	35.3	2.0	1.82	1.27	1.55
M3	173	8.1	1.9	0.1	466.5	4.4	0.8	4.79	0.80	1.54
M4	159	19.7	4.4	0.2	1095.1	14.6	0.9	4.78	0.78	1.15
M5	449	15.8	8.8	0.3	507.9	59.0	1.7	2.69	1.39	1.31
M6	122	12.6	2.5	0.2	205.7	2.5	0.7	2.16	n/a	1.58
O1	79	4.7	3.7	0.2	61.2	13.7	0.9	2.09	1.05	1.22
P2	77	5.9	12.2	0.3	53.1	104.0	1.5	1.34	1.77	1.26
R3	116	6.0	10.4	0.4	70.5	79.4	2.4	1.56	1.25	1.50
IJ1	10	2.2	25.0	0.4	5.8	126.9	1.3	0.71	1.61	1.18
IJ2	9	3.0	6.6	0.3	9.6	27.0	1.3	0.95	1.18	1.46
IJ3	6	2.8	10.2	0.2	4.6	33.0	0.8	0.68	1.25	1.76

14.5.5 Grade Capping

Capping is the process of reducing the grade of the outlier sample to a value that is representative of the surrounding grade distribution. Reducing the value of an outlier sample grade minimises the overestimation of adjacent blocks in the vicinity of an outlier grade value. At no stage are sample grades removed from the database if grade capping is applied. The risks associated with the treatment of the high grades are to potentially overestimate or underestimate the contained metal of the deposit.

Gold and silver are naturally nuggety (Poisson distribution) in nature and prone to outliers. Statistical parameters such as coefficient of variation and mean plots, metal loss, histograms and log probability plots were used as guides to determine the appropriate grade cap. The effect of capping can be seen by comparing statistics of uncapped and capped distributions.

In previous estimates of Irumafimpa-Kora, composite grades were capped to create an estimate. This was done because high grade outlier composites have an overwhelming influence on any blocks for which they are used to estimate. Capping the grade reduces the amount of metal that will be estimated into blocks informed by these outlier samples, hopefully preventing overestimation. Outlier composites have passed QA/QC, and so are considered real values that represent the grade of the vein at the composite location. The problem with these outlier composites is not strictly that the grades are too high or are not considered real or reliable, but that the effect on the blocks within the range of these high grade outlier composites will be higher than for other composites.

To effectively deal with high grade outliers in this resource, the composites for gold grades were reviewed and appropriate caps assessed.

Composite caps were applied to the grade values (g/t for Au and Ag, % for Cu) before estimation. Capped versus uncapped grade statistics were generated for gold, silver, and copper. Sulphur did not have enough samples so was not capped.

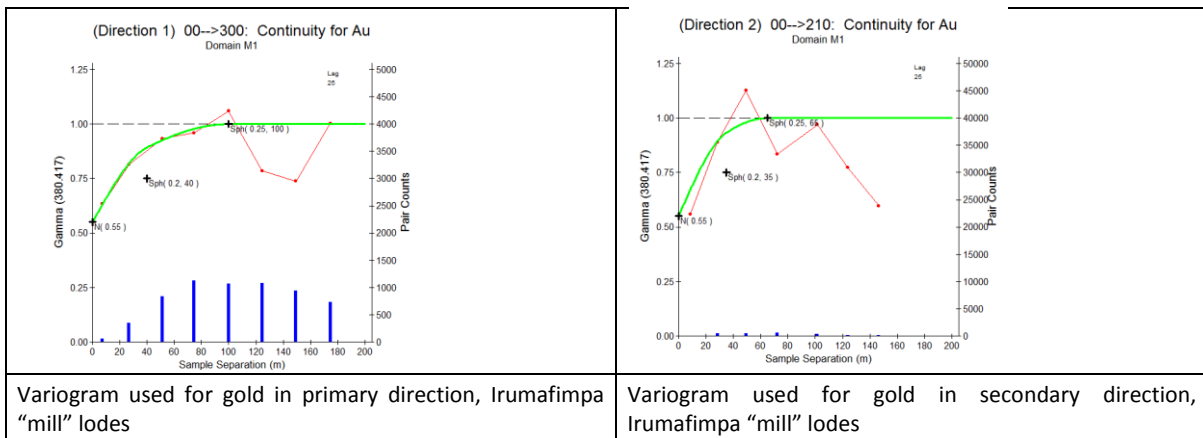
Table 18: Grade caps for gold by vein

Vein	No. Samples	Uncapped Mean Grade	Capped Mean Grade	Uncapped Coefficient of Variation	Capped Coefficient of Variation	Uncapped Maximum Grade	Suggested real cap	No. Samples capped
E4	21	4.4	4.3	1.23	1.21	19.5	n/a	0
J1	8	45.6	30.8	2.68	2.60	347.7	150	1
J2	7	0.7	0.7	0.64	0.63	1.6	n/a	0
J3	7	5.4	4.9	1.88	1.80	28.4	n/a	0
K1	32	9.8	9.3	1.95	1.85	93.1	80	1
K2	34	13.3	12.3	2.80	2.29	178.1	80	2
K3	13	1.7	1.7	1.26	1.26	6.8	n/a	0
K5	11	2.6	2.6	1.15	1.15	11.0	n/a	0
M1	321	10.7	10.2	1.82	1.65	155.0	80	6
M3	173	8.1	5.2	4.79	2.15	466.5	80	2
M4	159	19.7	10.7	4.78	2.03	1095.1	80	7
M5	449	15.8	13.7	2.69	1.91	507.9	150	6
M6	122	12.6	10.6	2.16	1.51	205.7	80	3
O1	79	4.7	4.3	2.09	1.84	61.2	38	2
P2	77	5.9	6.3	1.34	1.23	53.1	n/a	0
R3	116	6.0	5.8	1.56	1.37	70.5	n/a	0
IJ1	10	2.2	2.2	0.71	0.68	5.8	n/a	0
IJ2	9	3.0	2.9	0.95	0.91	9.6	n/a	0
IJ3	6	2.8	2.8	0.68	0.68	4.6	n/a	0
IJ4	3	1.8	1.8	0.57	0.57	2.5	n/a	0

14.6 VARIOGRAPHY

The most important bivariate statistic used in geostatistics is the semivariogram. The experimental semivariogram is estimated as half the average of squared differences between data separated exactly by a distance vector ‘h’. Semivariogram models used in grade estimation should incorporate the main spatial characteristics of the underlying grade distribution at the scale at which mining is likely to occur.

The semivariogram analysis was undertaken for individual elements within each vein domain that contain sufficient data to allow a semivariogram to be generated. 2D semivariograms were generated using two orthogonal principal directions.



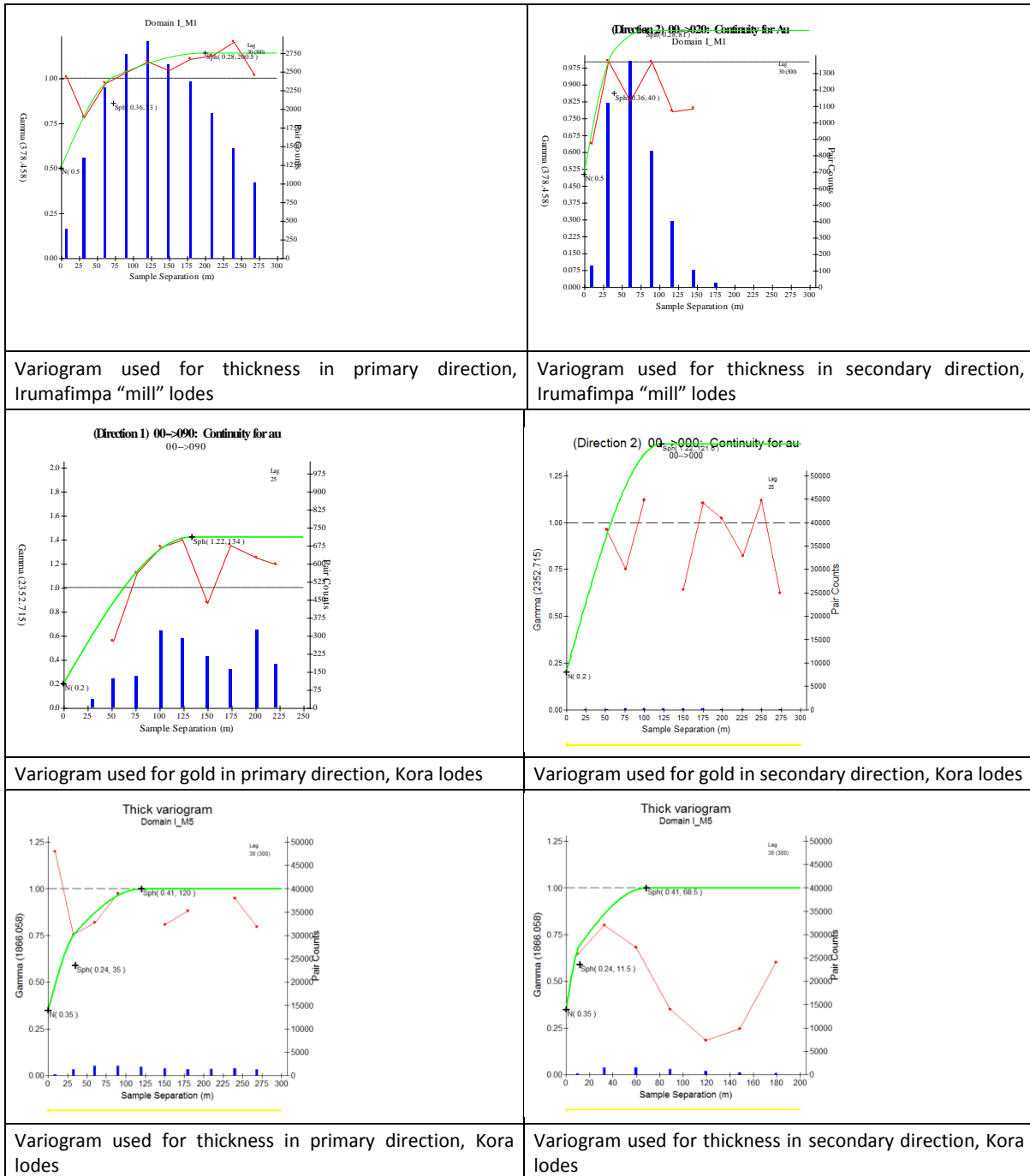


Figure 28 Variograms generated for veins.

14.6.1 Methodology

All variograms were 2D and in the plane of the vein. Anisotropic variograms were constructed for vein domain true widths, as well as gold and copper grade values in all vein domains. This was performed using vein composites individually, although most veins had too little data to generate reliable variograms. There were not as many sulphur and silver assays as gold or copper, so reliable variograms were not able to be constructed using these values. Sulphur and silver showed the closest relationship to gold values and so were interpolated using gold variogram parameters.

After extensive testing of changing variogram and other estimation parameters for each variable the estimation results were found to be sensible and consistent.

14.6.2 Variogram Models and Parameters

There were insufficient vein composites to allow variograms to be constructed for every vein. Variogram models were instead constructed for the veins or groups of veins with sufficient data and used for other veins nearby which did not have enough intercepts.

At Kora, a variogram was constructed for gold for the combined Kora veins (all Kora veins except for Judd). These variogram parameters were used to estimate gold for all of Kora and all of the Judd veins. At Irumafimpa, a gold variogram was constructed from domain M1. These variogram parameters were used to estimate gold for all Irumafimpa veins. A full table of estimation parameters for all veins and attributes can be found in Table 19.

Table 19: Semivariogram Parameters used for Irumafimpa-Kora estimation

Vein Set	Attribute	plunge	Max Range	C ₀	C ₁	A ₁	C ₂	A ₂	ratio1	ratio2
Kora, Judd	Au, Ag, S	80	130	0.2	1.24	130	0	0	2	2
Irumafimpa	Au, Ag, S	300	100	0.55	0.2	40	0.25	100	1.14	1.54
All veins	Cu	80	130	0.2	0.3	50	0.62	140	2	2
Kora, Judd	Width	80	130	0.3	0.38	88	0.18	256	1	1.44
Irumafimpa	Width	300	200	0.5	0.23	35	0.27	200	3.5	2.5

14.7 GRADE ESTIMATION

Estimates were made for the grades and true widths of veins. This is done in unfolded space using 10m x and y grid spacing. The estimation area is extended beyond the outer data points by expansion of a fixed distance to create a boundary perimeter; the boundary is then smoothed with the result that the expansion is reduced to less than the target thickness at the extremities. The expansion distance is therefore a maximum, rather than a fixed value. The expansion for Irumafimpa-Kora is a maximum of 50m.

Grade estimations are made using five different methods so that the results can be compared: Nearest Neighbour (capped), Inverse Distance Squared (capped), Ordinary Kriging (uncapped), Ordinary Kriging (capped) and metal content (gram-metres). True widths are estimated directly using Ordinary Kriging (no capping).

One block model was created, covering the entire deposit. The final 3D block model utilised 2.5(x)*10(y)*10(z) m cubic blocks sub-blocked to 0.625(x)*2.5(y)*2.5(z) metres.

14.7.1 Methodology

Comprises the following steps:

1. **Database** – validation of the drillhole database.
2. **Intercept Selection.** The drillhole data is displayed in section and elevation slices showing assays. Intercepts are selected and coded for each vein based on the following selection criteria, in priority order;
 - a. Grade – select intervals with a value above cut-off, in this case 3 g/t AuEq. Also, internal waste intervals and/or geologically continuous intervals just below cut-off may be included, as long as the composite remains above cut-off.
 - b. Continuity – waste (<3 g/t AuEq) values in the projected plane of continuity of a particular vein being modelled will be coded as that vein.
3. **Basic Statistics and Upper Caps.** The basic statistics of the vein composites for each vein are then examined using basic statistics for grades and true width. The mean, median, standard deviation and variance are calculated for both normal and log-transformed data. A cumulative probability plot is prepared for each data set in both normal and log-transformed formats. Breaks in the plot indicating more than one population are highlighted and their spatial

position relative to the total data set examined in 3D space. If more than one population is considered possible, the total population is decomposed into its component populations and these are highlighted again in 3D space. If a small high-grade population is indicated, and this cannot be physically domained from the remainder, then an estimate with an upper cap will be included in the resource estimates.

4. **Unfolding and Variography.** Vein composites are unfolded into a single plane. Original coordinates are stored in the model so the model may be refolded after estimation. Variography is then undertaken in this 2D space. Values for anisotropy and variogram models are recorded for gold, thickness and copper or silver as appropriate. Where no directional variograms are clearly determined (as commonly happens with less than 50 data points, or where the data is unevenly distributed) isotropic variograms were used or variograms from similar veins sets where utilised.
5. **Unfolded Grid Model and Extension** – Generates a model of the vein centre using coded intercepts, and estimates grades and vein true widths. This is done in unfolded space using selectable x and y grid spacings. The estimation area is extended beyond the outer data points by expansion of a fixed distance (50 m Kora, Eutompi and Judd, 25m Irumafimpa) to create a boundary perimeter; the boundary is then smoothed with the result that the expansion is reduced to less than the target expansion at the extremities. The expansion distance is therefore a maximum, rather than a fixed value. In extreme cases, say where the extension is based on an isolated single drillhole, no extension will occur at all. Expanded wireframes are checked in 3D space to ensure the expansion does not intersect waste drillholes. The thickness of this boundary is set to 0.2 m. This prevents an overflow of grade contours past the limits of estimation. Grade estimates are made using 5 different methods so that the results can be compared. These are Nearest Neighbour Capped, Inverse Distance Squared Capped, Ordinary Kriging Uncapped and Ordinary Kriging Upper Capped and gram-metre estimates. True widths are estimated directly using Ordinary Kriging.
6. **Minimum Width application and consequent Grade Dilution** – Every 10 x 10 m block in unfolded space with a vein width (in the perpendicular direction to strike) less than 1.2m is set to a width of 1.2 m. Grades for each block are then diluted according to the original width and waste grade (0.0 g/t), using the following formula:
$$\text{Diluted grade} = (\text{grade} \times [\text{true thickness}/\text{minimum thickness}]) + (0 \text{ g/t} \times [\text{dilution thickness}/\text{minimum thickness}])$$

Blocks with a width greater than 1.2 m have no change. This dilution will raise the tonnes and reduce the grade of the model; however, the total ounces of gold will remain about the same. The process of applying a minimum width is to reflect the minimum mining width and apply an appropriate dilution where veins are thinner than the mining width.
7. **Refolding and True Width Correction** – The grid is re-folded to its original 3D position. This is done by replacing the unfolded coordinates with the stored real coordinates. Some smoothing of the surface using surface modelling algorithms (not geostatistics) is undertaken; this removes local spikes and steps due to clustering of data. Changes are small, generally less than half the grid spacing. The “slope” of the surface in 3D space relative to the 2D surface is then measured as a percentage gradient; this value is recorded as it is similar to that used in “Connolly Diagrams”. The True Width value is then corrected using this factor. Note that “slope” value is measured at each node of the grid and is a function of the surface geometry; the more the surface moves from the projection plane the greater the correction – in effect an “auto-correction”. This is considered much better than using an average strike and dip for the surface (too general), a drill core measurement (too local) or geostatistics (too smoothed).

8. **Solid Creation** – The 3D centre plane of the vein is then converted to a closed 3D solid. Footwall and hanging wall surfaces are created by translating the 3D centre plane half the width of the vein to create footwall and hanging wall surface. These are then joined at the edge, which is a common boundary, to create a vein solid. If more than one vein is being estimated, then the interaction between the resultant solids is examined and portions of the minor veins removed via “clipping”.
9. **Block Model** – The volumes from the final closed 3D solids are used to flag blocks in the final 3D block model for each vein. The variables from the solids, including grades, widths, slope, kriging variance, number of informing samples, nearest drillhole name and distances, etc., are all stored in the block model. Each vein block is given a vein name and number.

Determining the Krige Combined Grade:
 - a. All blocks are assigned the capped Krige estimated grade.
 - b. The nearest neighbour estimate is performed using uncapped grades, if the NN grade is higher than the grade cap, and the Krige uncapped value is assigned, provided the block is within 25 m of the outlier assay.
 - c. NN estimate is then capped to the appropriate capped value.
10. **Bulk Density** – The bulk densities for each block below the topographical surface are set to a constant value.
11. **Missing Blocks** – blocks that are not present are flagged as air (above the original topography), pit (mined out in an open pit), stoped (removed by underground mining).
12. **Mineral Resource categories** – the resource categories are defined in long-section view for each vein, based on a combination of the number of informing samples, sample distances and kriging variance. The mineral resource categories are stored in the block model field.
13. **Validation** – The values within the block model are compared to the informing drill composites. Basic statistics for block model and drill composites are compared. Distributions of grades in space (by elevation and northing) are compared. Blocks nearest to drillholes are compared with the informing drillholes. The estimates using the different estimation methods are compared in total and above cut-off.
14. **Reporting** – the resource can be reported by resource category, by vein, by cut-off grades, by different methods (sensitivity to method and upper cuts), by elevation (tonnes per vertical m), by thickness, and by x and y dimensions.

14.7.2 Block Model

The Irumafimpa-Kora 3D block model uses regular shaped blocks measuring 10m (y) x 2.5m (x) x 10m (z) (Table 20). The choice of the block size was patterned with the trend and continuity of the mineralization, taking into account the dominant drill pattern and size and orientation of the veins. The orientation of the block model is normal to the direction of the local grid. To accurately measure the volume of the mineralized wireframe inside each block, volume sub-blocking to 2.5m (y) x 0.625m (x) x 2.5m (z) was used. Blocks above topography were tagged and excluded from model estimation.

Table 20: Block Model Extents

Type	Y	X	Z
Minimum Coordinates	58,000	29,700	900
Maximum Coordinates	61,200	30,250	2000
User Block Size	10	2.5	10
Min. Block Size	2.5	0.625	2.5
Rotation	0.0	0.0	0.0

14.7.3 Informing Samples and Search Parameters

Informing samples are composited across the vein, providing a local average across the vein width before estimation. Using average grades across a vein requires careful consideration of the number of informing samples used to prevent over smoothing of the estimate. A minimum of one vein composite and a maximum of eight vein composites were permitted to inform a block. The number of samples per vein composites depends on the thickness of the vein and the orientation of the drillhole to the vein.

Search radii were found to be optimal at or near the distance that the variogram reached the sill. Thus the variogram ranges were utilised in the maximum search distances (Table 21). Anisotropy apparent in the variogram analysis is reflected in the search ellipse. Only one pass was used to inform the blocks. All of the plunges in Table 21 are relative to the plane of the vein (dip-90, dip direction 270).

Table 21: Search Parameters

Veins	Gold			Width		
	Search Distance (Au)	2D Anisotropic ratio (Au)	Plunge Direction	Search Distance (m)	2D Anisotropic ratio (m)	Plunge Direction
K1	130	2	80	130	1.44	80
K2	130	2	80	130	1.44	80
K3	130	2	80	130	1.44	80
E4	130	2	80	130	1.44	80
K5	130	2	80	130	1.44	80
J1	130	2	80	130	1.44	80
J2	130	2	80	130	1.44	80
J3	130	2	80	130	1.44	80
J4	130	2	80	130	1.44	80
M1	100	1.54	300	200	2.5	80
M3	100	1.54	300	200	2.5	80
M4	100	1.54	300	200	2.5	80
M5	100	1.54	300	200	2.5	80
M6	100	1.54	300	200	2.5	80
M7	100	1.54	300	200	2.5	80
O1	100	1.54	300	200	2.5	80
P2	100	1.54	300	200	2.5	80
R3	100	1.54	300	200	2.5	80
IJ1	130	2	80	130	1.44	80
IJ2	130	2	80	130	1.44	80
IJ3	130	2	80	130	1.44	80

14.7.4 Discretisation

The Kriging estimate used a 4 x 4 x 1 discretization (XYZ), giving discretization nodes spaced evenly within the block. The projection plane direction has no thickness (2D unfolded space) thus one discretization point is applied, which corresponds with the across vein direction.

14.7.5 Block Model Attributes

Interpreted mineralized veins were coded to the block model. Sufficient variables were added to allow grade estimation, resource classification and reporting. Blocks above the original topography were coded as air and not estimated. Blocks that have been mined were flagged in the final block model; these blocks were estimated for reconciliation purposes. To simplify and reduce the size of the block model several attributes were removed from the final model. Block model attributes are defined in Table 22.

Table 22: Block Model Attributes

Attribute Name	Type	Decimals	Background	Description
ag_ok_ct	Real	3	0	Ag_ok_ct -> diluted
au_gm	Real	-	0	Au g.m
au_gm_ct	Real	3	0	Au g.m -> diluted
au_id_ct	Real	3	0	Au_id_un -> diluted
au_nn_ct	Real	3	0	Au_nn_ct -> diluted
au_ok_ct	Real	3	0	Au_ok_ct -> diluted
au_ok_un	Real	3	0	Au_ok_un -> diluted
cu_ok_ct	Real	3	0	Cu_ok_ct -> diluted
density	Real	2	2.5	density of rock
dh_length	Real	2	0	DH_length
hole_id	Character	-		hole id
hori_thk	Real	-	0	horizontal thickness
min_mining_wdh	Real	-	0	minimum mining width
min_thk	Real	-	0	minimum thickness
mined	Integer	-	0	0 insitu 1 mined
rescat	Integer	-	4	1 measured, 2 indicated, 3 inferred, 4 waste
s_ok_ct	Real	3	0	S_ok_ct -> diluted
slope	Real	-	0	slope
true_width	Real	-	0	true width
vein_name	Character	-	W	Vein Name
vert_thk	Real	-	0	vertical thick
zok_cbs	Real	-	0	Conditional bias
zok_dns	Real	-	0	distance to nearest sample
zok_kv	Real	-	0	kriging variance
zok_ns	Integer	-	0	number samples

14.8 VALIDATION AND COMPARISON WITH ALTERNATIVE ESTIMATES

Block models were validated by visual and statistical comparison of drillhole and block grades and through grade-tonnage analysis. Initial comparisons occurred visually on screen, using extracted composite samples and block models.

Table 23: Alternate estimation results at a 5g/t Au cut-off (drill samples only)

Estimation method cut-off used	Mt	Au g/t	Oz Au
Ordinary Kriging capped	3,882,000	9.7	1,211,000
Ordinary Kriging uncapped & undiluted	4,064,000	18.5	2,413,000
Gram Metres capped	4,352,000	9.2	1,281,000
Inverse Distance squared capped	3,546,000	10.4	1,185,000
Nearest Neighbour capped	2,726,000	13.0	1,141,000

Alternative estimation methods using drill samples only (Table 23) were utilised to ensure the kriging estimates were not reporting a global bias, such as nearest neighbour and the back calculated grades from grams x metres (g.m) estimates. The alternate estimates provided expected correlations. Nearest neighbour shows less tonnes and higher grade as it does not employ averaging techniques to assign the block grade. The Ordinary Krige uncapped undiluted estimate highlights the narrow nature of the deposit, not accounting for mining thickness allows significant narrow tonnes to be included (>5g/t) which are diluted to below 5g/t Au when mining thickness is considered, and also extends the very high grade areas much further than is realistic where drilling is sparse (inferred areas). The ID² estimate is closer to kriging as it uses distance weighted averages, but cannot assign anisotropy nor has the ability to decluster input data or nugget effect. Gold grades back-calculated from g.m appeared over-smoothed, a predictable consequence of using the thickness variogram for both g.m and thickness. The ordinary Krige estimate is the most reliable due to the ability of kriging to decluster data and weight the samples based on a variogram (which incorporates anisotropy). Grade capping has a deliberate impact on grade; a harsh grade cap was applied to limit the effect of outliers in areas of limited data. The ordinary Krige combined (the tightly controlled combination of uncapped blocks in close proximity to high grade drill intercepts and the capped Krige estimate) accounts for the expected

high grade shoots, without over-smoothing the outliers or increasing the expected number of high grade shoots in areas that have not demonstrated the existence of shoots.

The ordinary Kriging capped estimate is used for reporting of mineral resources.

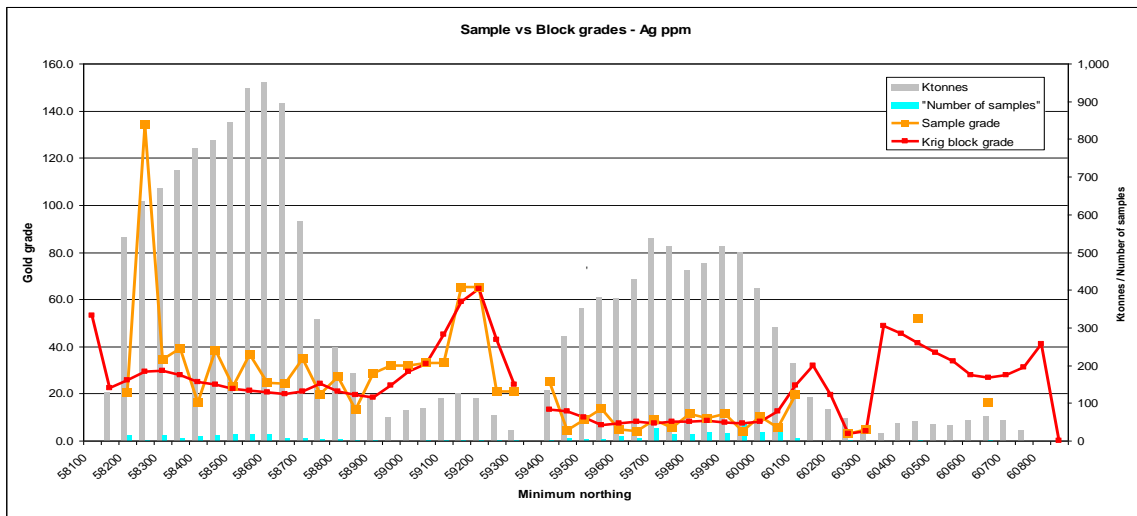


Figure 29: Trend analysis by northing for Ag g/t. (Kora to the left)

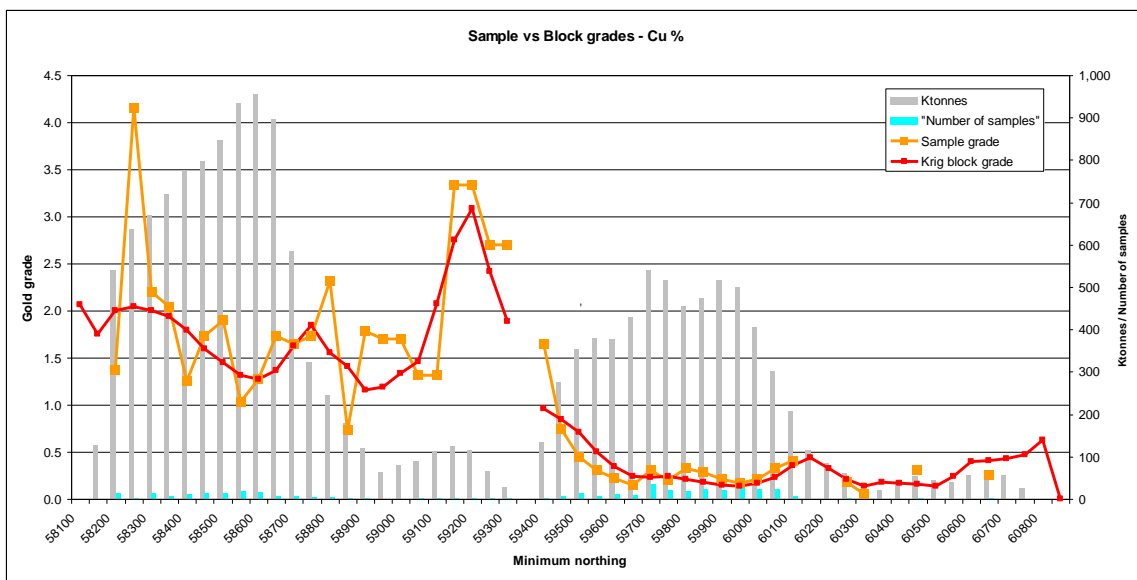


Figure 30: Trend analysis by northing for Cu % (Kora to the left)

Trend analysis was performed to compare input data (informing drillhole composites) with block estimates. Silver and copper (Figure 29 and Figure 30) showed good correlation across the entire deposit, and different mineralogy in the different areas become very apparent. Features expected from a successful trend analysis were shown, with block estimates showing a smoother, more averaged grade trend line than the more variable input data. More variability was found where there were fewer blocks or available informing composites.

Gold trend analyses were created with multiple estimation techniques displayed on them to further validate the resource. These did not initially show the same good correlation between input data and estimation results, especially at Irumafimpa (Figure 31).

Capping the high grade outliers at Kora caused much lower estimates than the uncapped input data. The very high input data grades shown are the product of only very few clustered high grade outliers. With the uncapped estimate trend line lying roughly between the input data and the final combined

estimate it is Nolidan’s opinion that the capping is applicable and the Kora estimate is correctly conservative in this case.

Initially at Irumafimpa none of the block estimates reflected the high grades shown in the input data, with the uncapped block estimate not even showing the same relative jump as was shown at Kora. It was found that when Irumafimpa veins were split into the Judd and Mill vein systems, a much more satisfactory sample block comparison was displayed (Figure 32).

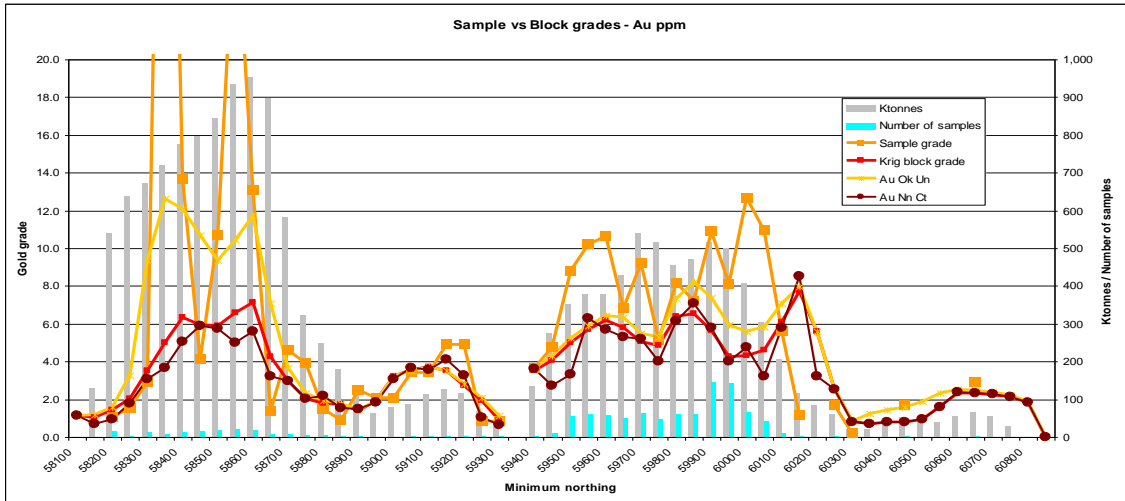


Figure 31: Trend analysis by northing for Au g/t (Kora to the left)

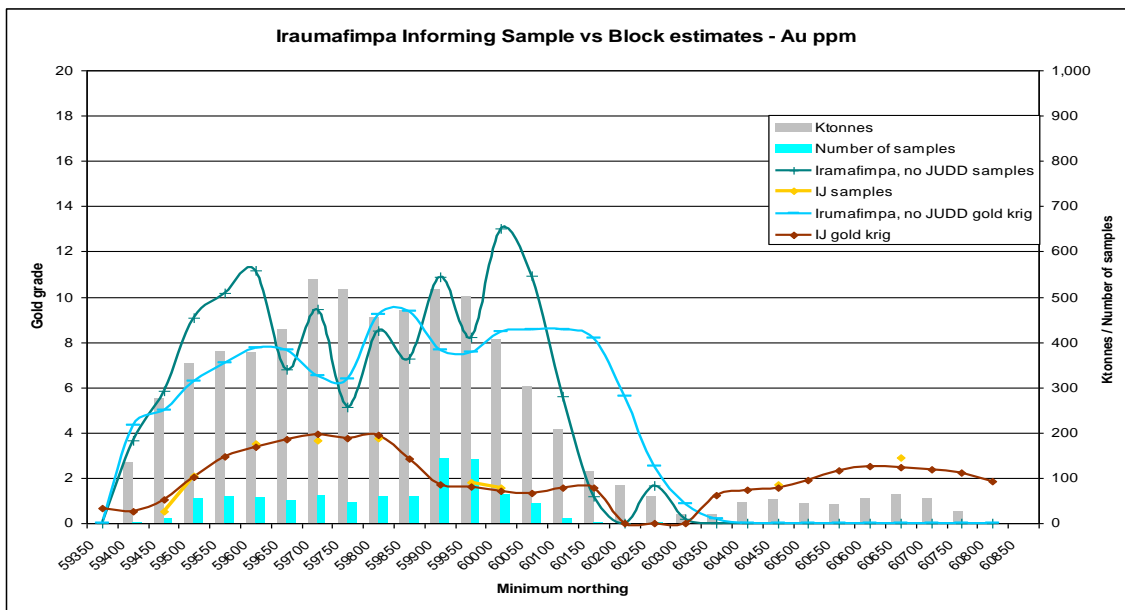


Figure 32: Trend analysis by northing for Au g/t at Irumafimpa with Judd and Mill veins separated

14.9 ECONOMIC CUT-OFF PARAMETERS

All resources have been stated above a combination gold equivalent and thickness cut-off. The model has been diluted to 1.2m thickness, so technically there are no resource blocks less than 1.2 m thick, however blocks still need to be above the grade cut-off. The two mutually exclusive cutoffs used (which took mining method, metallurgical recoveries, and royalties into consideration) were:

1. Narrow Vein -Shrink Stopes - 1.2m – 3m thick and $\geq 6\text{g/t AuEq}$
2. Wide Vein – Mechanised Stopes - $>3\text{m}$ thick and $\geq 5\text{g/t AuEq}$

These parameters are based on the different mining methods that would be used depending on the width of the vein. Parts of the vein between 1.2m and 3m thick could be most efficiently mined using

a method such as shrink stoping, which typically has a higher cost than methods used in larger stopes such as cut and fill. Veins greater than 3m thick are typically mined using cheaper mechanised mining techniques; hence the lower gold equivalent cut-off grades used in the thicker parts of veins. This combination of different mining methods matched with cut-offs is to ensure that all material reported in the resource has a reasonable prospect of extraction.

Grade tonnage charts (Figure 33) are reported above 0 g/t AuEq (irrespective of vein thickness, but with 1.2m width applied) in 1 g/t increments. The charts indicate the current indicated resource has only a slightly higher grade than the inferred resource, and similar charts. The resource is expected to be mined by different mining methods depending on vein thickness and geological complexity; both vein thickness and grade need to be considered when defining a resource cut-off.

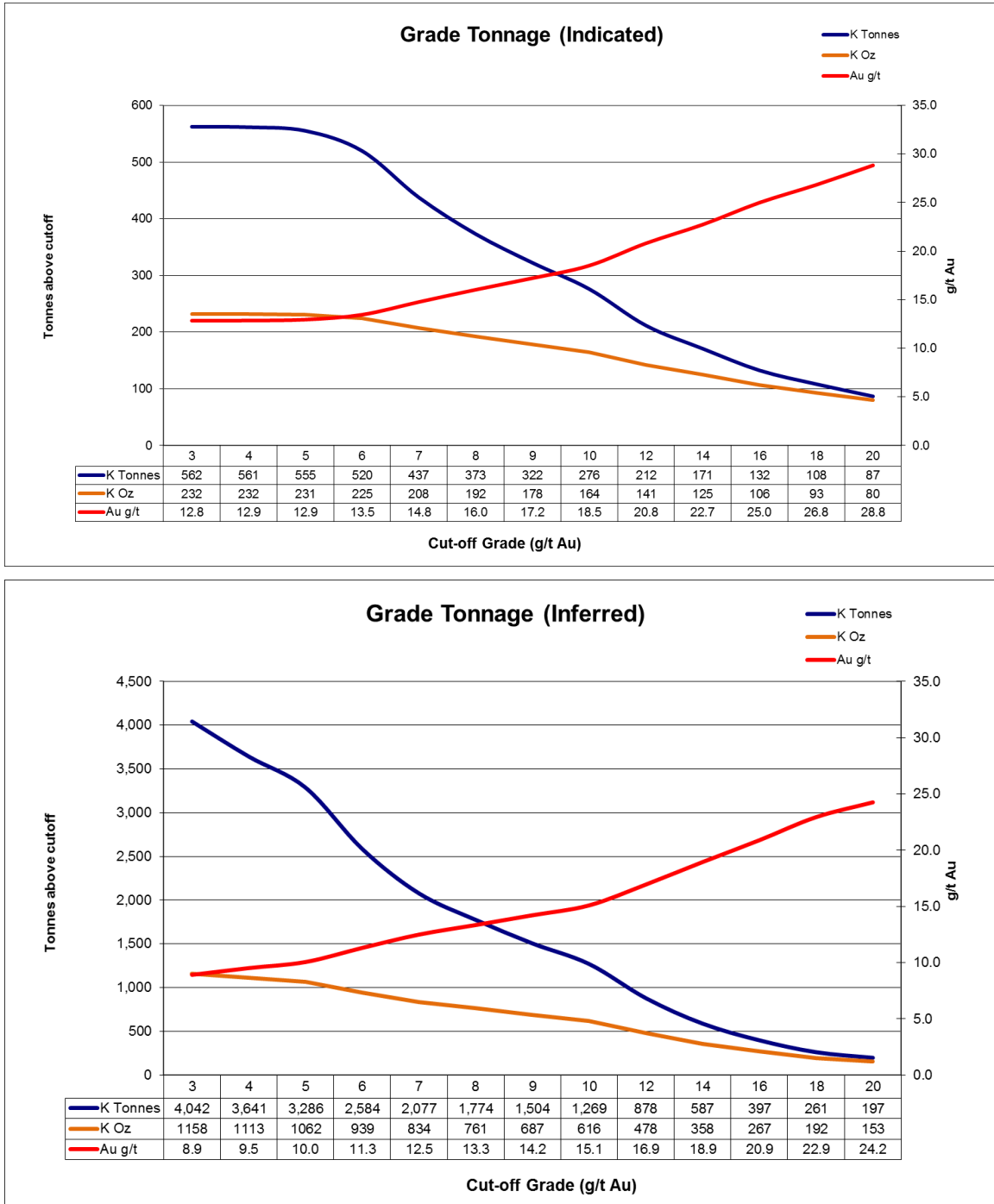


Figure 33: Irumafimpa-Kora Grade Tonnage Charts

14.10 BULK DENSITY

During the initial 2002 feasibility study HPL carried out density determinations on 35 samples sourced predominantly from the Irumafimpa exploration adit. Density of these samples ranged from 2.9 t/m³ to 3.7 t/m³. HPL used a default density of 2.9 t/m³. This incorporated a correction for voids which constitute approximately 10% of the total volume of the Irumafimpa lodes (SRK, 2006). Historic resource estimates by Hackchester (2005) and Mining Associates (2006) used an average density value of 2.9 t/m³.

Barrick made 428 density measurements of drillcore from Kora. These were mostly waste material but included 5 intersections of the interpreted Mill and Robinson veins. Densities were determined using the water immersion method (Bond, R., Dobe, J., & Fallon, M., 2009). Average values from measurements for lode material ranged from 2.58 t/m³ to 2.77 t/m³. These values were considered by Barrick to be too low for the sulphide rich zones and the average dry bulk density value was adjusted to 2.75 t/m³ for Kora vein material.

For the current resource estimate vein blocks in the Irumafimpa deposit have been assigned a density of 2.9 t/m³ and vein blocks in the Kora deposit have been assigned a density of 2.8 t/m³.

14.11 MOISTURE

No measurements were recorded; the bulk density figure applied was dry.

14.12 MINING & METALLURGICAL FACTORS

The tonnes and grade of the mineral resource estimates are reported in situ.

14.13 ASSUMPTIONS FOR 'REASONABLE PROSPECTS FOR EVENTUAL ECONOMIC EXTRACTION'

Assumptions for reasonable prospects for eventual economic extraction applied to this deposit include but may not be limited to the following:

- Underground mining by either shrink stoping or mechanised mining depending on vein width.
- Copper price at US\$2.66/lb (12 month average to December 2016 (\$2.23))
- Gold price at US\$1200/Oz (12 month average to December 2016 (\$1251); discounted due to apparent falling trend)
- Silver price at US\$16.91/Oz (12 month average to December 2016 (\$17.10); silver is a minor economic contributor)
- Assumed Mill Recoveries of 85% for all metals (and is therefore not a factor in the equivalence formula).

Gold equivalent values were generated in the database using the following formula:

$$AuEq = Au \text{ g/t} + Cu\% * 1.52 + Ag \text{ g/t} * 0.0141$$

*Metal prices were obtained from www.kitco.com and www.kitcometals.com

Therefore cut-off grades for reporting were a combination of thickness and grade reflecting mining methods:

- a. Narrow Vein - Shrink Stopes - 1.2 m – 3 m thick and $\geq 6\text{g/t AuEq}$
- b. Wide Vein – Mechanised Stopes - >3 m thick and $\geq 5\text{g/t AuEq}$

14.14 RESOURCE CLASSIFICATION

Based on the study herein reported, delineated mineralization of the Irumafimpa-Kora deposit are classified as a mineral resources according to the JORC Code 2012 Edition definitions which are

considered as being not materially different from how those terms are defined under CIM Definition Standards.

Reporting of tonnages and grade figures reflects the relative uncertainty of the estimate, and rounding to the appropriately significant figures have been reported, some discrepancy in the addition of rounded figures may occur. Mined blocks have been removed prior to reporting.

For the classification of Mineral Resources for the Project, a block had to pass the reasonable prospects for extraction criteria based on an assumed mining method, that is;

- 1.2m to 3m thick and $\geq 6\text{g/t AuEq}$, (assumed appropriate for hand held mining equipment)
- or
- $>3\text{m}$ thick and $\geq 5\text{g/t AuEq}$ (assumed appropriate for mechanical mining).

Table 24. Resources by Deposit, Mining Method and Category

Mineral Resource by Deposit, Category and Mining Method											
Deposit	Resource Category	Mining Method	Tonnes Mt	Gold g/t	Silver MOz	Copper %	Copper		Gold Equivalent		
							MOz	Mlb	g/t	MOz	
Irumafimpa	Indicated	Mechanical	0.01	11.9	0.00	2.2	0.00	0.3	0	12.4	0.00
		Hand	0.55	12.9	0.23	8.8	0.16	0.3	3	13.4	0.24
	Inferred	Mechanical	0.07	7.5	0.02	7.1	0.02	0.2	0	7.9	0.02
		Hand	0.46	11.4	0.17	9.5	0.14	0.3	3	12.0	0.18
Kora/ Eutompi	Inferred	Mechanical	3.37	7.2	0.78	33.0	3.58	2.2	163	11.0	1.19
		Hand	0.99	7.6	0.24	41.5	1.32	2.4	52	11.8	0.38
Total All Deposits	Indicated		0.56	12.8	0.23	9	0.16	0.3	4	13.4	0.24
	Inferred		4.89	7.7	1.21	32	5.06	2.0	218	11.2	1.76

M in Table is millions. Reported tonnage and grade figures have been rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. Gold equivalents are calculated as $\text{AuEq} = \text{Au g/t} + \text{Cu}\% \times 1.52 + \text{Ag g/t} \times 0.0141$.

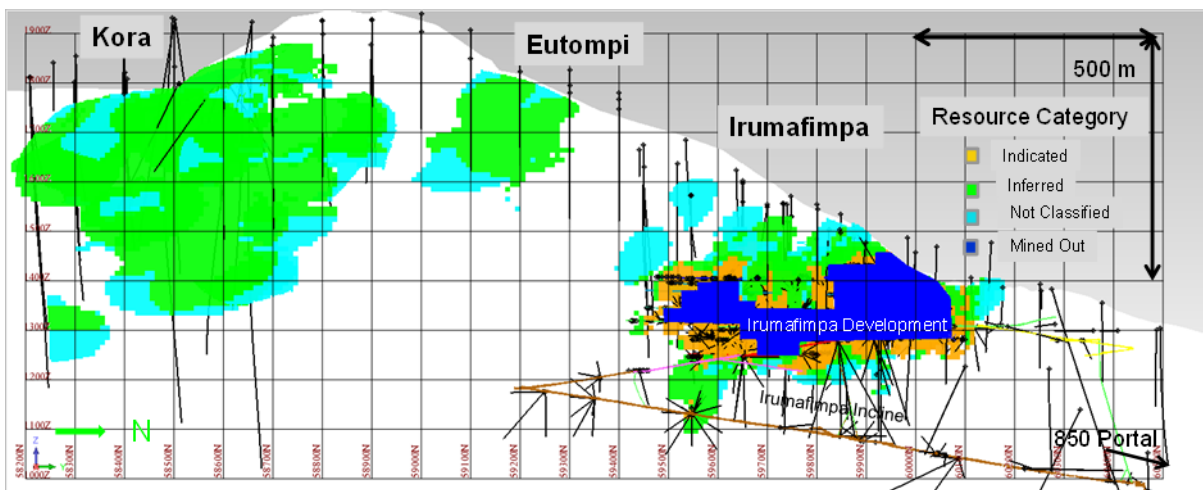


Figure 34: ML150 Long Section with blocks coloured by resource category (looking west)

In addition to passing the criteria listed above, the following definitions were adopted and applied to each domain separately;

14.14.1 Indicated Mineral Resource

- Defined as those portions of the deposit estimated with a drill spacing of 25m x 25m that demonstrates a high level of confidence in the geological continuity of the mineralization.
- Must have at least 8 informing samples

14.14.2 Inferred Mineral Resource

- Defined as those portions of the deposit with a smaller number of intersections but demonstrating a reasonable level of geological confidence.
- Must have at least 2 informing samples (i.e. drillholes).
- Maximum projection is half the drill spacing (50m).

14.15 DISCUSSION ON FACTORS POTENTIALLY AFFECTING MATERIALITY OF RESOURCES

The following factors could potentially impact on the materiality of the mineral resource estimate:

- An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
- The mineral resource is based on historical information generated by HPL and Barrick.
- Insufficient density measurements. A total of 428 measurements for Kora were reported by Barrick but most of these measurements were of waste not vein material. Densities reported by HPL for Irumafimpa were slightly higher but based on only 35 measurements.
- Potential underestimation or overestimation of gold grade due to poor core recovery in mineralized zones.
- The vein systems are structurally complex and this complexity may lead to problems with correct interpretation of vein continuity.
- A resource is an estimate of quantity and grade; the reported figures are rounded to reflect the uncertainty associated with such an approximation.
- Fluctuation in metal or commodity prices, results of additional drilling, metallurgical testing, receipt of new information and production and the evaluation of mine plans subsequent to the date of any mineral resource estimate may require revision of such an estimate.
- Nolidan has considered the Mineral Resource estimates in light of known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, and other relevant issues and has no reason to believe at this time that the Mineral Resources will be materially affected by these items.

14.16 MINERAL RESOURCE ESTIMATE STATEMENT

Mineral Resources for ML150 deposits have been classified in accordance with NI43-101 as Indicated and Inferred confidence categories on a spatial, areal and zone basis and are listed in Table 25.

Table 25: ML150 Resources by Deposit and Category

Resource by Deposit and Category										
Deposit	Resource Category	Tonnes	Gold		Silver		Copper		Gold Equivalent	
		Mt	g/t	MOz	g/t	MOz	%	Mlb	g/t	MOz
Irumafimpa	Indicated	0.56	12.8	0.23	9	0.16	0.28	37	13.4	0.24
	Inferred	0.53	10.9	0.19	9	0.16	0.27	74	11.5	0.20
Kora/Eutompi	Inferred	4.36	7.3	1.02	35	4.9	2.23	215	11.2	1.57
Total Indicated		0.56	12.8	0.23	9	0.16	0.3	4	13.4	0.24

Total Inferred	4.89	7.7	1.21	32	5.06	2.0	218	11.2	1.76
-----------------------	------	-----	------	----	------	-----	-----	------	------

*M in Table is millions. Reported tonnage and grade figures have been rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. Gold equivalents are calculated as AuEq = Au g/t + Cu%*1.52+ Ag g/t*0.0141.*

14.16.1 Notes to accompany resource statement:

1. The current sample exploration database was supplied by Barrick in MS Access format.
2. Estimation undertaken in Surpac™, using ordinary kriging (“OK”) in unfolded space.
3. The estimation block size was 10m in Y and 10m in Z with width estimated in unfolded space as a variable. Grade was interpolated by domain using OK estimation with parameters based on directional variography by domain. Thickness of the vein was also estimated by OK estimation.
4. Results validated against drill data and Inverse Distance Squared, Nearest Neighbour, Gram M Accumulation estimates and Ordinary Kriging uncapped estimates.
5. Minimum mining width of 1.2 m horizontal. Grade was diluted to account for minimum width.
6. This mineral resource estimate is based on 78,935 metres of drilling from 767 holes, and 18,312 metres of assayed intervals across all lodes. A single vein composite was used for each drill intercept on each lode – cut-off for selection was 3 m-gms Au Equivalent. There are a total of 2,003 vein composites across 19 veins, including 349 face composites.
7. A mined out area representing the extent of current mining projected across all lodes were removed from the final model as the exact location of individual stopes is not clear.
8. Top caps were applied to the composites for each vein. Grade caps were selected to restrict the influence of outliers where drilling was sparse, and varied by vein.
9. A minimum of 2 samples and maximum of 12 samples were used for each block. Search distances varied by lode and reflect the variogram ranges of 100-200 m, maximum projection beyond last drill-hole is 50 m.
10. The volume for each vein was defined by a wireframe in 3D space and is used to constrain the resource blocks.
11. Lower cut-off grades for reporting were a combination of thickness and grade reflecting mining methods, metallurgical recovery, and royalties:
 - a. Narrow Vein - Shrink Stopes - 1.2 m – 3 m thick and $\geq 6\text{g/t AuEq}$
 - b. Wide Vein – Mechanised Stopes - $>3\text{ m}$ thick and $\geq 5\text{g/t AuEq}$
12. Resource categories are based on estimation confidence and number of informing samples as a guide. Resource categories are based on estimation confidence and number of informing samples as a guide. Blocks with only one sample supporting them are not included in the resource estimate and are considered Unclassified (Figure 34, Figure 60).
13. Vein blocks in the Irumafimpa deposit have been assigned a density of 2.9 t/m³ and vein blocks in the Kora deposit have been assigned a density of 2.8 t/m³

15 MINERAL RESERVE ESTIMATES

This item is not applicable for this report.

16 MINING METHODS

During the mining operation at Irumafimpa between 2006 and 2009, mining was predominantly shrink stope with some bench (longhole) stoping. The method applied was based on the geological structure and varying vein widths. Multiple independent reviews have shown that previous operators had considerable difficulty with dilution issues during mining which has been mainly attributed to the geological complexity of the veins and a poor understanding of grade distribution within the veins.

Remedial work by K92ML on the main mine access from the 800 Portal at Irumafimpa commenced in May 2016 and access to the upper working levels is now available. Ventilation has been re-established and development to access Irumafimpa veins has commenced.

16.1 IRUMAFIMPA

16.1.1 Irumafimpa Scoping Study Inputs

In April 2016 AMDAD prepared a 3 Year Mine Plan for the Irumafimpa deposit in consultation with K92ML (AMDAD, 2016a). AMDAD anticipated that the 3 Year Mine Plan would commence effective 1 July 2016. A conceptual LOM Plan was also prepared.

As part of the mine plan AMDAD applied financial and processing parameters to determine cut off grades for stope design, generated 3-D stope shapes and mining inventory using the CAE Mineable Shape Optimiser (MSO) program, and created a conceptual development layout to suit the MSO inventory. AMDAD also produced a simple mining schedule as input for a simple project cashflow model.

Key project assumptions for determining the gold stoping cut-off grade for the Mineable Shape Optimiser (“MSO”) modelling are summarised in Table 26 below. The inputs were based on data provided by or confirmed by K92ML. Notional dilution and mining recovery factors were nominated by AMDAD after discussion and agreement with K92ML.

Table 26: Key project assumptions for Irumafimpa stope cut-off grade

PHYSICAL INPUTS	Unit	Assumption
Mining Dilution	%	35.8
Mining Recovery	%	87.8
Process Rate	ktpa	200
Process Recoveries		
Gold	%	94.0
Silver	%	50.0
FINANCIAL INPUTS (USD \$)	Unit	Assumption
Base Mining Cost	\$/t	77.79
Processing	\$/t ore	18.03
General and Administration	\$/t ore	24.61
Metal Price		
Gold	\$/oz	1200
Silver	\$/oz	15.00
Realisation Costs (Selling Costs) Payable:		
Gold	%	92
Silver	%	90
Royalty	%	2.25

AMDAD built up the base mining cost from the following unit costs provided by K92ML. These costs are based on extraction by longhole benching using the Avoca method (Table 27).

Table 27: Base mining cost allowances, \$/t, by activity

Mining Activity (USD \$)	Unit	Allowance
Fixed Operating Labour	\$/t	28.79
Plant & Equipment	\$/t	8.71
Fixed Capital	\$/t	7.84
Sub-total	\$/t	45.34

Variable Operating		
Development	\$/t	15.20
Production	\$/t	17.25
Sub-total	\$/t	32.45
TOTAL	\$/t	77.79

16.1.2 Irumafimpa Cutoff Grades

A cutoff grade was estimated using the mining, processing and economic assumptions listed above. The cutoff grade (applied in the MSO program), was estimated using a mining cost that covers all costs (downstream from establishment of the stope) which would be incurred by each potential incremental tonne of ore that could be included within the stope shape. This is the “incremental economic cutoff grade”, which will maximise the undiscounted cash value of the operation when it is applied at the point for which the downstream mining costs have been determined (Table 28).

The cutoff grade calculation also makes allowance for dilution. The applied dilution of 35.8% was based on 0.5m external dilution skins on the hangingwall and footwall of an average 1.8m stope design width, for an overall 2.8m wide stope including the dilution. A further 5% dilution is planned due to the requirement to backfill as the bench is progressed. This dilution allowance was intended as an average based on AMDAD’s understanding that the stope hangingwalls and footwalls are in variable ground conditions, from poor to moderate, and strongly influenced by the existence of sub-parallel shear zones.

Table 28: Irumafimpa Cutoff grades

Parameter	Unit	Value
Net value of Au in ore (USD \$)	\$/g	32.61
Cut-off head grade, Au	g/t	3.69
Cut-off resource grade, Au	g/t	5.20
Cut-off resource grade, Au post development *	g/t	2.00

*This is the marginal economic cutoff grade for development waste

AMDAD also assigned gold equivalent (AuEq) grades using the gold and silver grades and associated parameters. As the difference between the Au and AuEq grade was typically very small it was decided that further analysis of the resource would be undertaken using gold grades only, with silver and copper grades reported as well as gold.

16.1.3 Irumafimpa Mining Method

The 3 Year Mine Plan by AMDAD is based on longhole benching, a form of longhole open stope mining, as the main extraction method. This is similar to the method proposed for the Irumafimpa deposit by AMC Consultants Pty Ltd (“AMC”) in its 2015 mine plan. It is a selective mining method which allows extraction of high grade, yet relatively narrow, ore zones. The proposed benching method for Irumafimpa is based on drilling and blasting ore in vertical rings from drives spaced 15m apart vertically, forming stopes 18m high. Stope widths will range from 2m to 8m and strike lengths will vary, depending on ground conditions. Stopes will be extracted along strike and in a bottom-up sequence, with each stope progressively backfilled for stability and to provide a working base for the next stope above.

16.1.4 Irumafimpa Development Concept

The mine plan makes use of existing development, and in particular the existing decline, to provide access to the orebody for stope production activities. However, stope production is also dependent on excavation of the following new development:

- Cross cuts from the existing decline to levels, spaced at 15 metres vertically
- Orebody drives
- Access drives adjacent to orebodies along adjacent veins where available.
- Miscellaneous development including recesses for stockpiling, sumps, and drill cuddies.

16.1.5 Irumafimpa Backfill Strategy

Waste rock from Irumafimpa development mining will be the primary backfill material when available. Exploration development or surface waste rock will be alternative fill sources when local development waste is unavailable.

As depicted in the following figure, waste rock backfill will closely follow the bench blasting face to reduce the strike length of hanging wall and footwall left open at any one time (Figure 35). The void is filled from one end of the stope with dumped waste rock while ore is being extracted at the other end. A gap is maintained between the filling and extraction fronts to minimise dilution.

In poor ground conditions, Modified Avoca benching will be required, whereby the waste backfill is placed tight against the bench blasting face to maximise support for the stope walls.

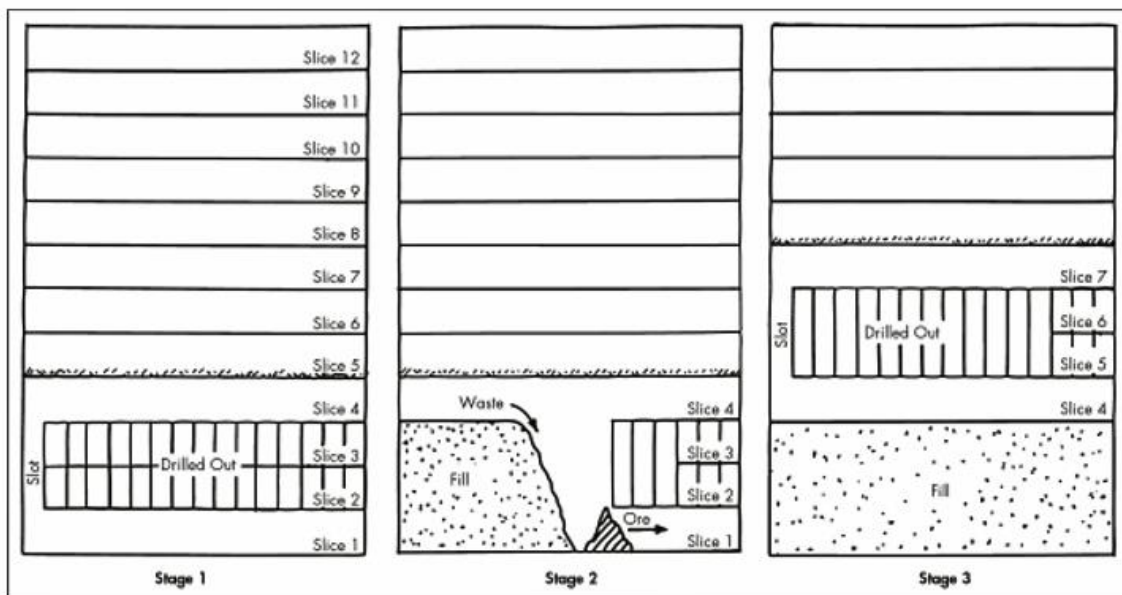


Figure 35: Avoca Benching method

(Bullock and Hustrulid 2001)

16.1.6 Irumafimpa Ventilation

AMDAD used the ventilation plan as proposed by AMC Consultants in the 2015 Irumafimpa mine plan. The key aspects of this plan are:

- Underground access at Irumafimpa is via three existing portals; a lower portal at 842RL (known as the 800 Portal) and upper portals at 1300RL and 1325RL (Puma Portal). The existing development connects between these portals. Previously the mine relied on natural ventilation and secondary ventilation for development and stoping using small fans.
- To accommodate a more mechanised mining operation, a primary extraction fan needs to be located at the Puma Portal (1325RL) with fresh air intake at the lower portal and the 1300RL portal.
- Several vent doors and barricades will be required to prevent recirculation and leakage of air.

- 20 Level (1300RL) would be the main return air way for the mine. As AMDAD has not included this level in planned mining activities during the 3 Year Mine Plan the level is available for use as a return airway.

The AMC proposed ventilation plan defines an airflow of approximately 120 m³/s, based on a requirement for 0.05 m³/s per kW of diesel power. AMDAD considers this appropriate for the proposed mine plan and its general understanding of proposed mine equipment.

AMC nominated a fan power of 150kW for the main exhaust fan at the Puma portal based on its initial mine design. AMDAD recommended an update of ventilation analysis using Ventsim or similar software once the 3 Year Mine Plan and equipment fleet are finalised and survey confirms current natural airflows. This data will then be used to confirm airflow and fan power requirements, and locations for vent doors and barricades.

16.1.7 Irumafimpa Stope Design

AMDAD used the MSO module in CAE Studio 3 for stope design. MSO automatically produces stope shapes from the resource block model that are economically optimised within specified geometrical and design constraints. For extensive multiple orebody deposits such as Irumafimpa the MSO program represents a powerful tool, generating several hundred stope shapes in a relatively short amount of time compared with manual design.

Table 29: MSO Parameters

Parameter	Value	Units
Optimisation Field	AuEq	
Default (waste) density	2.90	t/m ³
Sub-level Spacing	15	m (vertical)
Section Spacing (min)	10	m (horizontal)
Section Spacing (max)	30	m (horizontal)
Minimum Stope Width	2.0	m
Maximum Stope Width	8.0	m
Minimum Waste Pillar Width	8.0	m
Hangingwall Dilution	0.5	m
Footwall Dilution	0.5	m

MSO generated 710 individual stopes shapes ranging from 416t to 10,700t in size. The results, by vein, are summarised below in Table 30.

Table 30: Irumafimpa MSO stope shape quantities

Vein	Tonnes kt	AuEq g/t	Au g/t	Ag g/t	Cu %
R3	443	6.9	6.8	9.3	0.4
M1	264	8.3	8.3	1.9	0.1
M0	57	7.8	7.8	1.8	0.0
M3	17	6.4	6.3	10.2	0.1
M4	252	9.3	9.3	2.4	0.1
M5	462	10.9	10.9	7.4	0.1
M6	91	9.3	9.3	0.9	0.1

P2	114	7.6	7.6	2.0	0.1
Total	1700	8.8	8.7	5.4	0.2

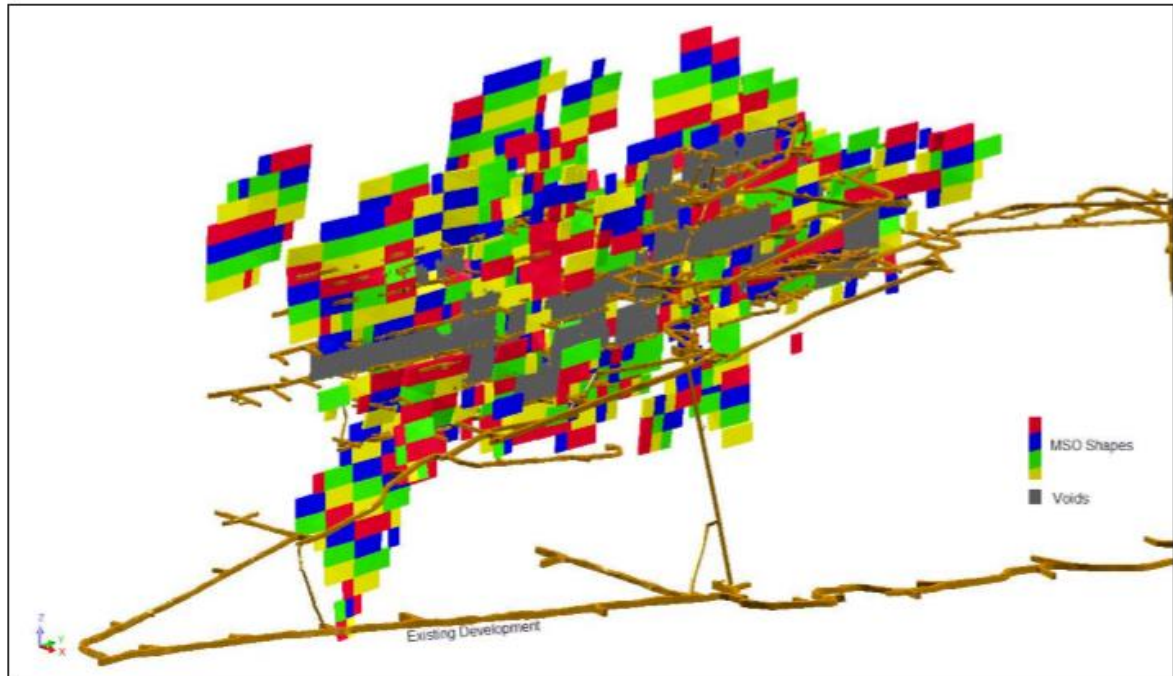


Figure 36: MSO shapes at Irumafimpa with current voids and existing development

The stope shapes created using the MSO program represent approximately 1.7 Mt of production. For the purpose of the 3 Year Mine Plan AMDAD subsequently selected a subset of these stopes, providing approximately 500,000t of production. The process to define these stopes was manual and subjective, taking into consideration the following criteria:

- Preference to stopes needing minimal vertical development (i.e. requiring limited extension of the decline above and below its current position)
- Minimizing the number of new levels needing to be developed
- Avoiding stopes where access around existing voids is problematic
- Ensuring a minimum 8m pillar along strike between stope and voids
- Targeting stopes/lifts with higher gold grades and ounces within the general sequencing constraints for the benching extraction
- Excluding stopes in locations where access and extraction was considered to be difficult due to potential stability problems associated with existing development, previously mined stopes, or proximity to the ground surface
- Consideration of infill diamond drilling and/or stope definition drilling requirements according to resource confidence level, available access for drill rigs, and drill hole coverage

The full set of stopes has a vertical extent from 1100RL to 1520RL. However, AMDAD disregarded the uppermost stopes (from 1505RL to 1520RL) due to their proximity to the ground surface.

Stopes were then excluded where existing shrinkage stoping voids prevent access. For example, Level 1310RL has extensive existing development but the existing voids appear to cut off access and often lie too close to the proposed new stopes to allow their extraction (Figure 37).

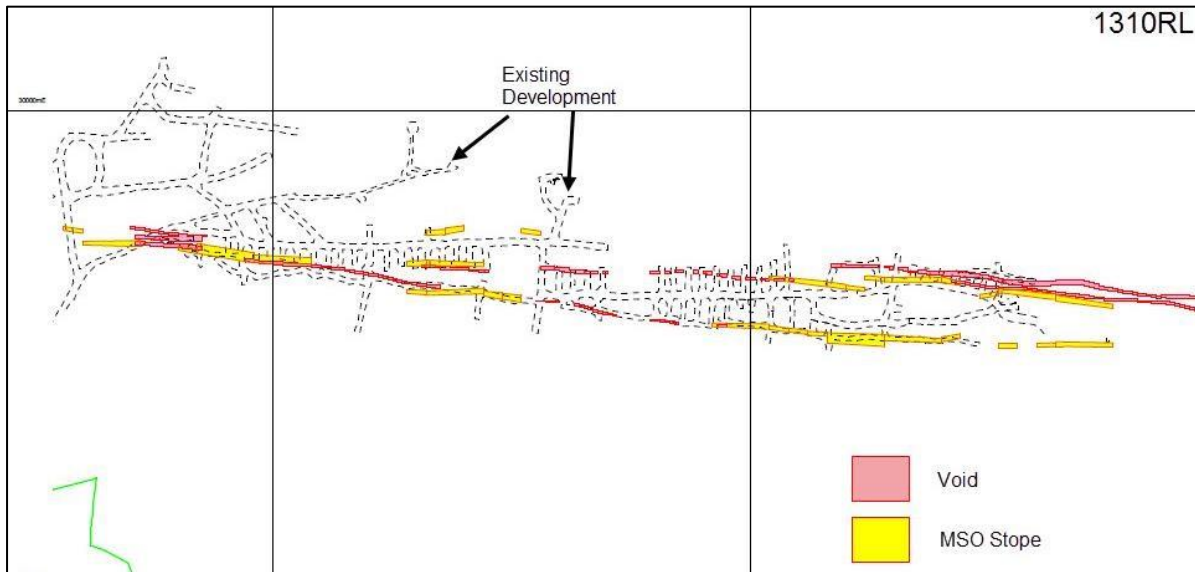


Figure 37: Level 1310RL showing voids and MSO stopes

The MSO shapes selected for the 3 Year Mine Plan are shown in Figure 38 below.

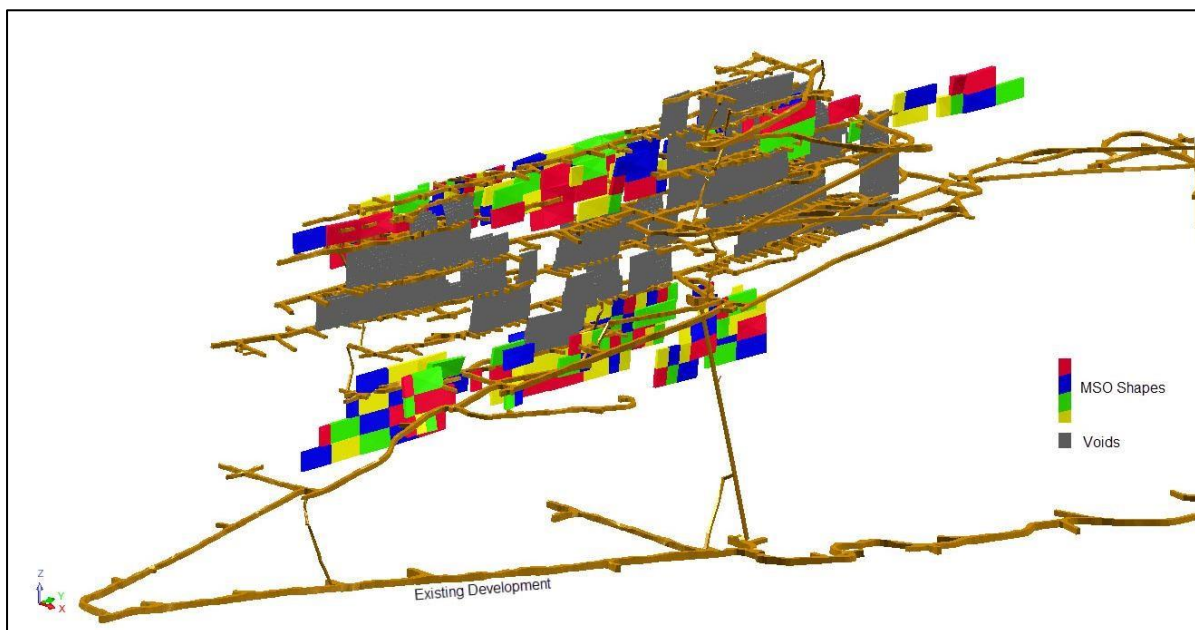


Figure 38: MSO shapes selected for the 3 Year Mine Plan schedule.

These stopes lie in two panels vertically; 1205RL to 1265RL and 1355RL to 1400RL.

These stopes generally offer efficiency of access from existing development and the lower panel should be amenable to infill diamond drilling from drill cuddies located off the existing Main Decline.

The mine design is based on assumptions regarding availability of safe access via existing development that will require verification once rehabilitation and new survey pickups are completed, and following inspections of existing stoping areas. The plan assumes that the existing ore pass system can be utilised.

The 15m level intervals and corresponding RLs were applied in the MSO process to the entire model. However, for stope lifts in the southern third of the deposit above 1370RL and 1385RL there is a mismatch between the MSO levels and the existing development, due to the gradients of the existing development that was mined southward from the level accesses. It will be necessary to redesign the

proposed orebody drives, and corresponding stope bottoms and tops to fit with the existing development in this area. In the 3 Year Mine Plan, production for these stope lifts is currently scheduled from Month 26. Development mining is scheduled to commence in Month 13 for 1370RL and Month 17 for 1385RL.

16.1.8 Irumafimpa Design Quantities

Individual stope tonnes selected for the 3 Year plan are listed in Table 31.

Table 31: Irumafimpa MSO stope shape quantities

Vein	Tonnes kt	Au g/t	Ag g/t	Cu %
R3	118	7.2	6.7	0.1
M1	97	8.4	3.0	0.1
M0	11	6.9	3.5	0.1
M3	4	6.5	15.8	0.1
M4	71	12.7	0.8	0.0
M5	101	8.4	14.0	0.2
M6	33	9.3	1.0	0.2
P2	47	7.7	1.3	0.1
Total	484	8.7	5.7	0.1

In addition to production from the selected stopes described above, there is potential to enhance early production by mining broken stocks from existing stopes or drilling and blasting stopes with existing development. This potential cannot be assessed with any confidence until the relevant areas can be re-accessed and inspected. Various records indicate that at least four stopes, listed in the table below, were in production during December 2008. The four stopes that were active when mining ceased may contain approximately 1,500t of broken stocks.

Table 32: Irumafimpa Possible active stopes

Stope	Level (mRL)	Vein	Comment
21L-17/18ShrM3	1285	M3	Commenced Dec 2008
22L-20/21ShrM5	1250	M5	Commenced Nov 2008
22L-21 StopeM4	1250	M4	Commenced Nov 2008 suspended after 2 lifts
22L-22 StopeM4	1250	M4	Mining during Dec 2008

Records also indicate that several stopes had either been developed or access to the stope developed when mining ceased in December 2008. This suggests that roughly 50,000 to 60,000 stope tonnes had been accessed when mining ceased. This potential for additional early production can only be assessed once those stope locations can be safely accessed and inspected. It is possible that deterioration in ground conditions in the intervening years has made early access to many of these tonnes physically or financially difficult.

16.1.9 Irumafimpa Mine Schedule

Due to the many complexities associated with this multi-vein and multi-level deposit AMDAD used the Minesched program to generate a detailed monthly mine schedule for the 3 Year Mine Plan.

The tonnes and grades for the MSO stope shapes were adjusted to allow for a mining recovery factor of 87.8%. Dilution of 5.0% at zero grade was added to the stope tonnes to allow for inclusion of fill material during tight firing against fill and mucking on top of fill. This fill dilution adjustment was only applied to the stope tonnes and grade and not to the development tonnes and grade. The stope tonnes reported within the MSO shapes already incorporate substantial dilution from application of the stope design parameters, including minimum width, as well as the allowance for an additional 0.5m thick dilution skin on the footwall and hangingwall.

Monthly scheduled Development and Production tonnes and Development metres are shown in the following table and charts for the 3 Year Mine Plan.

Table 33 Schedule quantities by month for Irumafimpa 3 Year Mine Plan.

Development		MONTH:	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	
Total	m		186	252	279	270	279	270	279	434	420	434	420	434	434	392	434	270	279	270	
Waste	m		177	155	144	153	91	51	54	168	158	171	213	229	126	174	162	70	147	120	
	kt		6.8	6.0	5.5	5.9	3.5	1.9	2.1	6.1	5.8	6.6	8.2	8.8	4.8	6.7	5.8	2.8	5.7	4.6	
Dev Mill Feed	m		9	97	135	117	188	219	225	266	262	263	207	205	308	218	272	200	132	150	
	kt		0.3	3.0	4.1	3.8	5.7	6.9	7.5	8.3	7.6	7.8	6.4	6.5	9.5	7.2	9.9	6.4	4.2	4.7	
	au g/t		8.76	7.27	6.49	6.06	6.84	8.85	13.72	8.43	7.58	6.54	6.75	6.08	9.35	10.10	9.16	8.94	7.57	9.02	
	ag g/t		0.32	0.77	5.18	10.44	9.17	5.18	1.83	5.41	5.14	2.91	4.48	8.87	4.61	4.14	16.23	4.43	5.62	12.45	
	cu g/t		0.02	0.05	0.12	0.19	0.14	0.09	0.04	0.10	0.06	0.05	0.05	0.13	0.12	0.10	0.21	0.12	0.17	0.17	
Stope Production																					
Mill Feed	kt		0.0	0.0	0.0	0.0	0.0	0.0	2.1	5.2	9.4	9.1	9.9	10.0	7.4	9.6	7.0	10.0	12.2	11.9	
	au g/t		0.00	0.00	0.00	0.00	0.00	0.00	6.94	10.35	9.86	11.55	8.06	5.99	6.71	8.24	17.54	6.30	7.16	8.72	
	ag g/t		0.00	0.00	0.00	0.00	0.00	0.00	14.70	7.52	6.02	5.09	6.51	4.11	4.64	7.85	0.19	7.35	7.39	2.75	
	cu g/t		0.00	0.00	0.00	0.00	0.00	0.00	0.30	0.15	0.08	0.04	0.12	0.13	0.07	0.05	0.00	0.09	0.09	0.04	
Total UG mining Inventory (Dev + Stp)																					
Mill Feed	kt		0.3	3.0	4.1	3.8	5.7	6.9	9.6	13.5	17.0	16.9	16.3	16.5	16.9	16.9	17.0	16.4	16.3	16.6	
	au g/t		8.76	7.27	6.49	6.06	6.84	8.85	12.22	9.17	8.84	9.23	7.55	6.03	8.20	9.04	12.64	7.33	7.26	8.81	
	ag g/t		0.32	0.77	5.18	10.44	9.17	5.18	4.69	6.23	5.62	4.08	5.72	5.99	4.62	6.26	9.58	6.21	6.94	5.51	
	cu g/t		0.02	0.05	0.12	0.19	0.14	0.09	0.10	0.12	0.07	0.04	0.09	0.13	0.10	0.07	0.12	0.10	0.11	0.08	
Development			MONTH:	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36
Total	m		279	279	270	279	270	279	279	252	279	173	172	70	0	0	0	0	0	0	
Waste	m		138	196	131	118	103	69	102	130	85	79	98	19	0	0	0	0	0	0	
	kt		5.3	7.5	5.0	4.5	3.9	2.5	3.9	5.0	3.3	3.0	3.7	0.7	0.0	0.0	0.0	0.0	0.0	0.0	
Dev Mill Feed	m		141	83	139	161	167	210	177	122	194	94	75	51	0	0	0	0	0	0	
	kt		4.4	2.6	4.0	4.9	5.2	6.2	5.7	4.1	5.8	3.0	2.0	1.4	0.0	0.0	0.0	0.0	0.0	0.0	
	au g/t		7.59	8.98	10.44	9.37	8.49	8.96	8.18	8.45	8.48	8.08	11.65	13.98	0.00	0.00	0.00	0.00	0.00	0.00	
	ag g/t		11.70	10.97	7.60	4.68	2.04	2.44	4.35	11.20	3.65	10.49	0.00	2.63	0.00	0.00	0.00	0.00	0.00	0.00	
	cu g/t		0.22	0.19	0.17	0.14	0.11	0.13	0.15	0.22	0.12	0.23	0.00	0.12	0.00	0.00	0.00	0.00	0.00	0.00	
Stope Production																					
Mill Feed	kt		12.2	14.0	12.4	11.5	11.7	10.7	10.8	12.8	11.0	13.6	14.5	15.1	16.4	16.4	15.9	16.0	14.1	0.0	
	au g/t		7.82	6.05	7.10	7.81	6.95	9.56	8.74	8.14	9.03	8.33	8.38	8.95	8.54	7.37	8.53	8.27	8.56	0.00	
	ag g/t		1.73	5.90	7.86	1.41	1.84	4.90	3.01	9.28	29.42	6.89	6.27	3.57	4.86	3.13	3.36	3.51	1.69	0.00	
	cu g/t		0.07	0.11	0.10	0.07	0.07	0.13	0.08	0.14	0.28	0.12	0.08	0.11	0.19	0.17	0.14	0.11	0.07	0.00	
Total UG mining Inventory (Dev + Stp)																					
Mill Feed	kt		16.6	16.5	16.4	16.4	16.9	16.9	16.4	16.9	16.8	16.6	16.5	16.5	16.4	16.4	15.9	16.0	14.1	0.0	
	au g/t		7.76	6.51	7.92	8.28	7.43	9.34	8.54	8.21	8.84	8.28	8.78	9.39	8.54	7.37	8.53	8.27	8.56	0.00	
	ag g/t		4.38	6.68	7.80	2.38	1.90	3.99	3.47	9.74	20.54	7.53	5.51	3.49	4.86	3.13	3.36	3.51	1.69	0.00	
	cu g/t		0.11	0.12	0.12	0.09	0.08	0.13	0.11	0.16	0.22	0.14	0.07	0.11	0.19	0.17	0.14	0.11	0.07	0.00	
		Total																			

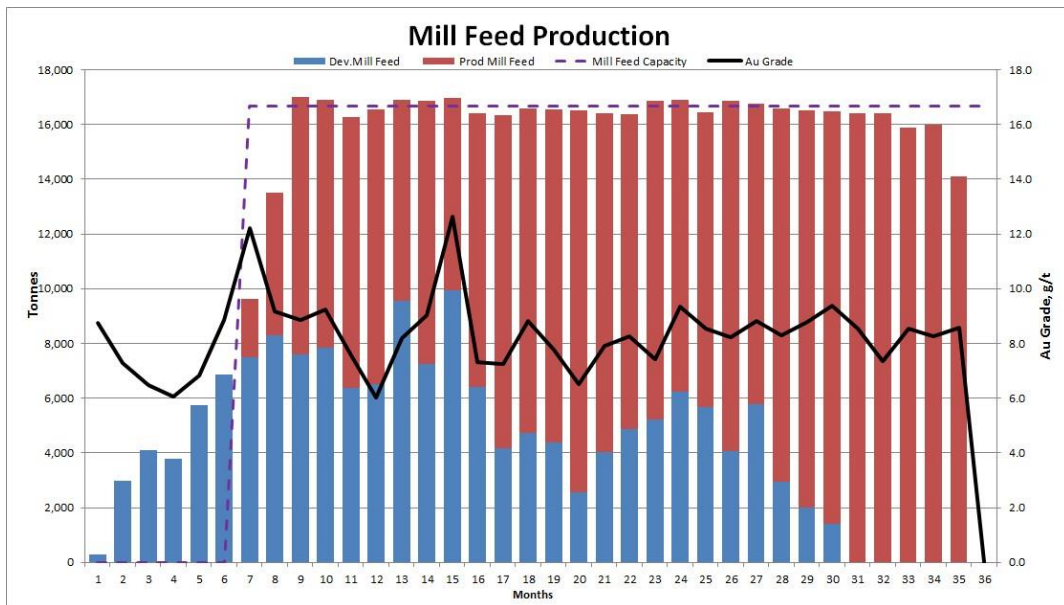


Figure 39: Monthly production from development and stopes for 3 Year Mine Plan

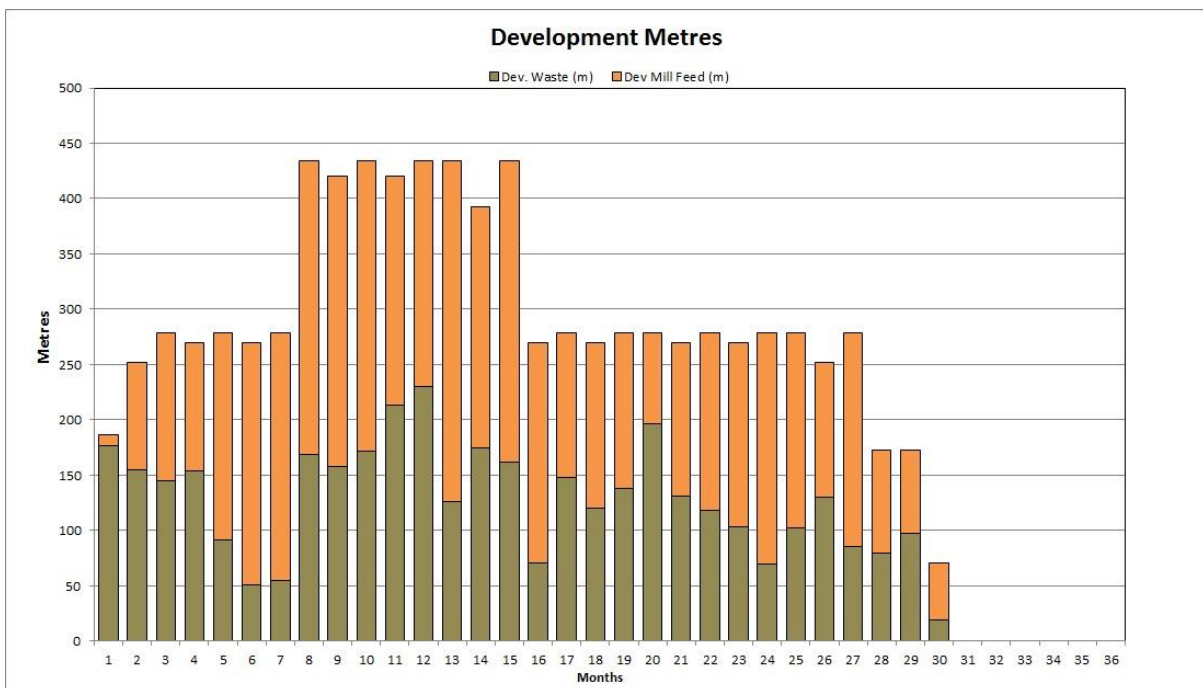


Figure 40 Monthly development metres for 3 Year Mine Plan

AMDAD also prepared a conceptual Life of Mine (LOM) Schedule made up of the 3 Year Mine Plan as described above, and including the remaining mining inventory defined by the MSO stopes generated (Table 34). AMDAD excluded some MSO stopes from the LOM Schedule according to the stope's proximity to either the ground surface or existing voids. The LOM stope layout extends from the 1100mRL Level to the 1505mRL Level. The LOM Schedule targets annual mill production of 200,000 tpa using underground production constraints of load haul capacity of 750t per day and face advance of 9m per day.

Table 34: Annual Irumafimpa LOM Schedule quantities

Development		YEAR:	1	2	3	4	5	6	7	8	Total
Total	m		3,957	3,735	2,183	2,473	3,216	3,294	1,814	0	20,672
Waste	m		1,765	1,556	870	1,032	1,566	1,962	889	0	9,639
	Kt		67	59	33	37	54	75	34	0	359
Dev Mill Feed	m		2,192	2,179	1,314	1,441	1,650	1,332	925	0	11,033
	Kt		68	69	45	55.6	63.7	51.4	35.7	0.0	389
	au g/t		7.93	9.07	7.93	6.53	6.92	7.64	8.07	0.00	7.74
	ag g/t		5.28	7.28	3.89	4.82	3.32	6.92	14.04	0.00	6.11
	cu g/t		0.09	0.15	0.11	0.11	0.17	0.45	0.61	0.00	0.21
Stope Production											
Mill Feed	Kt		46	130	169	149	162	172	149	37	1,015
	au g/t		8.88	8.02	8.22	7.86	7.85	8.61	9.10	9.22	8.35
	ag g/t		6.10	4.58	6.31	2.05	6.36	3.00	8.73	20.28	5.76
	cu g/t		0.11	0.08	0.13	0.05	0.12	0.17	0.40	0.87	0.18
Total UG mining Inventory (Dev + Stp)											
Mill Feed	Kt		114	200	214	205	226	224	185	37	1,403
	au g/t		8.31	8.39	8.16	7.50	7.59	8.39	8.91	9.22	8.18
	ag g/t		5.61	5.52	5.80	2.80	5.50	3.90	9.75	20.28	5.85
	cu g/t		0.10	0.10	0.13	0.07	0.14	0.23	0.44	0.87	0.19

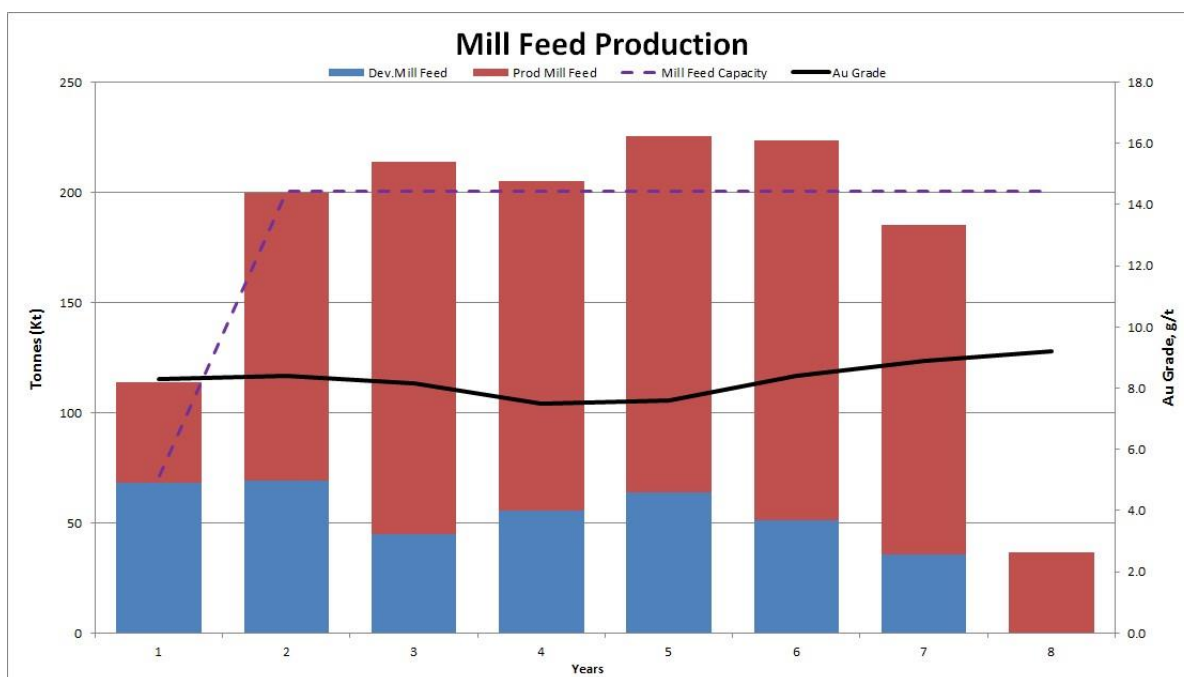


Figure 41 Annual LOM Irumafimpa production from development and stopes

16.2 KORA

16.2.1 Kora Scoping Study Inputs

In September 2016 AMDAD prepared a conceptual Mine Plan as part of a Scoping Study for the Kora deposit in consultation with K92ML (AMDDAD, 2016b).

As part of the Scoping Study AMDAD applied financial and processing parameters to determine cut off grades for stope design, generated 3-D stope shapes and mining inventory using the CAE Mineable Shape Optimiser (MSO) program, and created a conceptual development layout to suit the MSO

inventory. AMDAD also produced a simple mining schedule as input for a simple project cashflow model.

The conceptual mine plan prepared by AMDAD makes use of the two proposed exploration inclines to be mined to the south from Irumafimpa. The mine plan assumes that these drives have been completed and are a “sunk” cost. The mine plan also does not incorporate the proposed exploration drilling that will be undertaken from these exploration inclines.

Key project assumptions for determining the gold equivalent stoping cutoff grade for the MSO modelling are summarised below. The inputs were based on data provided by or confirmed by K92ML. Notional dilution and mining recovery factors were nominated by AMDAD after discussion and agreement with K92ML.

Table 35: Key project assumptions for Kora stope cutoff grade

PHYSICAL INPUTS	Unit	Assumption
Mining Dilution	%	22.4
Mining Recovery	%	90.0
Process Rate	ktpa	400
Process Recoveries		
Gold	%	91.5
Copper	%	91.5
Silver	%	90.0
FINANCIAL INPUTS (USD \$)	Unit	Assumption
Base Mining Cost	\$/t	87.7
Processing	\$/t ore	17.7
General and Administration	\$/t ore	30.0
Metal Price Gold	\$/oz	1300
Copper	\$/t	4800
Silver	\$/oz	18
Realisation Costs (Selling Costs) Payable:		
Gold	%	97.7
Copper	%	97.0
Silver	%	90.0
Total concentrate costs	\$/dmt	270
Royalty	%	2.25

It should be noted that the total operating costs shown in Table 35 were revised from the previous mine planning work that AMDAD completed for the Irumafimpa mining study in April 2016 due to:

- Addition of an allowance for cemented backfill
- Removal of the sunk cost of existing plant and equipment, and
- Revision of the G&A cost (as advised by K92ML).

Whereas the parameters above were adopted to determine the cutoff grade for the MSO program, the final economic analysis of the Kora deposit described in Section 22 Economic Analysis incorporates a further revision to Processing and G&A costs, as advised to AMDAD by K92ML.

16.2.2 Kora Cutoff Grades

A cutoff grade was estimated using the mining, processing and economic assumptions listed in Table 35. This cutoff grade (applied in the MSO program) was estimated using a mining cost that covers all costs (downstream from establishment of the stope) which would be incurred by each potential incremental tonne of ore that could be included within the stope shape. This is the “incremental economic cutoff grade” which will maximise the undiscounted cash value of the operation when it is applied at the point for which the downstream mining costs have been determined (Table 36).

The cutoff grade calculation also makes allowance for dilution. The applied dilution of 22% was based on the same 0.5m allowance for falloff on the stope hangingwall and footwall, as used for the Irumafimpa mine planning work. This equates to 20% dilution for an average 5m stope design width, resulting in an overall 6m wide stope including the dilution. A further 2% dilution was added to account for fall off of cemented fill and loading out on backfilled floors.

Table 36: Kora AuEq Cutoff grades

Parameter	Unit	Value
Net value of Au in ore (USD)	\$/g	36.52
Cut-off head grade, AuEq	g/t	3.71
Cut-off resource grade, AuEq	g/t	4.52
Au equivalent factor for Ag		0.0125
Au equivalent factor for Cu		0.8959

AMDAD assigned gold equivalent (AuEq) grades using the gold, copper and silver grades and associated parameters.

16.2.3 Kora Mining Method

The Scoping Study for Kora is based on open stoping by longhole benching, similar to the method proposed by AMDAD for the Irumafimpa deposit. This is a relatively selective mining method which allows extraction of high-grade, yet relatively narrow, ore zones. Production drilling and blasting of ore would be undertaken with vertical rings drilled from drives spaced 15m apart vertically, forming stopes 19m high. Stope widths will range from 2m to 8m and strike lengths will vary depending on ground conditions. Stopes will be extracted along strike and in a bottom-up sequence, with each stope progressively backfilled for stability and to provide a working base for the next stope above. The use of cemented fill will maximise recovery of the high grade ore.

AMDAD also prepared a mine plan for 25m level intervals to compare against the 15m level interval base case.

No geotechnical information is available for the Kora deposit at present, and AMDAD adopted dilution and recovery assumptions from the Irumafimpa mine plan. AMDAD concluded that longhole benching is likely to be an appropriate extraction method for Kora, provided that stope wall stability can be achieved by:

- Limiting stope strike span
- Use of cemented backfill for increased recovery and wall stability
- Use of cable bolting where required
- Attention to sub-parallel shears in stope walls
- Taking care not to undercut stope walls
- Careful management of drill and blast practices

16.2.4 Kora Development Concept

The conceptual mine plan for the Kora deposit proposed by AMDAD incorporates the existing decline development at Irumafimpa and two proposed exploration inclines that K92ML is planning to develop. The Kora exploration inclines would be the main access and, initially, the intake air sources. From the Kora exploration incline, the Kora mine plan consists of:

- A centrally located incline commencing from both Kora exploration inclines at 1270mRL and 1510mRL, and mined to the top of the resource at 1825mRL.
- Access development to the orebody at 15m level intervals.
- Footwall drives extending to the northern and southern ends of the deposit, providing flexibility of access for multiple concurrent ore sources on each level.
- Orebody drives at the base of each stope on each level
- Loading and stockpile bay development on each level.

Ventilation requirements would include a central fresh air rise (FAR), collared at surface, which would also provide a secondary means of egress to the surface. Fresh air would be supplied via the lower Kora exploration incline and the proposed FAR. Two return air rise systems, would need to be established between levels using the longhole rise method (LHR). These will remove exhaust air from Kora via the upper Kora exploration incline, or alternatively via an exhaust adit or rise collar at Kora.

AMDAD propose that two ore passes be established up through the Kora deposit, with the base and loading point at 1270mRL. All ore would be loaded into trucks at the base of the ore passes and hauled down to the Irumafimpa portal. Haulage distance is approximately 3.1 km from the 1270mRL loading area to the Portal. The Kora development concept is shown schematically in longitudinal section in Figure 42 and Figure 43 below.

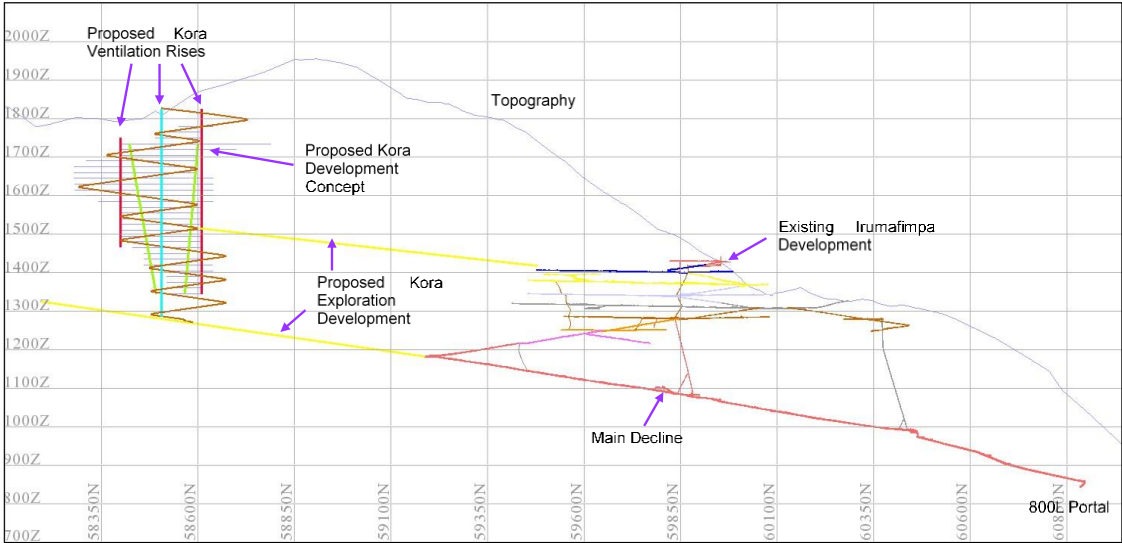


Figure 42: Longitudinal projection looking west showing proposed Kora development in relation to existing Irumafimpa development.

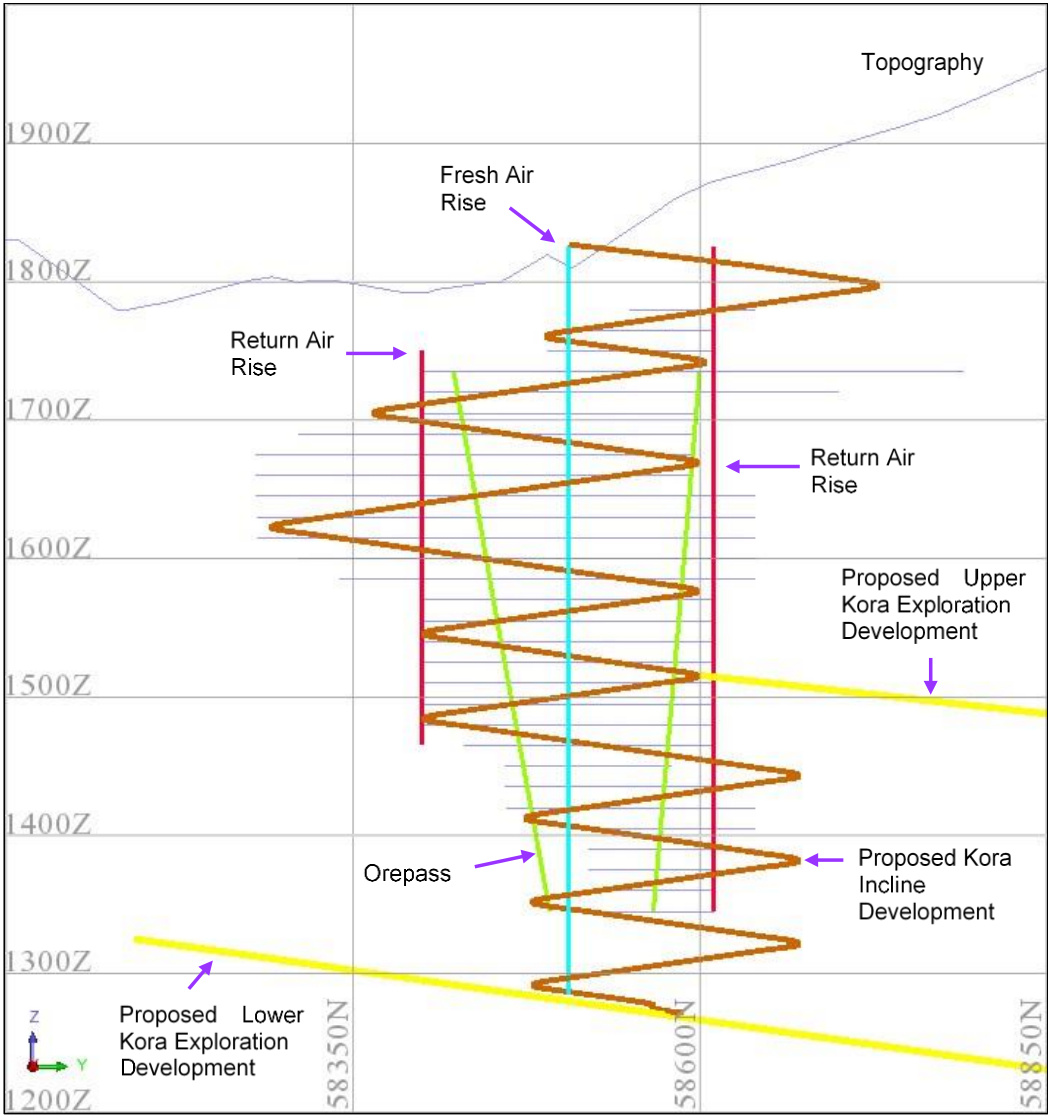


Figure 43: Kora development concept, longitudinal projection looking west

16.2.5 Kora Backfill Strategy

Waste rock from development mining will be the main source of backfill material. External sources such as quarried surface waste rock will be the alternative fill source when local development waste is unavailable. AMDAD has assumed use of cemented backfill where appropriate to increase the mining recovery of the Kora resource. Cemented backfill will reduce the requirement for in-situ rib pillars and regional pillars.

A second stoping panel will be required to increase the Kora production capacity. AMDAD has proposed the 1645mRL level as the base of this second panel. The backfill in stopes on 1645mRL will require a higher cement content compared to other benches so as to enable the mining of stopes immediately below on 1630mRL.

Analysis of cement content will be required for different stoping areas and a materials balance of waste rock mined to determine the quantity of external rock required as backfill. A simple cement addition system is proposed in which haul trucks loaded with waste rock would reverse underneath a cement slurry tank or hopper that would supply the required dosage of cement. Issues that will need to be resolved for such a system include tipping arrangements at the stope, stope drive height required to accommodate truck tipping, and homogeneity of cemented rock fill mix.

16.2.6 Kora Ventilation

The ventilation system for Kora will initially use the existing infrastructure of the Irumafimpa development until the Kora primary ventilation system is established.

The proposed Kora exploration incline from the 1180mRL “switchback” corner of the existing Irumafimpa decline, will have forced ventilation via a low-friction vent duct along its length. Development fan(s) will be installed in the existing Irumafimpa decline to ventilate the Kora exploration incline as it is mined to the south. Additional forced ventilation is required in the upper exploration incline to facilitate mining of the Kora incline from 1510mRL upwards (Figure 44).

The proposed start of the Kora operations incline at 1250mRL is approximately 615m from the 1180mRL “switchback” and initially the incline will use this forced ventilation system. Mining of the upper Kora operations incline at 1510mRL is approximately 890m from existing Irumafimpa development at the 1415mRL. The Kora operations incline development may require upgrade of the fan and vent duct in use in both Kora exploration inclines.

A raise bored rise or Alimak rise would need to be developed at Kora between 1250mRL and 1510mRL to establish the primary ventilation system (i.e. intake air along the lower exploration incline and return air out the upper exploration incline). An additional vent rise will need to be developed as soon as practical from the surface to the base of the Kora operations incline, approximately 540m in length. This would also provide a secondary means of egress from the mine. All development and benching at Kora will be able to be serviced by this primary system.

Fresh air would enter Kora via the lower Kora exploration incline and FAR and be directed into level development by forced ventilation with secondary fans and ventilation ducting. Exhaust air would exit level development into the Northern and Southern RARs, and exhaust air would exit Kora via the upper Kora exploration incline or alternative exhaust routes to the surface.

AMDAD believes that it may be necessary to strip the existing Irumafimpa 1415 Level to a larger profile for the primary vent system at Kora to operate with an effective pressure and airflow.

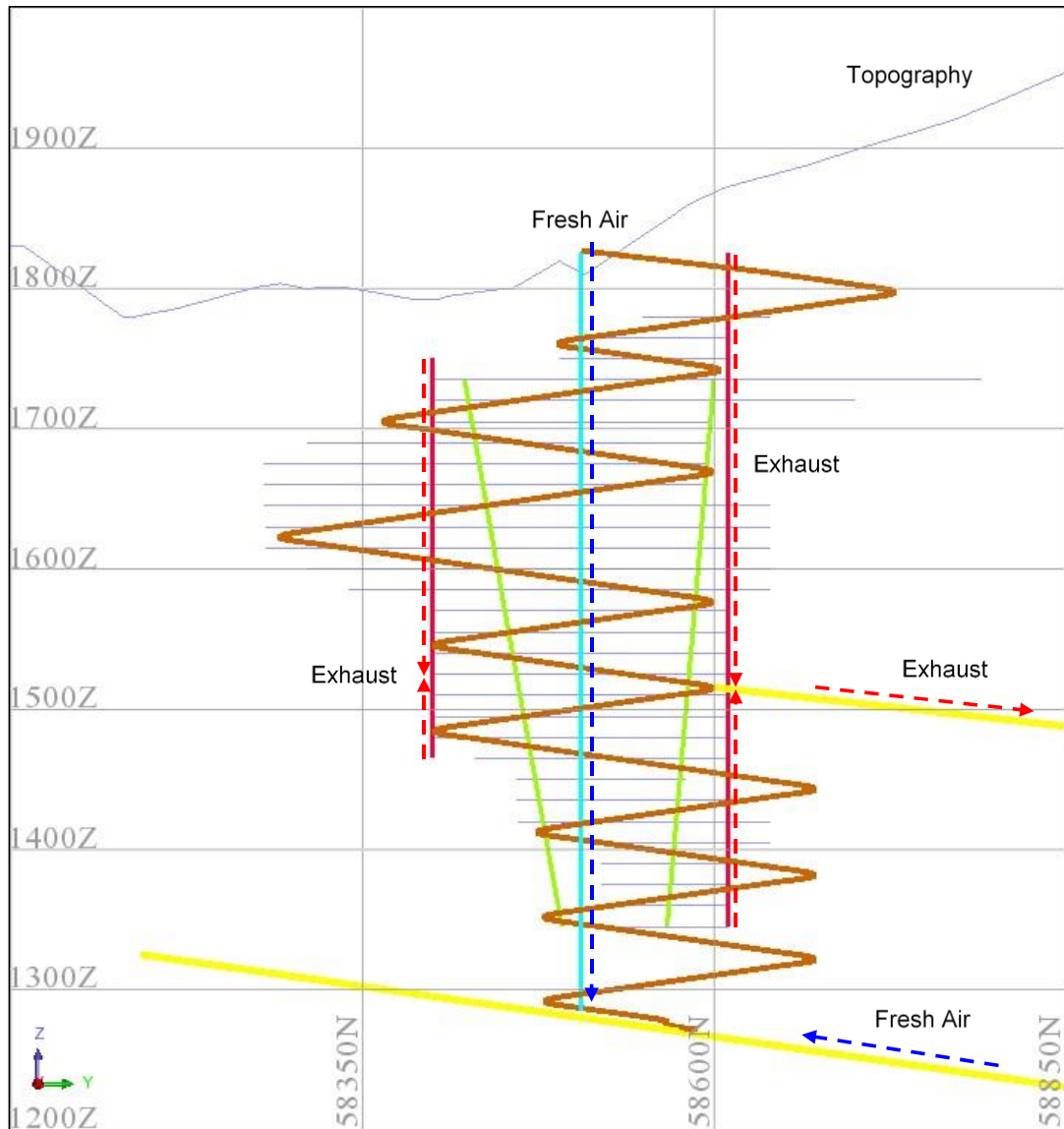


Figure 44: Ventilation concept for Kora development

16.2.7 Kora Stope Design

AMDAD used the MSO module in CAE Studio 3 for stope design. MSO automatically produces stope shapes from the resource block model that are economically optimised within specified geometrical and design constraints. For extensive multiple orebody deposits such as Kora the MSO program represents a powerful tool, generating several hundred stope shapes in a relatively short amount of time compared with manual design.

The stope parameters applied in the MSO modelling are tabulated below.

Table 37: MSO Parameters

Parameter	Units	Value
Optimization Field		AuEq
Cut off grade	g/t	4.5
Default density	t/m ³	2.80

Level Spacing	m (vertical)	15 (base case) 25 (alt case)
Section Spacing	m (horizontal)	30
Minimum Stope Width	m	2.0
Maximum Stope Width	m	8.0
Minimum Waste Pillar Width	m	8.0
Hanging Wall Dilution	m	0.5
Footwall Dilution	m	0.5

The MSO generated 556 individual stope shapes for the 15m level case ranging from 3,755t to 6,822t in size. The results, by vein, are summarised below.

Table 38: MSO stope shape quantities – 15m level case

Vein	Tonnes kt	AuEq g/t	Au g/t	Ag g/t	Cu %
K1	2,298	9.2	7.1	26	2.0
K2	883	9.8	8.3	26	1.2
K5	9	5.3	2.2	53	2.7
J1	169	6.5	5.9	13	0.5
E4	145	6.2	4.6	22	1.5
Total	3,504	9.1	7.2	25	1.7

The MSO generated 318 individual stopes shapes for the 25m level case ranging from 6,250t to 19,100t in size. The results, by vein, are summarised below.

Table 39: Kora MSO stope shape quantities – 25m level case

Vein	Tonnes kt	AuEq g/t	Au g/t	Ag g/t	Cu %
K1	2,285	9.1	7.1	25	1.9
K2	868	9.6	8.2	26	1.2
K5	13	5.5	3.4	44	1.6
J1	150	6.5	6.0	13	0.4
E4	133	6.3	4.6	24	1.6
Total	3,449	9.0	7.2	25	1.7

Figure 45 below shows the MSO stope shapes for the 15m level MSO run and Figure 46 shows a plan view of stope shapes and mineralized veins.

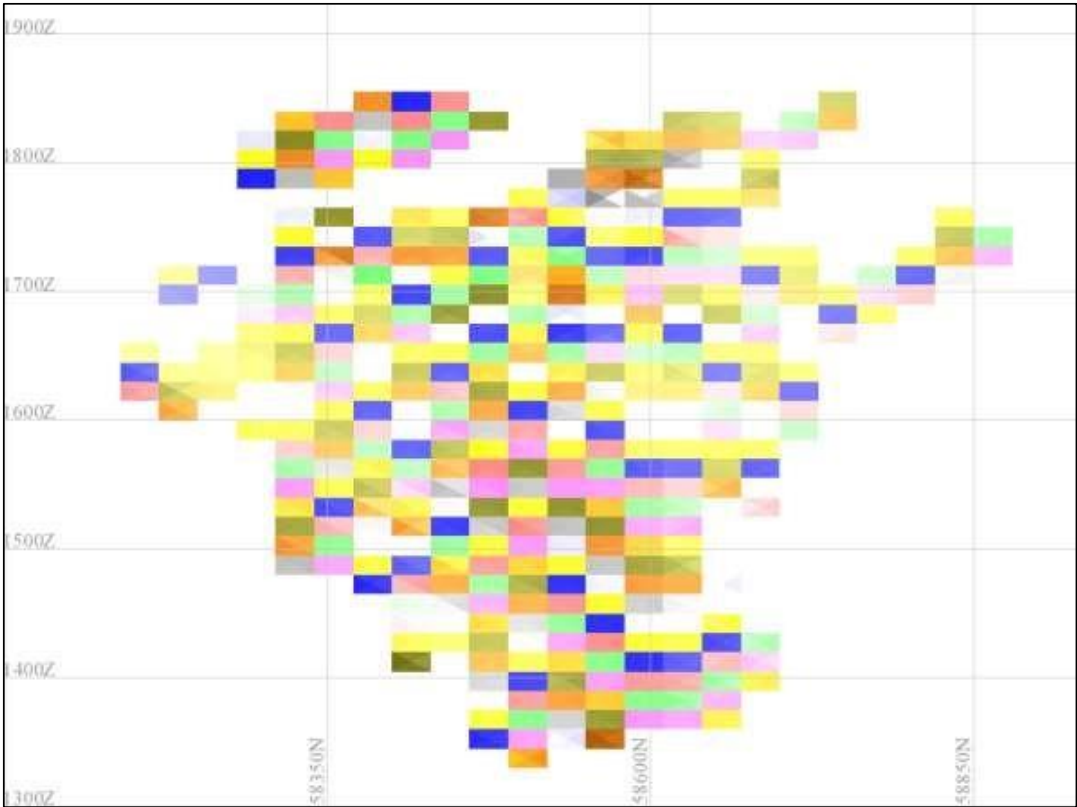


Figure 45: Kora MSO shapes, 15m levels (longitudinal projection looking west)

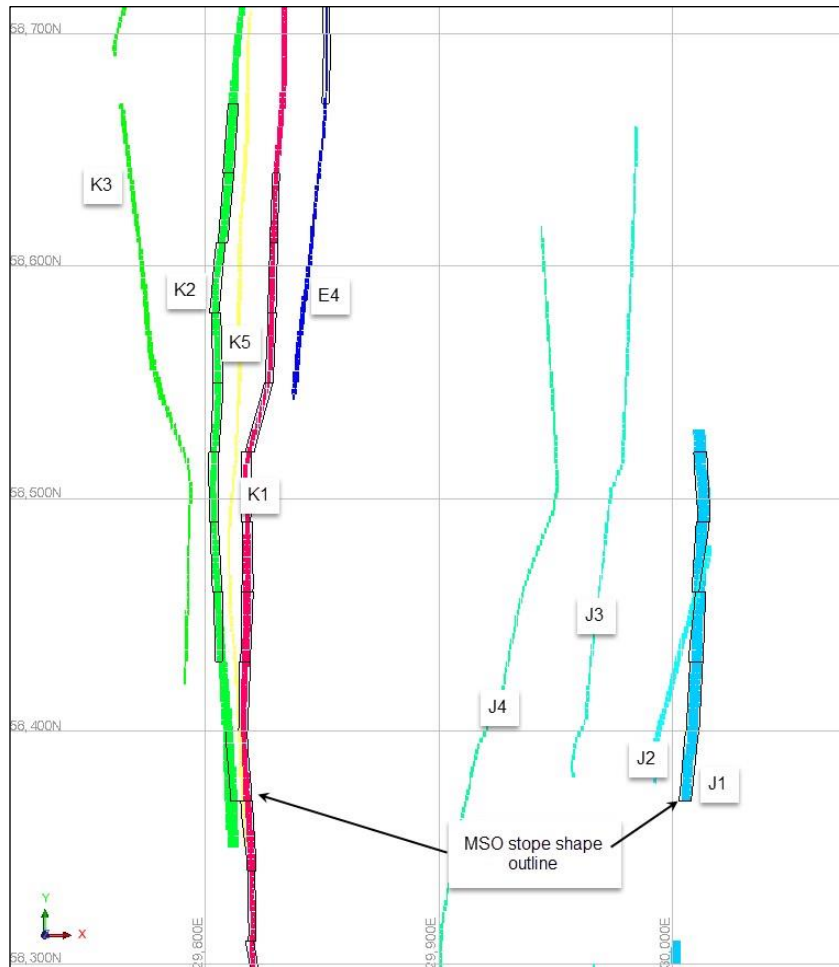


Figure 46: Kora MSO stope shapes with mineralised veins, typical plan view at 1700mRL

16.2.8 Mine Design Quantities

The stope shapes created using the MSO program for both the 15m and 25 level cases represent approximately 3.2Mt of mill feed, after mining recovery is applied. The access for these stopes will be from the base up, with a centralised incline developed from the Kora exploration drive spiralling upwards to the top of the Kora deposit, as described in Section 16.2.8.

AMDAD created a simple centreline development design in Surpac for the 15m level case only and generated development quantities.

Table 40: Development quantities for 15m levels

Development type	Units	Quantity
Kora Incline	m	4,212
Stockpile Bays	m	720
Level Access	m	3,567
Loading Bays/sumps/Cuddies	m	1,650
Footwall drive	m	6,463

Orebody access	m	1,627
Orebody drive	m	15,904
FAR access	m	80
FAR rise	m	540
RAR access	m	1,334
RAR rise	m	750
Orepass access	m	642
Orepass rise	m	788
Total Lateral Development	m	36,108
Total Vertical Development	m	2,078

16.2.9 Kora Mine Schedule

The schedules prepared by AMDAD for the Kora stoping and development concepts used the following parameters.

Table 41: Schedule Parameters

Parameter	Value	Units
Incline Development	150	metres adv. /month
Maximum Development	800	metres adv. /month
Raise Construction		
Site Preparation and Rig Setup	60	Days
Drill Pilot Hole	10	m/day
Raise Reaming	3	m/day
Stoping Rates – applied to upper and lower horizons		
First Quarter	20	Kt/qtr
Second Quarter	40	Kt/qtr
Maximum Production Rate	60	Kt/qtr

The maximum development rate assumes that two additional development jumbos would be mobilised in addition to the existing K92 Kora Incline jumbo to complete waste development early in the project, with each jumbo boring three faces per day at 85% availability.

The schedule prepared for the Scoping Study does not include the pre-development exploration period which involves the establishment of the two exploration inclines from Irumafimpa.

Year 1 will involve:-

- Development of the lower and upper sections of the Kora operations incline from the two exploration inclines

- Establishment of a ventilation circuit following the completion of the FAR and establishment of the Northern and Southern RAR using between sub-levels
- Establishment of initial stoping horizons in the lower levels above 1345RL and upper levels, above 1615RL

Year 2 will involve:-

- Continued development of the Kora operation incline
- Completion of the ventilation circuit as the lower portion of the Kora operations incline connects to the upper incline, with exhaust via the upper exploration incline
- Stopping rates increasing up to 100,000t per quarter

Production rates of approximately 50,000t per quarter are scheduled on both the upper and lower stopping horizons. Steady production is scheduled thereafter until production ends in Year 10.

Development is scheduled to establish ventilation early in the project, with the bulk of waste development completed by the end of Year 3. Ore development then continues with stopes established on each level as the upper and lower stopping horizons advance upwards. Future evaluation work will optimise development sequencing by delaying waste development, and deferring waste development costs wherever possible.

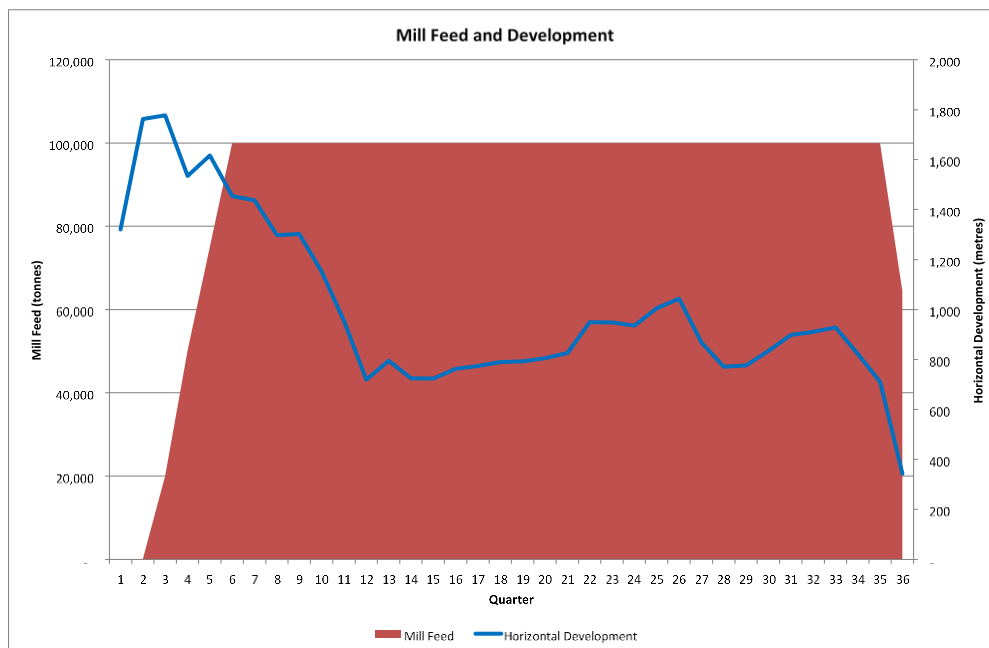


Figure 47: Mill feed tonnes and development metres

Production from stopes commences in Quarter 3. Production on the upper and lower stopping horizons gradually increases until the target production rate of 400,000 tpa is achieved in Quarter 7.

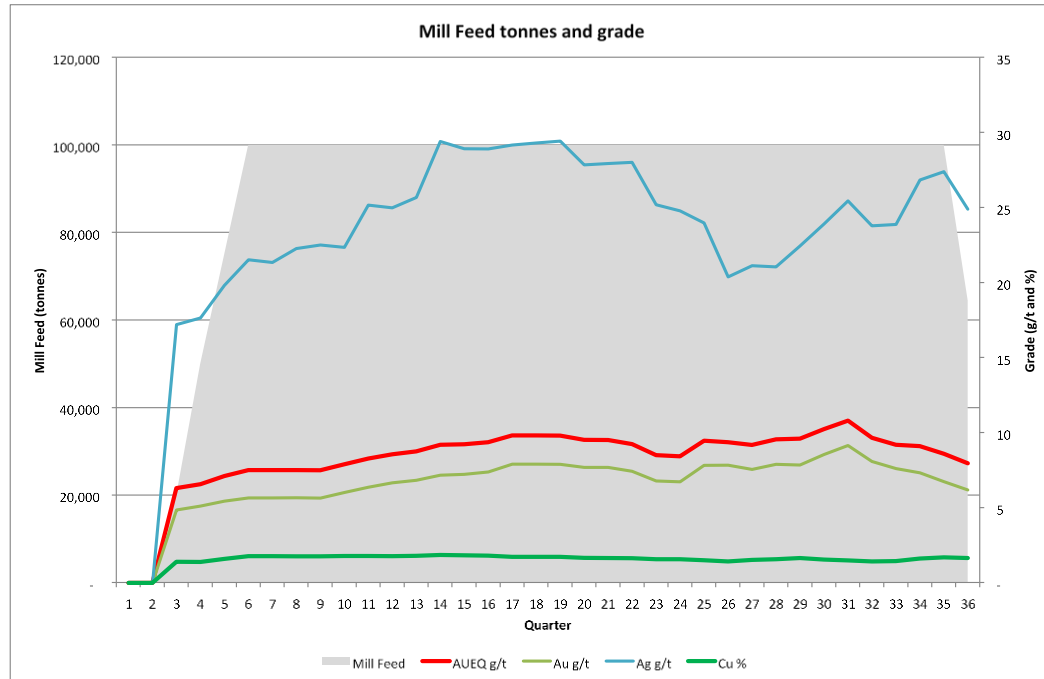


Figure 48: Mill feed tonnes and grade

17 RECOVERY METHODS

17.1 INTRODUCTION

The Kainantu processing plant is located approximately 7km from the opening of the 800 portal which accesses the Irumafimpa Mine. Simple processing technology was used to treat Irumafimpa ore. Following crushing, screening and grinding the sulphide bearing material was separated from the non-mineralized host rock by flotation. The design throughput of the plant was 21 tonnes per hour (170,000tpa) and approximately 10% of the ore was recovered as a high-grade gold bearing flotation concentrate with the waste material pumped to an engineered tailings storage facility. The gold bearing concentrate was packed in containers and trucked to Lae from where it was shipped to a smelter/refinery for the recovery of the gold.

The plant was designed and constructed in 2005 and treated ore from the Irumafimpa lodes over two separate periods between 2006 and 2008 (HPL and Barrick). Concentrate from the Kainantu Mine was sold to a number of smelters including Japanese smelters. The specification generally fell into that acceptable to copper smelters seeking high gold and high sulphur feedstock, although it did not contain significant copper.

The Process Plant consisted of the following unit processes:

- Ore Receiving and Crushing; to reduce the ROM sizing prior to reclamation for grinding. Screening and recycling was found to be problematical in previous operation and may be removed and replaced with a more suitable crusher operating in open circuit.
- Grinding and Classification; in which the crushed ore is reclaimed and ground to the required size for flotation.
- Differential Flotation; commencing with an Outokumpu Skimair Flash Flotation unit in the classification circuit, combined with Outokumpu tank cells treating grinding product, to recover a gold bearing sulphide concentrate for export.
- Flotation tailings deposition in the tailings storage facility.

- On-site reagent storage and mixing facilities.
- Services for plant air and water distribution.



Figure 50: Photograph of the processing plant (2008)



Figure 51: Photograph of the Crushing Circuit (2012)

17.2 CURRENT PLANT CONDITION

The processing plant has been idle, under care and maintenance, since processing ceased in December 2008.

Following completion of studies on the redesign of the crushing circuit to handle wet clay rich mill feed Mincore Pty Ltd (“Mincore”) was engaged by K92ML to undertake the plant refurbishment and the installation of a drum scrubber (Mincore, 2015). Refurbishment and repair of the process plant by Mincore commenced in March 2016.

A new main 415V switchboard has been installed as part of the refurbishment of the plant. In addition to general rehabilitation of existing equipment the process plant was enhanced by the addition of a Drum Scrubber which was commissioned in October 2016. The assay laboratory has been recommissioned under the control of Intertek, an internationally recognized assaying company. Equipment to allow on-site fire assaying of mine samples is currently being installed.

17.3 PLANT UPGRADE SCOPING STUDY

In August 2016 Mincore completed a scoping study on the requirements for an upgrade of the existing plant to allow treatment of Kora ore at a proposed rate of 400,000 tpa (Mincore, 2016). Mincore concluded that:

- The crushing and milling (comminution) power in the current plant to grind 50tph to the required grind size of P_{80} of 106 μm will require further investigation in the next stage, as it is

limited in the grinding circuit. The current two stage crushing circuit is rated at 68tph producing a product size P_{80} of 10-12mm.

- Additional flotation capacity is required to achieve acceptable residence times for each cell. There is sufficient space to install additional cells if future testwork identifies a requirement for longer residence time.
- The existing concentrate thickener and filter is adequate for 400,000tpa of Kora feed averaging 1.7% copper.
- The existing tailings line is adequate but a full pump upgrade will be required.

Mincore suggested the following circuit modifications:

- Liner wear should be optimized by installation of a twin deck grizzly cassette in the ROM bin feeder to bypass -50mm fines from the jaw crusher. Additionally a standard short head concave profile is needed in the secondary crusher to optimize available crushing area and product size distribution in the 10-15mm range.
- The grinding mill may be power limited based on the expansion criteria. Mincore believe that by increasing the media charge (possibly with the addition of a trunnion insert or discharge grate to retain the charge) and a larger motor the ball mill will draw full installed power and achieve P_{80} 106 μ m grind size.
- The existing 150CVX cyclones will have to be upgraded to meet the higher feed rate and recirculating load. For optimum feed into the unit cell (flash flotation) the cyclone tower will need to be raised and a distribution box installed.
- A gravity concentrator is required in the milling circuit for recovery of any gravity gold. The gravity concentrate would be treated using a Gemini table and direct smelted in an electric furnace.
- Reconfiguration of the current flotation circuit is based on the preliminary test results generated by Barrick Gold which indicated that the Kora ore can produce a single high copper with gold concentrate. The existing rougher and cleaner circuits are proposed to be used for the new rougher duty since the current cleaner circuit comprises four cells which are exact duplicate of the four cells which make up the current rougher circuit. This will effectively double the capacity and provide satisfactory retention time and requires minimal modification. A new bank of Cleaner and Recleaner cells will need to be added and connected to the existing concentrate thickener and filter.



Figure 52: Upgraded Process Plant Flowsheet

Source: Mincore (2016)

Mincore consider that testwork is warranted to address:

- Increasing grinding capacity by reduction of the feed size to the ball mill should be investigated. This could be accomplished by creation of a midsize stockpile by removing product at the scrubber situated below the crusher discharge size. This ore would be fed to the belt after the Fine Ore Bin but before the Weightometer.
- Further work to establish the most suitable grind size to achieve balanced liberation, recovery, concentrate grade and production rate.
- Investigation of reagent type, addition rate and addition point, including use of new reagents (all at variable pH).
- Investigation of ore blending procedures by use of laboratory flotation tests at various grind sizes to facilitate processing in the grinding, flotation and filtering circuits.
- Variability testwork to confirm comminution and flotation parameters.
- Barrick 2009 test work suggests that rougher mass pull balancing with cleaner cells will require consideration as a matter of urgency, particularly for high copper feed. Mincore suggest configuration of the 3rd and 4th Rougher Cells as Scavenger cells, and investigating the requirement for regrind of the rougher, and/or rougher scav concentrates. Mincore also suggest investigating opening and closed circuiting the cleaner tails, and undertaking locked cycle testwork to confirm performance.
- Flash flotation optimization including forwarding of concentrate from the Concentrate Thickener and Cleaners. For flotation design it is assumed that this is off line.
- Investigate reducing reagent consumption, particularly lime and caustic soda, by installation of a tailings thickener. Pumping power costs, water consumption and time would also be saved.
- Investigate the effect of acid mine water on alkali consumption, suspended solids, and TDS precipitation (including copper and gold).

18 PROJECT INFRASTRUCTURE

The Kainantu mine is located within ML150 and the main Kainantu exploration camp and processing plant are located within LMP78 which is located within EL693. The Property includes all mine infrastructure, exploration camps, exploration data and diamond drill core.

The property is well supported by regional infrastructure, and contains all the necessary site infrastructure for mining operations. The following descriptions are summarised from Barrick (2014).

18.1 POWER AND WATER

18.1.1 Power

Power is supplied to the Property from two sources. The primary source is the PNG Power national grid from the Ramu sub-station, located 20 km from the processing plant site. The electrical energy for Kainantu operations is delivered by PNG Power from the nearby Yonki Dam Hydroelectric Plant. In early 2010, back-up power capacity was reduced to one 530 kVa containerised 415V generator at the plant site. Power from the national grid services both the plant area and is available up to the lower portal of the underground mine. Power reticulation is 11kV.

Current back-up power is supplied by a 530 kVa generator at the 800RL Mine Portal, a 600 kVa generator for the Treatment Plant and Offices and a 200 kVa generator for the accommodation camp.

K92ML are planning to install 4MW of stand-by diesel generated power which will be sufficient to run the mine and treatment plant.

18.1.2 Water

Water for potable use is drawn from two bore wells and treated at an on-site treatment plant. Raw water for use in the process plant is provided primarily from diverted discharge from the underground mine, backed up by additional capacity from bore wells and the option to draw water from Baupa Creek. Make-up water can also be supplemented by decant water from the TSF.

18.2 MINE

Underground mining at Kainantu operated from 2004 to 2008. The majority of the mining infrastructure remains in place and is summarised below:

18.2.1 Lower 800 Portal And Workshop

The Lower 800 Portal area encompasses infrastructure for utilisation and security of the Irumafimpa Mine. Key elements of the infrastructure are:

- Power generation platform; This raised concrete platform formerly housed and sheltered five generator units and power regulation infrastructure providing underground back-up power
- Workshop and secure store rooms; a facility comprising four containers stacked two high and roofed with sheeting iron. The facility provides secure storage for cap lamp recharging station, re-breather units, small equipment and general consumables. A covered work deck provides shelter from weather during maintenance and servicing of underground plant. The underground tag-board and mine entry log is also housed here.
- Reinforced underground portal including security gates.
- Washdown bay, ablutions hut, laydown area.

The lower portal facility is located less than three hundred meters from a local settlement named Kokomo, comprised of Pomasi residents and Billimoian settlers. There have been no security issues for the portal from the settlement.

18.2.2 Underground Mine

The Irumafimpa Underground Mine comprises:

- Lower 800 Portal, Upper 1300 Portal, Puma manway Portal (1325RL), and various escape ways.
- 6 km incline to working levels. The incline is 5m x 5m, from the 840 portal to the switchback at the Kora turnoff, where breakthrough of the decline from the working levels occurred. The upper section of the incline from the switchback is 4m x 4m.
- Working levels 16 to 23, each developed with footwall drive, ore development drives, and ancillary crosscuts and stoping development. The working levels are constructed at 3m x 3.5m.
- Two ore passes dropping from the upper levels to the lower section of the incline.

18.2.3 Upper 1300 Portal

Most of the infrastructure at the Upper 1300 Portal which had been used during mine operations has been removed from the site. The site is currently not accessible from the underground mine due to a collapse along the internal access route.

18.2.4 1400 Level Camp

Following closure of the underground mine in 2009, the majority of the 250 man 1400 Level Camp was decommissioned and removed from the site. One building remains which facilitates security services for the upper mine openings and prevention of illegal mining.

18.3 PROCESSING PLANT

The Kainantu processing plant is located approximately 7 km from the opening of the 800 portal which accesses the Irumafimpa Mine. The plant was on care and maintenance between December 2008 and September 2016. Simple processing technology was used and following crushing, screening and grinding, sulphide bearing material was separated from non-mineralized host rock by flotation and a gold-rich flotation concentrate sold. Further details of the processing plant are in Section 13 Mineral Processing and Metallurgical Testing.



Figure 53. Oblique view of Process Plant and office infrastructure area (circa 2012)

Source: Barrick 2012

18.4 OFFICE

Additional infrastructure at the property includes an accommodation camp at Kunian, administration offices, warehouses, equipment workshops, exploration area and an assay laboratory.

18.5 EXPLORATION AREA

An office facility measuring 16m x 11m provides facility for an exploration team to operate at the site. The office is accompanied by a 23m x 21m core processing shed with extensive roller-racking for core logging (Figure 54). A warehouse facility of 7m x 22m provides secure locked storage for all exploration equipment and consumables, and a container laydown provides further storage for equipment and sulfidic core which would otherwise be susceptible to weather. A palletised core farm contains all available core from the history of the Project. A separate ablutions building is also located at the site.



Figure 54. Profile view of Exploration Coreshed (left foreground), warehouse (left background) and offices (right foreground) (circa 2014)

18.6 ACCOMMODATION CAMP

Accommodation at Kumian Camp (Figure 55) consists of a series of single person/shared ablution type facilities, as well as fully ensuited rooms for senior personnel. The current optimum capacity of the camp is 365 personnel. This can be expanded by refurbishment of existing (closed) accommodation in the camp to accommodate a further 203 personnel (568 personnel in total). The primary security post and gate house is located in a 6m x 4½m building at the entrance to the site on the access road.



Figure 55 Aerial and Ground view of Kumian Accommodation Camp (circa 2009)

Mess/catering facilities for 116 persons provide three meals a day for site personnel in accordance with required health standards. These facilities are officially inspected monthly and are randomly monitored by site OH&S staff on a weekly basis. Grounds and surrounds will continue to be maintained by a contract company, but Mess buildings and infrastructure will be maintained by K92ML.

The camp also contains a health/first aid clinic for the benefit of K92ML's employees. The clinic is sufficiently furnished to stabilise injured personnel prior to transport. It contains a paramedic's office, treatment couch, emergency treatment room, bathroom, dispensary, records storage and a waiting area. The clinic is supported by a mobile ambulance for paramedics and clinic staff.



Figure 56. Aerial view of Process Plant, Tailings Storage Facility and Accommodation Camp (circa 2009)
Source: Barrick 2012

19 MARKET STUDIES AND CONTRACTS

K92 signed an offtake agreement in June 2016 with InterAlloys Trading Limited (“ITL”) covering the first three years of concentrate production from the Kainantu mine. The terms provide for payment of gold, silver and copper contained in the concentrate.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL LIABILITIES AND MINE CLOSURE

To the extent known by Nolidan, there are no known environmental liabilities on the property which were not fully disclosed in the Mine Closure Plan by Barrick dated November 2010, a summary of which is given below:

The estimated closure costs are reported in two ways, namely as the Asset Retirement Obligation (ARO) and Life-of-Mine (LOM) costs. The ARO reflects expected costs as of the end of a calendar year (the ARO Year) as defined by the Financial Standards Board (FASB) Statement 143. Both the ARO and LOM costs calculated are undiscounted and based on third party cost rates.

The mine closure costs have been calculated in accordance with the Barrick Mine Closure Planning and Cost Estimation Guideline which outlines the approach to estimating costs associated with mine site reclamation, closure and decommissioning. The Barrick Reclamation Cost Estimator (BRCE) model has been used to determine the 2010 costs.

The un-discounted ARO closure cost as at 31 December 2009 was determined as \$5.94m. This estimate has been reviewed based on operational changes and closure review. The un-discounted ARO closure cost estimate for 31 December 2010 is \$ 6.86m.

The un-discounted LOM closure cost as at 31 December 2009 was determined as \$5.97m. This estimate has been reviewed based on operational changes and the closure review. The LOM closure cost estimate for 31 December 2010 is \$6.89m.

It should be noted that in 2010 the 'Direct Total' cost includes a 16% contractor profit and administration fee within the labour rate, whereas in previous years a 20% P&G fees was applied to the overall total cost.

Table 42 Mine Closure Costs - Barrick 2010

KAINANTU	Description	2009 ARO (from BRCE)	2010 ARO (from BRCE)
Waste rock Dumps	Waste Rock Dumps	\$32,186	\$37,179
Tailings impoundments	Tailings Impoundments	\$236,238	\$263,413
Pits	Pits	\$0	\$0
Roads	Roads	\$2,494	\$0
Processing areas	Heap Leach Facilities	\$0	\$2,885
	Landfills	\$3,645	\$4,107
	Buildings	\$194,330	\$306,034
	Other Demo & Equip Removal	\$680,000	\$708,800
	Yards	\$29,436	\$33,577
	Process Ponds	\$6,089	\$6,285
Backfilling, Adits, Shafts	General rock Hauling/backfilling	\$0	\$0
	Adits & Declines	\$38,651	\$41,166
	Shafts	\$878	\$1,015
Drainage and Sediment Control	Drainage & Sediment Control	\$8,075	\$12,097
Wells and Bores	Wells & Bores	\$3,816	\$4,537
Exploration rehabilitation	Exploration Holes	\$0	\$0
	Exploration Roads & Pads	\$0	\$0
	Trenches	\$0	\$0
Waste, Decontamination and Effluent Disposal	Waste Disposal	\$48,309	\$50,643
	Solution/water pumping	\$0	\$0
	Solution/water Evaporation	\$0	\$0
	Solution/water Management	\$0	\$0
	Decontamination	\$0	\$0
Other Costs including Closure Management, Admin etc	Other user costs	\$1,891,640	\$2,081,982
	Miscellaneous costs	\$849,534	\$942,814
	Closure Plan Management	\$20,000	\$120,750
	Construction Management	\$256,656	\$300,558
	Monitoring & Maintenance	\$220,993	\$458,207
	General & Administration	\$0	\$1,060,000
	Human Resources	\$430,000	\$424,000
	Direct Totals	\$4,952,970	\$6,860,049
	Contractor P&G (including profit)	\$990,594	\$0
	Total	\$5,943,563	\$6,860,049

20.2 TAILINGS STORAGE FACILITY

A tailing storage facility (TSF) is located downstream of the process plant adjacent to the Kumian Creek, which flows into the Baupa River. Tailings are reject from the flotation circuit.

The tailings storage facility is classified as a high hazard dam and contains tailings material. Runoff from within the dam is captured in catchment ponds behind the dam wall and is intermittently decanted into the tailings treatment ponds prior to discharge to Kumian Creek. The tailings material remains saturated as meteoric waters have been allowed to pond in the TSF. Water quality of the discharge from the ponds indicates that the water quality does not pose a risk to the receiving environment.

20.2.1 Tailings Disposal

It is reported that nominally 285,000 tonnes of tailings were produced by the plant during the years of production. The waste stream generated from the processing of ore comprises of sand tailings from the flotation circuit. The flotation tailings were relatively inert, composed primarily of quartz and waste rock sand and only very minor sulphur bearing minerals. However inspections of the tailings material indicate it does possess acid producing potential. A water cover is maintained over the material within the TSF which has prevented oxidation. No detailed studies have been completed on tails characterisation.

The only water discharge from the plant was contained in the flotation tailings, and pumped to the tailings dam. Any over-accumulation of decant water in the TSF was discharged to the overflow wetland system. Overflow and decant from the TSF flows through a wetland system prior to discharge to Kumian Creek. Barrick verbally confirmed that the amount of AMD water currently being discharged into the creek is within acceptable levels as governed by the Department of Environment.

20.2.2 Future Tailings Capacity

It is reported that approximately 307,000t of ore was fed to the plant over the life of the mine, with 93% reporting to the tailings for 285,000t.

Assuming a total capacity of 545,000t, and utilisation of 285,000t to date, the remaining capacity of the TSF would be around 238,000t dry or 170,000m³. In 2013 Golders estimated nominally 280,000m³ capacity remaining based on the observation of 2m remaining freeboard on the TSF wall.

No detailed survey reconciliation has been completed on the current capacity of the TSF. The two estimation methods presented here have a variance of 110,000m³. A detailed survey reconciliation will be completed as a high priority during the process of refurbishment of the Plant.

20.3 REQUIRED PERMITS

The following permits are required for mining operations:

- License to keep, store or possess explosives;
 - The mine is currently licensed to keep, store and possess explosives
- Permit for Persons using Explosives;
 - A process has been followed whereby Shotfirers have been trained and certificated through the Mine Regulatory Authority the Explosives Inspectorate falling under the Department of Labour and Industrial Relations
- Conveyance of Explosives & Dangerous Goods;
 - Explosives transport from Lae to the mine site are outsourced to a transport contractor (Mapai Transport) who has the necessary permits and approvals to transport dangerous goods. Kainantu Gold Mine also has the required permit to transport explosives on the mine site between the Surface Explosives Magazine and the underground workings.
- Approval to recruit non-citizens;
 - K92 Mining Limited has the necessary approvals to employ non-citizens through the PNG Department of Immigration
- Gold Export License;
 - K92 Mining Limited has applied for and is in possession of the required Gold Export License

- Exchange Control for Establishing Foreign Bank Accounts;
 - Approved by the Bank of PNG.
- Tax Clearance Certificates for Transfer of Funds out of PNG;
 - K92ML will apply for this clearance from the Commissioner of Taxation as and when needed.
- Liquor License;
 - A management decision was made that the mine accommodation camp would remain a “Dry Camp” and a liquor license is therefore not required.
- Certificate to Conduct Business as a Foreign Enterprise;
 - Not required as K92ML will be operating through a PNG company.
- Registration of an Overseas Company under the Companies Act;
 - Not required as K92ML will be operating through a PNG company.
- Data Transmission VSAT;
 - Not required at this time.
- Radio Licenses;
 - Licenses have been issued to the mine site by NICTA (National Information and Communications Technology Authority).

20.4 ENVIRONMENTAL PERMITS

Environmental Permits for the Property are for Water Extraction and Waste Discharge. Environmental permits for the site are current until 31st December 2053. The various iterations of the Permits are described here:

- 14/06/2002; Grant of permits - Water Extraction WE-L3(9), Waste Discharge WD-L3(32)
- 30/08/2004; Amendment for Water Extraction WE-L3(13), Waste Discharge WD-L3(34).
- 12/09/2005; Amendment for Water Extraction WE-L3(13), Waste Discharge WD-L3(34).
- 11/12/2007; Transfer for Water Extraction WE-L3(13), Waste Discharge WD-L3(34). Transferred from Highlands Kainantu Ltd to Barrick Kainantu Ltd.

20.5 MEMORANDUM OF AGREEMENT (MOA)

The original tenement holder, Highlands Pacific Limited (“HPL”) signed a Memorandum of Agreement (MOA) with the State, the Eastern Highlands Province (“EHP”) Government, the Kainantu LLG, the Billimoian Landowners Association (“BLA”), and Associated Landowners on 11th November 2003. This MOA provides for the allocation and use of the royalties derived from the project for the benefit of all stakeholders.

The agreement was to be reviewed five years after consummation, i.e. in 2008, and bi-annually thereafter. There have been no reviews of the MOA due initially to delays in completion of an investigation into Landholding at the Project by the Land Titles Commission (“LTC”), and subsequently due to further delays from appeals to the determination by the LTC in 2009.

The MOA would normally have expired with ML150 on 13th June 2014. However, in line with the continuance of the mining lease under Section 112 of the Mining Act 1992, the MOA will continue in force unless the Minister for Mining decides not to extend the term of the mining lease.

K92ML has discussed and agreed with the MRA that the review of the MOA and Compensation Agreement (see 5.4 below) will be delayed until the LTC has finalised review of all appellants to the 2009 LTC determination, and the primary Landholders for the Project have been declared. In the interim, K92ML will comply with the tenets of the MOA and will resurrect aspects of the MOA which have been closed while the project has been in care and maintenance.

20.5.1 Memorandum of Understanding (MOU)

HPL signed a Memorandum of Understanding (MOU) on 21st August 2003 with the Billimoian Landowners Association (BLA). The MOU was presented to the MRA as an attachment to the MOA. The document provides the framework and understanding for the Landowners to receive a 5% interest in the Project.

The agreement to provide to the Landowners a 5% carried equity in the Project was established by the Chief Warden Mr Timothy Kota through mediation after a breakdown in negotiations between the parties over the draft Compensation Agreement.

The MOU provides for Landowners to be issued a 5% carried equity in the Project through the issuing of shares in Highlands Kainantu Limited (“HKL”). The 5% interest was not issued due to uncertainty in relation to the parties who constitute Landowners which is being determined through the Land Title Commission (“LTC”) Appeals Review. The obligation in relation to the MOU now resides with K92 Holdings to issue a 5% carried equity interest in K92 Mining Limited once the LTC has issued its determination.

The MOU also provides that 65% of the dividends from the 5% equity will be used to repay capital costs to the parent company and 35% will be paid to the Landholders until the capital has been fully repaid.

This MOU has no legal or binding effect, however K92PNG agreed with Barrick Niugini under the K92ML Purchase Agreement to pursue in good faith negotiations to implement the terms of the MOU and convey a 5% equity interest in K92ML to the BLA.

20.5.2 Local Business Development Policy (LBDP)

This document, dated August 2003, was prepared as Annexure A to the MOA. The policy sets out the principles by which direct assistance will be given to the Landowners and local Community. K92ML will continue to operate under the tenets of this Policy.

20.5.3 Community Sustainable Development Plan (CSDP)

This document, dated August 2003, was presented to the MRA as Annexure B to the MOA.

The Plan provides for coordinated management of the benefit streams arising from the mining operation, to ensure that community development was delivered in a sustainable manner.

Key obligations to the Developer under the Plan are:

- Royalties. Distribution of royalties to be to the Public Infrastructure Trust Fund for management under the CSDP.
- Community Facilities Grant (CFG). K600,000 allocated by HPL for high priority community development projects.
- Structural Support Grant (SSG). A grant provided between the commencement of commercial production and commencement of payment of company tax.
- Tax Credit Scheme (TCS). The TCS of applicable tax credits to fund local infrastructure projects.

20.6 COMPENSATION AGREEMENT

HPL signed a Lands and Environment Compensation Agreement with identified impact communities in June 2003. The agreement was to be reviewed three years from commencing commercial

production, and every three years thereafter. There have been no reviews of the agreement due initially to delays in completion of an investigation into Landholding at the Project by the Land Titles Commission (LTC), and subsequently due to further delays from appeals to the determination by the LTC in 2009.

K92ML has discussed and agreed with the MRA that the review of the MOA and Compensation Agreement will be delayed until the LTC has finalised review of all appellants to the 2009 LTC determination, and the primary Landholders for the Project have been declared.

Upon re-commencement of the Project, K92ML will convene a forum for discussion to determine and ratify a method for implementation of the Compensation Agreement in an operational phase now that the LTC has made its 2009 determination. These forums will involve the signatories to the Compensation Agreement (which includes all beneficiaries of the 2009 LTC determination), the LTC, the Provincial Administration, and the Development Coordination Division arm of the MRA.

20.7 OTHER SIGNIFICANT FACTORS AND RISKS

Barrick conducted an extensive investigation into the matter of the all outstanding sales royalties and compensations payable by K92ML since the commencement of the project. Some of these monies remain outstanding due to internal disputes over land ownership, the resolution of which is beyond K92ML's control. Barrick, in conjunction with the K92ML Purchase Agreement, set up bank accounts under K92ML to hold these monies in trust. Considerable effort was expended to ensure that Barrick had determined the entire value of the amounts outstanding. Where there are discrepancies, Barrick has erred on the side of caution with respect to determining amounts payable. However, any discrepancies discovered after closing of the K92ML Purchase Agreement are the responsibility of the new management. Barrick considers that once the bank accounts are in place and the populated with the relevant monies, they have concluded their obligation to fully investigate and hand over the outstanding monies for the new administration's future management and dispersal.

Access to areas with existing surface miners is challenging, although well under control at the present time. K92ML maintains a security presence at the main artisanal mining areas (Kora and Irumafimpa). The Security teams are supervised by K92ML personnel, but are comprised of local Billimoian security contractors who source their personnel from the nearby Billimoian villages. There have been no significant artisanal mining issues since this approach was employed (Barrick, 2014).

Land Ownership and access issues result from inter-clan fighting. This results in delays in assessment and advancement of exploration properties. The risk to property is minimal and is mitigated by ongoing and proactive Community Relations ("CR") engagement.

Strong community relations are imperative to exploring in PNG with community agreement required before any exploration activities can take place. The Kainantu area has been beset with CR issues since modern exploration commenced, resulting in many prospective areas not being explored and very limited drilling. The K92ML CR team have worked to gain the trust of the local landowners and this has resulted in access being granted in many areas which have not previously undergone detailed exploration.

As part of Barrick's commitment to deal equitably with local communities, Community Engagement Agreements between Barrick and local landowners were put in place prior to any exploration activities commencing. These set out what the community could expect from Barrick, including incentive payments, rental payments and dispute resolution procedures. The Exploration CR team includes up to four community relations officers and six village liaison officers supported by a community relations coordinator and Community Relations Manager.

Community relations personnel deal with all access negotiations prior to any exploration activities being undertaken, calculate, resolve and payout compensation payments and attend all Warden's Hearings. (Barrick 2014)

K92ML has undertaken to continue this pro-active CR engagement with affected landowners.

As to political risk, Nolidan notes that on the Fraser Institute's Investment Attractiveness Index for 2014 Papua New Guinea ranks higher than Indonesia and the Philippines but below Australia and New Zealand (Jackson and Green, 2015). Its score was 48.5 compared with 56.2 in 2013.

21 CAPITAL AND OPERATING COSTS

Note Capital allowances were only included in cashflows estimated for Kora.

21.1 KORA CAPITAL EXPENDITURE EQUIPMENT AND FACILITIES

Conceptual capital expenditure allowances in the cashflow model prepared by AMDAD are summarised in Table 43. In addition to mining equipment and infrastructure an estimate is included for expansion of the existing process plant to 400,000 tpa capacity. This estimate was provided to K92ML by Mincore in its report on the Process Plant upgrade (Mincore, 2016).

Table 43: Kora Capital Expenditure – Equipment and Facilities

Item	Cost \$M (USD)	Source
Ventilation Fans	1.0	Infomine & AMDAD allowance
Ventilation Civils	1.0	AMDAD allowance
Electrical Infrastructure	2.0	AMDAD allowance
Portal (1825mRL)	1.0	AMDAD allowance
Mobile Equipment	5.0	AMDAD allowance
Cement Backfill Infrastructure	1.0	AMDAD allowance
Kora Mine Facilities	2.5	AMDAD allowance. Provision for construction of any additional underground facilities.
Processing Plant Expansion	3.3	Mincore report September 2016
Total	16.8	

Kora Mine Facilities consist of any infrastructure development, and infrastructure items to be constructed underground at Kora instead of being shared with Irumafimpa. These include:

- Offices and pre-start facilities
- Workshop, refuelling and equipment parking bays
- Magazine
- Ladderways

21.2 KORA CAPITALISED DEVELOPMENT COSTS

All waste development (lateral and vertical), except crosscuts from footwall drives to the orebody drives, is treated as a capital cost. Table 44 is a summary of quantities and costs for capitalised development as estimated by AMDAD.

Table 44: Kora Capital Expenditure - Development

Item	Quantity (m)	Cost (USD) \$M
Decline Development	4,841	21.8
Lateral Waste Development	13,737	48.1
Vertical Waste Development	2,078	10.1

21.3 KORA MOBILE FLEET

Table 45 summarises the mobile fleet requirements for the LOM schedule.

Table 45: Kora Capital Expenditure – Mobile Plant

Mobile Fleet	Number	Note
Development Jumbo	1	Twin-boom jumbo suitable for rockbolting and face boring. Sandvik DD420 onsite is intended for Kora exploration drive.
LHD Unit	1	~6m ³ LHD unit. Toro 1400 (5.4m ³) onsite is intended for Kora exploration drive.
Haul Truck	2	~35 tonnes ADT. Used Volvo A25D sourced for Kora exploration drive. Komatsu HM350 and Atlas Copco MT2010 ADTs onsite for Irumafimpa ore haulage.
Development Jumbo - Contractor	2	Contractor units for waste development. Twin-boom jumbo suitable for rockbolting and face boring.
LHD Unit – Contractor	1	Contractor units for waste development. ~6m ³ LHD unit.
Articulated Haul Truck - Contractor	2	Contractor units for waste development. ~35 tonnes articulated dump trucks.
Production Drill Rig	1	Suitable for 64-89mm upholes. Boart StopeMate to be purchased for Irumafimpa.
Cable Bolt Rig	1	Compact mechanised cable bolt rig (e.g. Sandvik DS421)
Shotcrete Rig	1	Compact mobile shotcrete rig (e.g. Jacon Maxijet)
Integrated Tool carrier	1	CAT 924IT has been purchased for Irumafimpa. Will also be used for chargeup.
Telehandler/Manitou	1	Backup for integrated tool carrier. JCB Telehandler onsite at Irumafimpa.
Grader	1	CAT 12G onsite at Irumafimpa may be suitable if access is suitable.

21.4 IRUMAFIMPA OPERATING COSTS

Total operating cost estimate for Irumafimpa 3 year plan is \$70.3M

Total operating cost estimate for Irumafimpa LOM plan is \$198.7M

These costs are derived from the estimated development and production quantities from AMDAD's schedule applied to the unit costs provided by or confirmed by K92ML.

These costs are listed in the table below

Table 46: Operating costs for the Irumafimpa Life Of Mine Plan

	Unit Cost US\$/t	Cost US\$M								
		Total	YR1	YR2	YR3	YR4	YR5	YR6	YR7	YR8
Mining	77.8	138.8	12.7	20.0	20.3	19.7	22.3	22.8	17.7	3.2
Processing and site costs	42.6	59.8	4.8	8.5	9.1	8.7	9.6	9.5	7.9	1.6
Total	120.4	198.7	17.6	28.6	29.4	28.5	32.0	32.3	25.6	4.8

21.5 KORA OPERATING COSTS

Total operating cost estimate for Kora scoping study is \$403.5M. These costs are derived from the estimated development and production quantities from AMDAD's schedule applied to the unit costs provided by or confirmed by K92ML.

These costs are listed in the table below

Table 47: Operating costs for the Kora Mine Plan

	Unit Cost US\$/t	Cost US\$M									
		Total	YR1	YR2	YR3	YR4	YR5	YR6	YR7	YR8	YR9
Mining	89.5	287.2	7.7	34.2	35.9	35.4	35.4	35.5	35.4	35.4	32.3
Processing and site costs	36.0	116.3	3.3	13.5	14.4	14.4	14.4	14.4	14.4	14.4	13.1
Total	123.7	403.5	11.0	47.7	50.3	49.8	49.8	49.9	49.8	49.8	45.4

22 ECONOMIC ANALYSIS

22.1 IRUMAFIMPA MINE PLAN

In addition to producing schedules with production tonnes and grades and development metres (Section 16.1), AMDAD also prepared a conceptual cashflow and discounted cashflow (DCF) derived from these quantities, with allowances for mine capital expenditure.

When reviewing these figures it should be noted that:-

- Non-mining economic and processing parameters assumed and referred to in the study are conceptual. They were applied for the purpose of identifying the part of the Inferred Resource that notionally may be economic, in order to prepare conceptual extraction designs.
- Schedules are based on conceptual development and stoping quantities and not practical designs.
- Where cashflow schedules are provided based on these assumed parameters they should be treated with caution, and they should not be interpreted as a measure of the value of the deposit.

The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessments will be realized.

The 3 Year Mine Plan includes a simplistic cashflow model based on the preliminary unit costs and revenue assumptions that were adopted to define the cutoff grade for the MSO modelling. These cost and revenue parameters were applied to the scheduled mining activity and production.

Table 48: Simplistic Cashflow Model for 3 Year Mine Plan

	Month	1	2	3	4	5	6	7	8	9	10	11	12	
Production (mill feed)	Total													
Prod. & Dev. Ore	tonnes	491,834	280	2,985	4,107	3,789	5,749	6,859	9,625	13,516	17,000	16,899	16,270	16,550
	au g/t	8.40	8.76	7.27	6.49	6.06	6.84	8.85	12.22	9.17	8.84	9.23	7.55	6.03
	ag g/t	5.78	0.32	0.77	5.18	10.44	9.17	5.18	4.69	6.23	5.62	4.08	5.72	5.99
	cu g/t	0.11	0.02	0.05	0.12	0.19	0.14	0.09	0.10	0.12	0.07	0.04	0.09	0.13
	Au oz.	132,895	79	697	857	738	1,265	1,951	3,782	3,987	4,832	5,015	3,950	3,206
Development (waste)	m	3,834	176.6	154.6	144.4	153.3	91.1	50.8	54.2	168.2	157.8	171.3	213.4	229.5
Development (ore)	m	5,083	9.4	97.4	134.6	116.7	187.9	219.2	224.8	265.8	262.2	262.7	206.6	204.5
Cost														
dev.	\$M	5.8	0.3	0.2	0.2	0.2	0.1	0.1	0.1	0.3	0.2	0.3	0.3	0.3
prod.	\$M	64.6	0.0	0.4	0.5	0.5	0.8	0.9	1.3	1.8	2.2	2.2	2.1	2.2
Net revenue	\$M	126.7	0.1	0.7	0.8	0.7	1.2	1.9	3.6	3.8	4.6	4.8	3.8	3.1
Net cashflow	\$M	56.4	-0.2	0.0	0.1	0.0	0.3	0.9	2.3	1.8	2.1	2.3	1.3	0.5
DCF, \$M	8%	\$49.2												

	Month	13	14	15	16	17	18	19	20	21	22	23	24	
Production (mill feed)	Total													
Prod. & Dev. Ore	tonnes	491,834	16,908	16,871	16,964	16,416	16,322	16,599	16,555	16,521	16,422	16,377	16,866	16,902
	au g/t	8.40	8.20	9.04	12.64	7.33	7.26	8.81	7.76	6.51	7.92	8.28	7.43	9.34
	ag g/t	5.78	4.62	6.26	9.58	6.21	6.94	5.51	4.38	6.68	7.80	2.38	1.90	3.99
	cu g/t	0.11	0.10	0.07	0.12	0.10	0.11	0.08	0.11	0.12	0.12	0.09	0.08	0.13
	Au oz.	132,895	4,459	4,901	6,892	3,869	3,811	4,699	4,131	3,456	4,183	4,358	4,027	5,074
Development (waste)	m	3,834	126.4	174.3	161.9	70.5	147.2	119.9	138.0	195.9	130.6	118.4	103.4	69.3
Development (ore)	m	5,083	307.6	217.7	272.1	199.5	131.8	150.1	141.0	83.1	139.4	160.6	166.6	209.7
Cost														
dev.	\$M	5.8	0.2	0.3	0.2	0.1	0.2	0.2	0.2	0.3	0.2	0.2	0.2	0.1
prod.	\$M	64.6	2.2	2.2	2.2	2.2	2.1	2.2	2.2	2.2	2.2	2.1	2.2	2.2
Net revenue	\$M	126.7	4.3	4.7	6.6	3.7	3.6	4.5	3.9	3.3	4.0	4.2	3.8	4.8
Net cashflow	\$M	56.4	1.8	2.2	4.1	1.4	1.3	2.1	1.6	0.8	1.6	1.8	1.5	2.5
DCF, \$M	8%	\$49.2												

		Month	25	26	27	28	29	30	31	32	33	34	35	36
Production (mill feed)		Total	16,667	16,667	16,667	16,667	16,667	16,667	16,667	16,667	16,667	16,667	16,667	16,667
Prod. & Dev. Ore	tonnes	491,834	16,435	16,865	16,757	16,577	16,499	16,494	16,424	16,424	15,894	15,997	14,115	0
	au g/t	8.40	8.54	8.21	8.84	8.28	8.78	9.39	8.54	7.37	8.53	8.27	8.56	0.00
	ag g/t	5.78	3.47	9.74	20.54	7.53	5.51	3.49	4.86	3.13	3.36	3.51	1.69	0.00
	cu g/t	0.11	0.11	0.16	0.22	0.14	0.07	0.11	0.19	0.17	0.14	0.11	0.07	0.00
	Au oz.	132,895	4,514	4,453	4,762	4,415	4,655	4,977	4,508	3,891	4,361	4,252	3,886	0
Development (waste)	m	3,834	102.1	130.3	85.0	78.9	97.6	19.3	0.0	0.0	0.0	0.0	0.0	0.0
Development (ore)	m	5,083	176.9	121.7	194.0	94.1	74.5	51.1	0.0	0.0	0.0	0.0	0.0	0.0
Cost														
dev.	\$M	5.8	0.2	0.2	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
prod.	\$M	64.6	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.1	2.1	1.9	0.0
Net revenue	\$M	126.7	4.3	4.2	4.5	4.2	4.4	4.7	4.3	3.7	4.2	4.1	3.7	0.0
Net cashflow	\$M	56.4	2.0	1.8	2.2	1.9	2.1	2.6	2.1	1.6	2.1	2.0	1.9	0.0
DCF, \$M	8%	\$49.2												

The cashflow model is simplistic in that the revenues are “instantaneous”, and attributed directly to the ore production from the mine without any time lag for stockpiling and processing. It is only for operating costs and revenue, and does not incorporate any deduction for project capital costs and it does not incorporate taxation and detailed financial and accounting factors.

AMDAD also prepared a conceptual Life of Mine (LOM) Schedule made up of the 3 Year Mine Plan as described above in Section 16.1, and including the remaining mining inventory defined by the MSO stopes generated. Some MSO stopes were excluded from the LOM Schedule according to the stope’s proximity to either the ground surface or existing voids. The LOM stope layout extends from the 1100mRL Level to the 1505mRL Level.

The LOM Schedule targets annual mill production of 200,000tpa using underground production constraints of load haul capacity of 750t per day and face advance of 9m per day.

AMDAD also generated a simplistic cashflow model for the LOM Schedule, based on the preliminary unit costs and revenue assumptions as for the 3 Year Mine Plan. Note, there is no allowance for capital expenditure in this cashflow model. The resultant cashflow is summarised below.

Table 49: Simplistic Irumafimpa LOM cashflow model

		Year	1	2	3	4	5	6	7	8
Production (mill feed)		Total	100,000	200,000	200,000	200,000	200,000	200,000	200,000	114,364
Prod. & Dev. Ore	Kt	1,403.3	113.6	199.7	214.0	205.0	225.6	223.7	185.1	36.6
	au g/t	8.18	8.31	8.39	8.16	7.50	7.59	8.39	8.91	9.22
	ag g/t	5.85	5.61	5.52	5.80	2.80	5.50	3.90	9.75	20.28
	cu g/t	0.19	0.10	0.10	0.13	0.07	0.14	0.23	0.44	0.87
	Au Koz.	369.0	30.4	53.9	56.1	49.4	55.0	60.3	53.0	10.9
Development (waste)	m	9,639	1,765	1,556	870	1,032	1,566	1,962	889	0
Development (ore)	m	11,033	2,192	2,179	1,314	1,441	1,650	1,332	925	0
Cost										
dev.	\$M	14.5	2.6	2.3	1.3	1.5	2.3	2.9	1.3	0.0
prod.	\$M	184.2	14.9	26.2	28.1	26.9	29.6	29.4	24.3	4.8
Net revenue	\$M	351.9	28.9	51.4	53.5	47.1	52.5	57.5	50.5	10.4
Net cashflow	\$M	153.2	11.4	22.8	24.1	18.7	20.5	25.2	24.9	5.5
DCF, \$M	8%	\$110.4								

Key estimates from the Irumafimpa Mine Plan prepared by AMDAD are:

- Planned treatment of 0.49Mt tonnes at 8.4 g/t Au, 5.8 g/t Ag, 0.11%Cu over the 3 years of the mine plan generating a net cashflow of USD \$56 million.
- Planned treatment of 1.40Mt tonnes at 8.2 g/t Au, 5.8 g/t Ag, 0.19%Cu over 8 years generating a net cashflow of USD \$153 million

22.2 KORA MINE PLAN

In addition to producing schedules with production tonnes and grades and development metres (Section 16.2), AMDAD also prepared a conceptual cashflow and discounted cashflow (DCF) derived from these quantities, with allowances for mine capital expenditure.

When reviewing these figures it should be noted that:-

- Non-mining economic and processing parameters assumed and referred to in the study are conceptual. They were applied for the purpose of identifying the part of the Inferred Resource that notionally may be economic, in order to prepare conceptual extraction designs.
- Schedules are based on conceptual development and stoping quantities and not practical designs.
- Where cashflow schedules are provided based on these assumed parameters they should be treated with caution, and they should not be interpreted as a measure of the value of the deposit.
- Operating cost estimates by Mincore for the treatment of the Kora ore were based on current operating costs for the K92ML concentrator, reagent consumptions determined by historical production, and calculated power consumption of new and modified equipment.
- Subsequent to the preliminary \$30/t G&A cost applied by AMDAD for the MSO work, K92ML advised a lower G&A cost of \$20/t should be used for cashflow modelling. This cost decrease was based on the latest site costs and the proposed increase in throughput from 200,000tpa to 400,000tpa. In addition the processing cost in the AMC Kainantu Mine Plan report (AMC, 2015) has a fixed portion of approximately 35%, which is primarily labour costs. K92ML advised that based on the Mincore scoping study the processing cost should be reduced to \$16/t. This means that the cutoff for MSO would be reduced, and that there may be some scope to further optimise the stope shapes with this lower cutoff. However AMDAD concluded that the nature of the grade distribution is such that the orebody is relatively insensitive to cutoff grade and the impact of any lower cutoff is likely to be minor.
- The estimates of tonnes and grade reported and scheduled in the Kora Scoping Study study do not constitute an Ore Reserve because:-
 - a) The resource estimate on which the tonnes and grade are based on an Inferred Resource. Inferred Resources are at too low a level of confidence to allow conversion to Ore Reserves.
 - b) There is insufficient geotechnical information to be confident in development and extraction design parameters and costs and the mine plan can only be considered conceptual.
 - c) Limited metallurgical testwork has been completed for the copper-gold mineralization and further work will be required to confirm the processing cost and recovery assumptions.

- Any reference to “ore” in the Scoping Study is simply a reference to that part of the resource, with appropriate adjustment for dilution and loss, that would be intended as mill feed, rather than waste, and which would be an Ore Reserve if all requirements of the JORC Code were met.

The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessments will be realized.

Key estimates from the Kora Mine Scoping Study prepared by AMDAD are:

- Over a 9 year operating life the plant would treat 3.2 Million tonnes averaging 7.1 g/t Au, 25 g/t Ag and 1.7% Cu (9.3 g/t Au Eq*).
- This would generate an estimated positive cash flow of US\$537 million using current metal prices if 15m levels are used in mining. If 25m mining levels are used then net cashflows are estimated as US\$558 million. This cashflow includes conceptual allowances for capital.
- Production of an estimated average of 108,000 Au Eq* ozs per annum over an 8 year period from Year 2 through to Year 9.
- An estimated Pre-tax NPV of US\$415 Million for 25m mining levels; or US\$397 Million for 15m levels; using current metal prices, exchange rate and a 5% discount rate;
- Initial Capital Cost is estimated to be US\$13.8 Million, including the US\$3.3 million for the plant upgrade identified in the Mincore Scoping Study, but excluding the proposed Kora exploration inclines and diamond drilling. Sustaining Capital Cost is estimated to be a further US\$64 million spent over the life of the Kora mining for 25m mining levels or US\$83 for 15m mining levels.
- Operating Cost per tonne is estimated to be US\$125/tonne for 25m mining levels or US\$126/tonne for 15m mining levels.
- Excluding Initial Capital Expenditure of US\$14M, Cash Cost is estimated to be US\$547/oz Au Eq (inclusive of a 2.5% NSR) and AISC of US\$619/oz Au Eq for 25m mining levels; or US\$549/oz Au Eq (inclusive of a 2.5% NSR) and AISC of US\$644/oz Au Eq for 15m mining levels.

Current Metal Prices used were: Au – US\$1,300/oz; Ag – US\$18/oz; Cu – US\$4,800/tonne.

*Au Eq – calculated on above Metal Prices.

22.2.1 15m Level Case

Annual cashflows are presented below for the 15m Level scenario. Project payback, on an undiscounted basis, is achieved in Quarter 8 (Year 2). Positive cashflow is expected from Qtr 5 (Year 2) onwards.

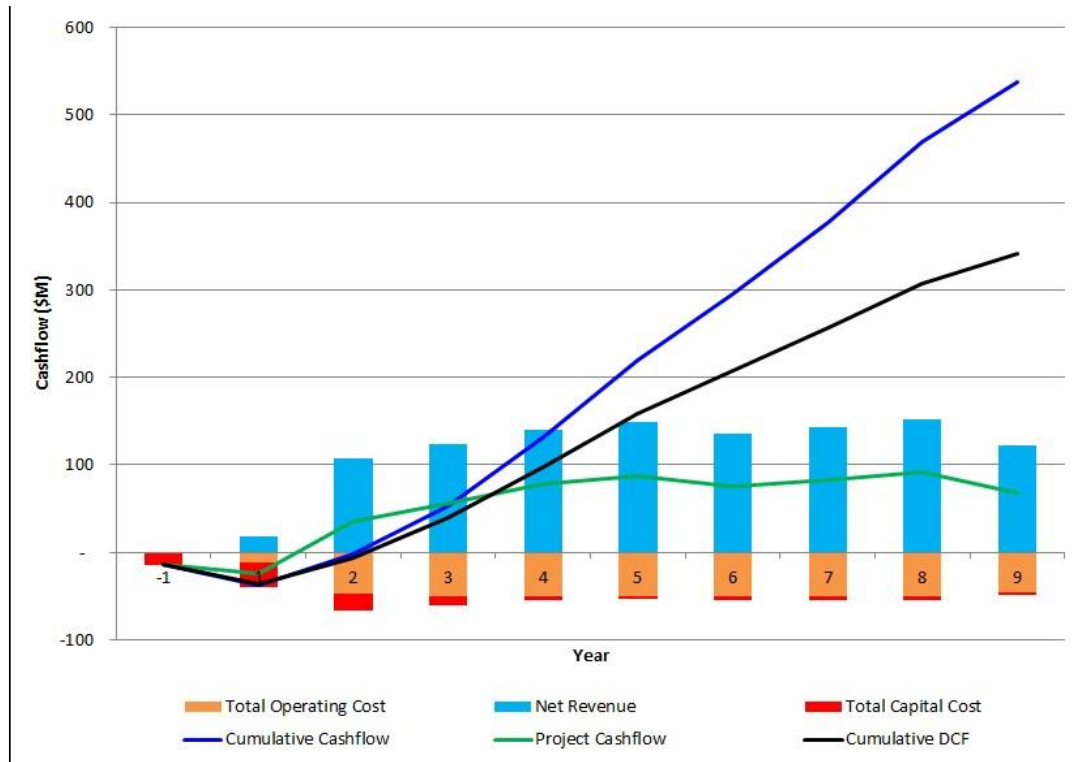


Figure 57: Kora Annual cashflow, cumulative cashflow and DCF; 15m Levels

Table 50: Kora Annual LOM Schedule; 15m Levels

Kora 15m Levels				unit cost	units	TOTAL	0	1	2	3	4	5	6	7	8	9	
Development																	
Horizontal		metres	36,088				-	6,398	5,809	4,130	3,009	3,165	3,662	3,690	3,421	2,806	
Vertical		metres	2,078			-	1,066	349	438	30	60	30	45	30	30	30	
Production																	
Tonnes		kt	3,209			-	70	375	400	400	400	400	400	400	400	364	
AUEQ		g/t	9.3				6.8	7.9	8.5	9.6	10.2	9.3	9.8	10.4	9.2		
Au		g/t	7.1				5.0	5.6	6.2	7.1	7.8	7.1	7.8	8.4	7.0		
Ag		g/t	25				18	21	24	28	29	26	22	24	26		
Cu		%	1.7				1.4	1.7	1.8	1.8	1.7	1.6	1.5	1.5	1.6		
Concentrate		kt dry	194			-	4	24	26	27	25	24	22	22	21		
		kt wet	210			-	4	26	28	29	27	25	24	24	23		
Metal in Concentrate																	
Gold Produced, AUEQ		oz.	878,572			0	14,076	86,639	99,920	112,696	119,556	109,575	114,811	122,935	98,364		
Gold Produced, AU only		oz.	671,004			0	10,354	61,881	72,567	84,070	92,286	84,125	91,436	98,880	75,405		
Silver		oz.	2,308,751			0	35,463	231,393	274,822	326,623	334,847	306,441	250,295	276,508	272,359		
Copper		t	48,589			0	892	5,949	6,505	6,672	6,274	5,876	5,506	5,600	5,315		
Economics																	
General Capital Cost	\$5.2/t	\$M	17			14	1	2	-	-	-	-	-	-	-	-	
Capitalised Waste Development Cost	\$24.9/t	\$M	80			-	28	17	11	4	4	5	5	4	3		
Total Capital	\$30.2/t	\$M	97			14	29	19	11	4	4	5	5	4	3		
Waste Development - Operating	\$1.8/t	\$M	6			-	2	1	1	0	0	0	0	0	0		
Stopping Cost - incld ore dev't	\$87.7/t	\$M	282			-	6	33	35	35	35	35	35	35	32		
Processing	\$16.0/t	\$M	51			-	1	6	6	6	6	6	6	6	6		
G&A	\$20.2/t	\$M	65			-	2	8	8	8	8	8	8	8	7		
Total Operating Cost	\$125.7/t	\$M	403			-	11	48	50	50	50	50	50	50	45		
Gold Revenue	\$265.5/t	\$M	852			-	13	79	92	107	117	107	116	126	96		
Silver Revenue	\$11.7/t	\$M	37			-	1	4	4	5	5	5	4	4	4		
Copper Revenue	\$70.5/t	\$M	226			-	4	28	30	31	29	27	26	26	25		
Total Revenue	\$347.7/t	\$M	1,116			-	18	110	127	143	152	139	146	156	125		
Production Cashflow (excl Capital & cost of sales)	\$222.0/t	\$M	712			-	7	62	77	93	102	89	96	106	80		
Cost of Sales	\$16.4/t	\$M	52			-	1	6	7	7	7	6	6	6	6		
Operating Cashflow (excl Capital)	\$205.6/t	\$M	660			-	6	56	70	86	95	83	90	100	74		
Royalty, Levy & NSR (Au, Ag & Cu)	\$8.3/t	\$M	27			-	0	3	3	3	4	3	3	4	3		
Project Cashflow (before tax), incl Capex	\$167.2/t	\$M	537			-14	-24	35	56	78	88	75	82	92	68		
Cumulative Cashflow (before tax), incl Capex	\$M	\$M	537			-14	-37	-3	53	132	220	294	376	468	537		
DCF @ 5%, incl Capex	\$123.8/t	\$M	397			-14	-22	31	48	65	69	56	58	62	44		

22.2.2 25m Level Case

When horizontal development costs are adjusted to reflect 25m level intervals, rather than 15m, project payback on an undiscounted basis is achieved in Year 2. Positive cashflow is expected from Year 2 onwards. The overall undiscounted project cashflow increases by \$21M compared to the base case with 15m levels.

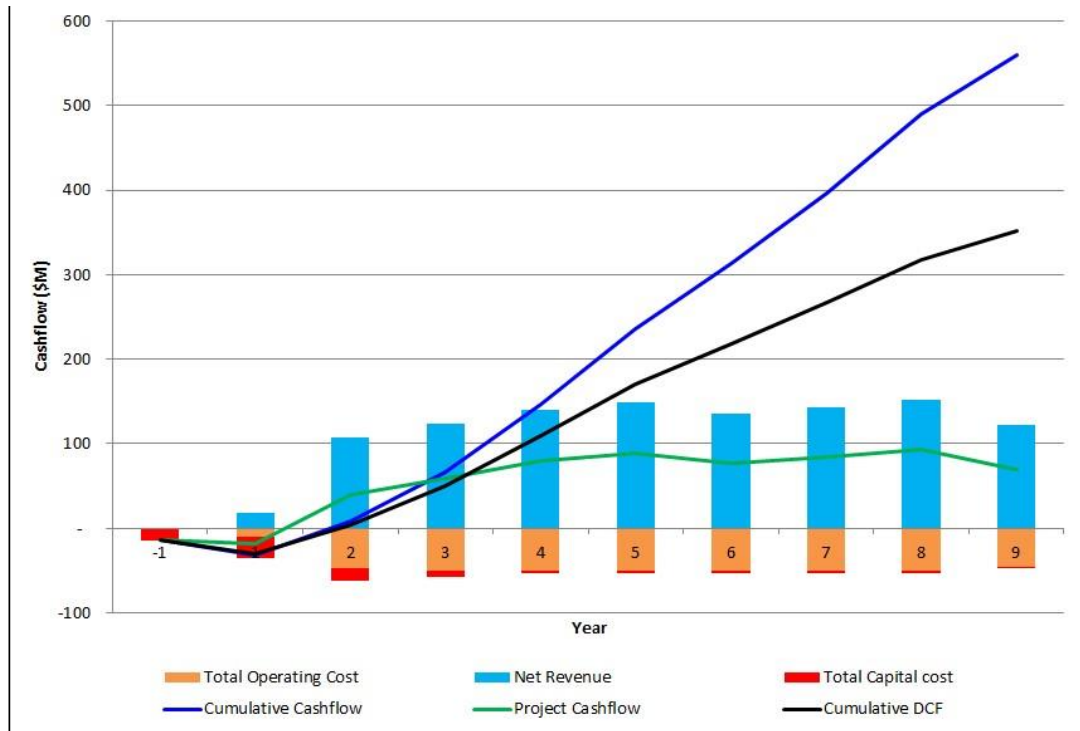


Figure 58: Kora 25m Level Case Annual cashflow, cumulative cashflow and DCF

The detailed LOM Development, Production and Economic schedule for the 25m level interval case is shown in the following table.

Table 51: Kora Annual LOM Schedule; 25m Levels

Kora 25m Levels	unit cost	Units	TOTAL	0	1	2	3	4	5	6	7	8	9
Development													
Horizontal		metres	29,951	-	4,819	4,391	3,307	2,624	2,808	3,148	3,277	3,059	2,517
Vertical		metres	2,078	-	1,066	349	438	30	60	30	45	30	30
Production													
Tonnes		kt	3,209	-	70	375	400	400	400	400	400	400	364
AUEQ		g/t	9.3		6.8	7.9	8.5	9.6	10.2	9.3	9.8	10.4	9.2
Au		g/t	7.1		5.0	5.6	6.2	7.1	7.8	7.1	7.8	8.4	7.0
Ag		g/t	25		18	21	24	28	29	26	22	24	26
Cu		%	1.7		1.4	1.7	1.8	1.8	1.7	1.6	1.5	1.5	1.6
Concentrate		kt dry	194	-	4	24	26	27	25	24	22	22	21
		kt wet	210	-	4	26	28	29	27	25	24	24	23
Metal in Concentrate													
Gold Produced, AUEQ		oz.	878,572	0	14,076	86,639	99,920	112,696	119,556	109,575	114,811	122,935	98,364
Gold Produced, AU only		oz.	671,004	0	10,354	61,881	72,567	84,070	92,286	84,125	91,436	98,880	75,405
Silver		oz.	2,308,751	0	35,463	231,393	274,822	326,623	334,847	306,441	250,295	276,508	272,359
Copper		t	48,589	0	892	5,949	6,505	6,672	6,274	5,876	5,506	5,600	5,315
Economics													
General Capital Cost	\$5.23/t	\$M	17	14	1	2	-	-	-	-	-	-	-
Capitalised Waste Development Cost	\$18.93/t	\$M	61	-	23	12	8	3	3	3	3	3	2
Total Capital	\$24.17/t	\$M	78	14	24	14	8	3	3	3	3	3	2
Waste Development - Operating	\$1.05/t	\$M	3	-	1	1	0	0	0	0	0	0	0
Stopping Cost - incld ore dev't	\$87.72/t	\$M	282	-	6	33	35	35	35	35	35	35	32
Processing	\$16.00/t	\$M	51	-	1	6	6	6	6	6	6	6	6
G&A	\$20.25/t	\$M	65	-	2	8	8	8	8	8	8	8	7
Total Operating Cost	\$125.02/t	\$M	401	-	10	47	50	50	50	50	50	50	45
Gold Revenue	\$265.54/t	\$M	852	-	13	79	92	107	117	107	116	126	96
Silver Revenue	\$11.65/t	\$M	37	-	1	4	4	5	5	5	4	4	4
Copper Revenue	\$70.49/t	\$M	226	-	4	28	30	31	29	27	26	26	25
Total Revenue	\$347.68/t	\$M	1,116	-	18	110	127	143	152	139	146	156	125
Production Cashflow (excl Capital & cost of sales)	\$222.66/t	\$M	715	-	7	63	77	93	102	89	96	106	80
Cost of Sales	\$16.35/t	\$M	52	-	1	6	7	7	7	6	6	6	6
Operating Cashflow (excl Capital)	\$206.31/t	\$M	662	-	7	56	70	86	95	83	90	100	74
Royalty, Levy & NSR (Au only)	\$8.28/t	\$M	27	-	0	3	3	3	4	3	3	4	3
Project Cashflow (before tax)	\$173.86/t	\$M	558	-14	-18	40	59	80	89	76	83	94	69
Cumulative Cashflow (before tax)		\$M		-14	-32	8	67	147	235	312	395	489	558
DCF @ 5%	\$129.44/t	\$M	415	-14	-17	36	51	66	70	57	59	63	45

23 ADJACENT PROPERTIES

Kainantu occurs within a well-endowed belt of epithermal and porphyry style mineralization that reportedly contains several major deposits (Figure 59). Nolidan is unable to verify this information and the information is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

K92ML does not have any interest in any adjacent properties.

24 OTHER RELEVANT DATA AND INFORMATION

Rehabilitation of the mine workings by K92ML commenced in March 2016. Refurbishment of the treatment plant by Mincore and Sun Engineering commenced in May 2016 and the plant was re-commissioned in September 2016. In order to comply with the terms of the ML150 renewal K92ML was required to refurbish the mine and mill by December 31, 2016. Rehabilitation of the mine and mill as required by the terms of ML150 has now been completed.

The remaining capacity of the tailings facility (TSF) is approximately 280,000m³. This equates to approximately 3 years at a planned tonnage of 180,000 tpa. Hence, additional geotechnical studies and approvals will be required prior to construction of a second lift to allow extra capacity to accommodate tailings for any future mine production.

25 INTERPRETATION AND CONCLUSIONS

25.1 EXPLORATION POTENTIAL

The Kainantu project is located in a recognized copper-gold province, as evidenced by the underlying geology and presence of nearby major projects operated by global majors Barrick, Newcrest and Harmony (Figure 59). There remain a significant number of major untested and early stage targets. Within ML150 are the Kora lodes which are strongly mineralized at the limit of drilling and open in all directions, as well as the Judd, Karempa and other unnamed mineralized lodes parallel to defined resources which have economically attractive grade in surface and/or drill samples from very limited work to date.

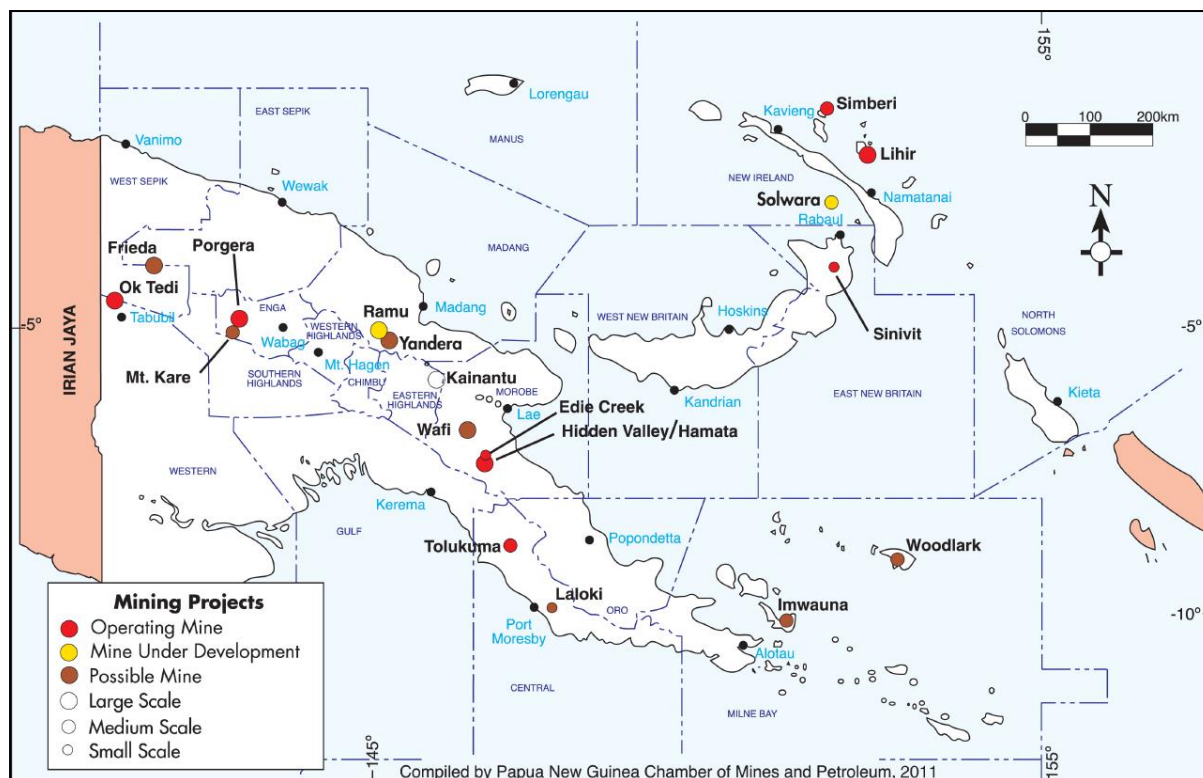


Figure 59. Location of Kainantu project and gold deposits within major mineralized province.

Source: PNG Chamber Mines and Petroleum 2011

25.1.1 ML150

Further investigation is required to understand the geological complexity of the veins at Kainantu and the controls on high grade shoots. This will require better resource definition. K92ML proposes close spaced drilling from existing underground workings to confirm indicated resources at Irumafimpa. The mineral resource is summarized in Table 25 and detailed in Table 24 as well as in the 'Summary' chapter of this report. Significant opportunity remains for resource extension within the immediate mine environment, including:

- The Irumafimpa-Kora vein system is open at depth, in the central areas beneath the top of the mountain (Eutompi) and to the South (Kora) beyond the ML150 boundary.
- Blocks shown in the Longitudinal Section below have been coloured by resource category. Turquoise blocks are blocks with only one sample supporting them and are not included in the resource estimate. These unclassified areas are extensive and represent obvious targets for immediate drillhole targeting with significant upside to possible production and mine life. AMDAD estimated there are approximately 1Mt of unclassified material above 4.5 g/t AuEq.

However the width of some of these veins may not be sufficient for economic mining (Figure 45).

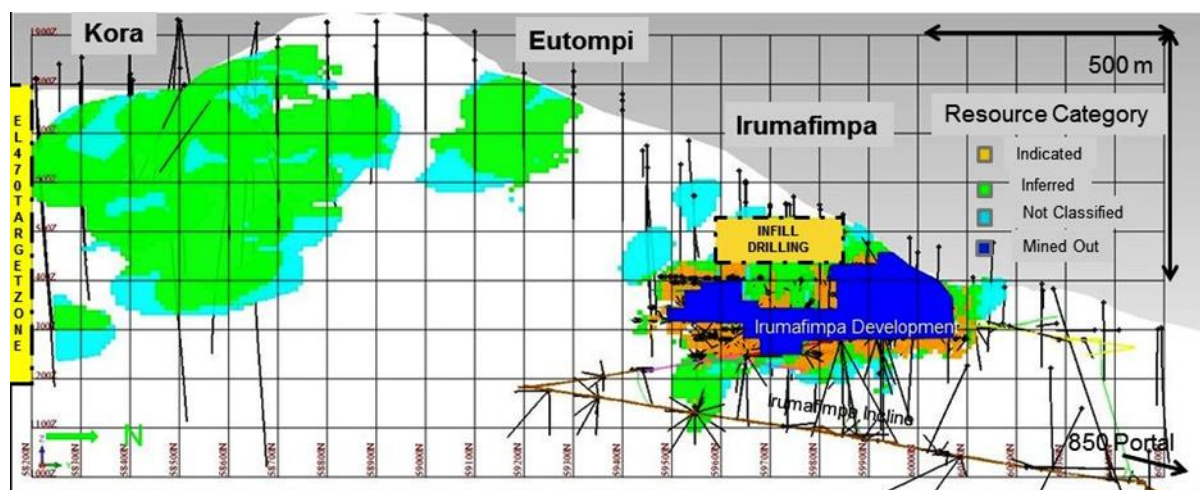


Figure 60: Kainantu Exploration Targets

- The parallel lodes on ML150, the Judd and Karempa in particular, have been outlined at surface showing similar widths and grades but have had little drill testing. The Judd vein is located 200m east of Kora on ML150. Holes designed to specifically target the Judd lode have the potential to yield resources within close proximity to the immediate mine environment. Diamond drilling has now commenced to test the Judd veins from underground.

A preliminary ranking and prioritisation allocation for vein targets within ML150 is shown in Table 52. A more comprehensive listing including both vein and porphyry targets is presented in Section 25 of the “Independent Technical Report, Resource Estimate and Summary of Mine Facilities, Kainantu Project, Papua New Guinea” by Nolidan Mineral Consultants, Author Anthony Woodward, April 15, 2016 which is filed on SEDAR.

Table 52. ML150 Exploration Prospect Ranking Vein Targets

Prospect	Style	Lease	Rank	Resource	Target Size	Access	Infrastructure	Stage
NARROW VEIN TARGETS								
Irumafimpa (in-mine)	Alkalic Vein ± Sulfidic Vein	ML150	1-1	Y	M	1	In place	DD
Irumafimpa (near mine)	Alkalic Vein ± Sulfidic Vein	ML150	1-2	N	S	1	In place	AE
Kora	Sulfidic Vein ± Alkalic Vein	ML150	1-3	Y	L	1	<1km	AE
Eutompi	Sulfidic Vein ± Alkalic Vein	ML150	1-4	Y	L	1	<1km	AE
Judd	Alkalic Vein	ML150	1-5	Y	M	1	<1km	DT
Karempa	Alkalic Vein	ML150/EL 470	1-6	N	M	1	<1km	DT

- Resource: Y=Yes, resource available; N=No, No resource available. (historic) = not verified by Qualified Person.
- Target Size: S=small; M=medium; L=large; U=unknown, P=porphyry
- Access: 1=Ready access; 2=variably available; 3=variably challenging; 4=challenging
- Stage: DD=delineation development and drilling; AE=advanced exploration; DT=drill testing; TD=target delineation

25.2 SCOPING STUDY RESULTS

The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

It should be noted that the mine plan and scoping studies prepared by AMAD for the Irumafimpa and Kora deposits are not based on Ore Reserves. The estimates of tonnes and grade reported and scheduled in both the Irumafimpa and Kora Scoping Studies do not constitute an Ore Reserve because:-

- Most of the resource estimate on which the tonnes and grade are based on are at too low a level of confidence to allow conversion to Ore Reserves.
- There is insufficient geotechnical information to be confident in development and extraction design parameters and costs and the mine plan can only be considered conceptual.
- Limited metallurgical testwork has been completed for the copper-gold mineralization and further work will be required to confirm the processing cost and recovery assumptions.

Non-mining economic and processing parameters assumed and referred to in the studies are conceptual. They were applied for the purpose of identifying the part of the Resource that notionally may be economic, in order to prepare conceptual extraction designs. Schedules are based on conceptual development and stoping quantities and not practical designs. Cashflow schedules are based on these assumed parameters. They should be treated with caution, and they should not be interpreted as a measure of the value of the deposit.

25.2.1 Irumafimpa

The Irumafimpa preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

Key estimates from the Irumafimpa Mine Plan prepared by AMDAD are:

- Over the 3 years of the mine plan treatment of 0.49Mt tonnes at 8.4 g/t Au, 5.8 g/t Ag, 0.11%Cu would generate a net cashflow of USD \$56 million.
- Over the 8 years of the Mine Life treatment of 1.40Mt tonnes at 8.2 g/t Au, 5.8 g/t Ag, 0.19%Cu would generate a net cashflow of USD \$153 million

25.2.2 Kora

The Kora preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

Key estimates from the Kora Mine Scoping Study prepared by AMDAD are:

- Over a 9 year operating life the plant would treat 3.2 Million tonnes averaging 7.1 g/t Au, 25 g/t Ag and 1.7% Cu (9.3 g/t Au Eq*).
- This would generate an estimated positive cash flow of US\$537 million using current metal prices if 15m levels are used in mining. If 25m mining levels are used then net cashflows are estimated as US\$558 million. This cashflow includes conceptual allowances for capital.

- Production of an estimated average of 108,000 Au Eq* ozs per annum over an 8 year period from Year 2 through to Year 9.
- An estimated Pre-tax NPV of US\$415 Million for 25m mining levels; or US\$397 Million for 15m levels; using current metal prices, exchange rate and a 5% discount rate;
- An estimated After-Tax NPV of US\$329 Million for 25m mining levels; or US\$316 Million for 15m levels; using current metal prices, exchange rate and a 5% discount rate;
- Initial Capital Cost is estimated to be US\$13.8 Million, including the US\$3.3 million for the plant upgrade identified in the Mincore Scoping Study, but excluding the proposed Kora exploration inclines and diamond drilling. Sustaining Capital Cost is estimated to be a further US\$64 million spent over the life of the Kora mining for 25m mining levels or US\$83 for 15m mining levels.
- Operating Cost per tonne is estimated to be US\$125/tonne for 25m mining levels or US\$126/tonne for 15m mining levels.
- Excluding Initial Capital Expenditure of US\$14M, Cash Cost is estimated to be US\$547/oz Au Eq (inclusive of a 2.5% NSR) and AISC of US\$619/oz Au Eq for 25m mining levels; or US\$549/oz Au Eq (inclusive of a 2.5% NSR) and AISC of US\$644/oz Au Eq for 15m mining levels.

Current Metal Prices used were: Au – US\$1,300/oz; Ag – US\$18/oz; Cu – US\$4,800/tonne.

*Au Eq – calculated on above Metal Prices.

25.2.3 Treatment Plant Upgrade

Key conclusions from the study by Mincore on requirements for upgrading the treatment plant to 400,000tpa are:

- There is sufficient installed crushing capacity, however the grinding mill power is limited to grind 50tph to P80 of 106 µm and requires further investigation.
- Additional flotation capacity is required to achieve acceptable residence times for each cell and stage. There is sufficient space to install additional cells if future testwork identifies a requirement for longer residence time.
- The existing concentrate thickener and filter is adequate for 400,000tpa Kora feed averaging 1.7% copper.
- The existing tailings line is adequate but a full pump upgrade will be required.
- Construction time for the plant upgrade was estimated as 10 months.

25.3 RISK ASSESSMENT

Key Risks to the success of the Kainantu project are considered to be:

- The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.
- The Resource model is mostly inferred because of drill spacing at Kora and limited confidence in underground sampling results from Irumafimpa. Reliance on historical data; the effect of poor core recovery on reliability of gold values, and possible inaccuracies in density determination are also considered risk factors.

- It was not possible to assess the validity of historical Reserve or Resource Models due to the inability to produce reconciliations. Further drilling is required to improve confidence in existing resources (upgrade to indicated and measured) and allow conversion to reserves
- Possible breakdown in government and community relations.
- Failure to commence mining operations on the Kora deposit by 30 June 2018 could lead to cancellation of ML150.
- Mining cost estimations may be in error and need to be refined using actual costs from Irumafimpa once operations are at a steady state
- Inadequate water for the expanded treatment plant
- Power demand to meet the 400,000 tpa target may be in excess of available supply
- Poor geotechnical conditions may increase requirement for concrete and structural earthworks
- Cost overruns due to design changes and delays due to slow equipment procurement and wet weather

26 RECOMMENDATIONS

26.1 EXPLORATION

Drilling should concentrate on infill drilling of current resources and extensions to veins within ML 150. Infill drilling has commenced from existing underground workings at Irumafimpa.

26.2 MINE

The estimated costs used in producing the preliminary mine plans and scoping studies for mining of the Irumafimpa and Kora gold deposits need further refinement using actual costs from Irumafimpa once operations reach a steady state.

Geotechnical studies of the mine workings need to be advanced to determine ground conditions and support requirements for development within waste and the mineralised veins.

The position and condition of existing development and stope workings at Irumafimpa needs to be confirmed.

Stope stability analysis is required to guide the selection of level interval (15m or 18m) and stope strike lengths suitable for the next stage of Kora mine design.

Groundwater conditions need to be investigated.

More detailed ventilation planning is required including analysis of ventilation options including VentSim modelling of airways to determine airflows, pressures, air power and fan specifications. Vent rise paths will need geotechnical investigations.

The feasibility of raiseboring >500m long holes from surface has to be investigated considering the implications, timing, and costs involved

Development profiles for the Kora incline and lateral access development require further analysis in relation to materials handling requirements. More analysis to reduce initial waste development is recommended.

The source and cost of any surface waste rock sources should be investigated and the various cement backfill options for Kora should be reviewed.

26.3 TREATMENT PLANT

Further metallurgical testwork is required prior to process design on the expanded treatment plant. Operating and capital cost estimates for the expanded plant need to be updated.

For and on behalf of Nolidan Mining Consultants

Anthony Woodward BSc Hons., M.Sc., MAIG

For and on behalf of AMDAD

Christopher Desoe BE (Min)(Hons), FAusIMM, RPEQ, MMICA

For and on behalf of Mincore

Lisa J Park GAICD FAusIMM.

Effective Date: 2nd March 2017

27 REFERENCES

- AMC Consultants, 2015. Update of Kainantu mine plan and technical discussion. 07 April 2015.
- AMC Consultants, 2015. Kainantu Mine Plan (Irumafimpa Orebody) for K92 Inc. 21 July 2015.
- AMDAD, 2016a. Kainantu Irumafimpa Deposit. 3 Year Mine Plan. 29 April 2016.
- AMDAD, 2016b. Kainantu Kora Deposit. Mining Scoping Study. 27 September 2016.
- AMMTEC, 2009. Metallurgical Testwork conducted upon Samples of Ore from Kora Kainantu Copper and Gold Deposit for Barrick Gold of Australia Limited, Report No. A11713. May 2009. Unpublished Consultants Report.
- Barrick, 2014. Kainantu Mine Project Proposals For Development - Tenure Extension Application 2014 (ML150, LMP78, ME80 & ME81 Tenures). Barrick report submitted for Lease Renewal.
- Barrick, 2008. Barrick Australia Pacific Technical Services Desktop Evaluation of the Kainantu Project. Australia Pacific RBU Technical Services. March 2008. Barrick internal report.
- Barrick, 2010. Project 9010 Kainantu Project ML150 Technical Report For The Period January 2008 – February 2011. Barrick (PNG Exploration) Limited. Barrick internal report.
- Barrick, 2010. Kainantu Mine 2010 Closure Provision Review. November 2010.
- Barrick, 2012. Kainantu Project Divestment Information Memorandum. Barrick internal Memo.
- Blenkinsop, T., 2005. Structural Geology Presentation – Irumafimpa-Kora. Unpublished Consultants Presentation.
- Bond, R., Dobe, J., & Fallon, M., 2009. Kora Geology & Estimate of Mineralized Inventory, Draft, April 2009. Barrick internal report.
- Bullock, R.L. & Hustrulid, W.A., Editors. 2001. Underground Mining Methods : Engineering Fundamentals and International Case Studies. SME 2001, 718p.
- Butcher, R. & Fairburn, G., 2010. Kora Underground Desktop Mining Study, September 2010. Barrick internal report.
- Clark, A.M., 2007. Technical Review of Mine Geological Systems of Kainantu Mine, CdeK Geological & Mining Services, July 2007. Unpublished Consultants Report.
- Corbett, G., 2009. Comments on Au-Cu exploration project at the Oro project and environs, Papua New Guinea.
- Corbett, G., & Leach, T. M., 1997. Southwest Pacific rim gold-copper systems: Structure, alteration and mineralization, Short course manual.
- Cox, D.P., and Bagby, W.C., 1986. Descriptive model of Au-Ag-Te veins, in Cox, D.P., and Singer, D.A., eds., Mineral deposit models: U.S. Geological Survey Bulletin 1693, p. 124.
- Espi et al. 2006. Geology Wall-Rock Alteration and Vein Paragenesis of the Bilimoia Gold Deposit Kainantu Metallogenic Region Papua New Guinea. Resource Geology, v.57 No.3: 249-268
- Fallon, M., 2010. Kora – Au-Cu-Ag Deposit, Geology & Estimate of Mineralized Inventory March 2010. Barrick internal report.
- Glacken, I., Blackney, P., Grey, D., and Fogden, N., 2014. Resource Estimation in Folded Deposits – A review of Practice and Case Studies. Good Practice in Resource and Ore Reserve Estimation, Australian Institute of Mining and Metallurgy Monograph 30, p 351-361.
- Gauthier, L., 2008b. Review of the Kainantu prospects 7 May 2008 File Code: 9012 – 501. Barrick internal report.

- Gauthier, L., and Pridmore, C., 2007. Kainantu (Highlands Pacific) Exploration and Irumafimpa resource review. Internal Barrick memorandum.
- Gauthier, L., 2008. Maniape prospect potential review. 13 March 2008 File Code: 9012 – 501. Barrick internal report.
- HGL, 1992. Highlands Gold Annual Report 1992 (number 964).
- HPL, 2003. Highlands Pacific Group Kainantu Gold Project Definitive Feasibility Study
- HPL, 2006. Highlands Pacific Group Processing Summary- Commissioning to August 2006
- Jackson, T and K.P.Green, 2015. Fraser Institute Annual Survey of Mining Companies, 2014. Fraser Institute. February 2015.
- Jenkins, 2008. Barrick Gold Australia Kainantu Gold Mine Geotechnical Assessment Final Report March 2008. Dempers & Seymour Pty Ltd Geotechnical and Mining Consultants. Unpublished Consultants Report.
- JKTech, 2007. Flotation Circuit Optimisation at Highlands Pacific – Kainantu. Unpublished Consultants Report.
- Kramer Ausenco, 2014. Kainantu O&M Readiness. Ausenco Services Pty Ltd. December 2014.
- Lewins, J., 2016. Proposed Bulk Sampling at Irumafimpa. K92 Ltd Memo. 25 February 2016.
- Logan, R., 2006. Eutompi 2005 Drilling, Ross Logan and Associates, August 2006. Unpublished Consultants Report.
- MRA, 2016. Application for change of name from Barrick (Kainantu) Limited to K92Mining Limited. Letter to O'Briens Lawyers from Mineral Resources Authority. 17 March 2016.
- Mincore, 2015. Proposal for Process Plant Re-Start and Scrubber. Mincore Pty Ltd. 11 February 2015.
- Mincore, 2016. K92 Mining Limited. 400KTPA Process Plant Upgrade Study. 30 August 2016.
- Mining Associates, 2006. Highlands Pacific Limited Kainantu Project Resource Update October 2006 - Mining Associates Pty Ltd (MA611). Unpublished Consultants Report.
- O'Brien, S., 2014. Letter to the Directors K92 Resources Corp dated 21 November 2014. O'Briens Lawyers.
- Peters, T., 2008. Barrick Gold of Australia Ltd Kainantu Gold Mine Mining Assessment and Ventilation Review. Piran Mining Pty Ltd. March 2008. Unpublished Consultants Report.
- Richards, J.P., 1998. Alkalic-Type Epithermal Gold Deposits; in Metallogeny of Volcanic Arcs, B.C. Geological Survey, Short Course Notes, Open File 1998-8, Section H.
- Smith, G., and Thomas, M., 2008. Kainantu Mine Reconciliation Review, November 2008. Barrick internal memorandum.
- Smith, G., 2008. Desktop Evaluation of Kainantu Project, March 2008. Barrick internal report.
- Smith, G., 2008. Kainantu Cut Off Grade Calculations, January 2008. Barrick internal memorandum.
- SRK, 2006. Kainantu Gold Mine Review. Report Prepared for Highlands Pacific Limited. SRK Project Number HPL301. December 2006. Unpublished Consultants Report.
- Tingey, R.J. and Grainger, D.J., 1976. Markham-Papua New Guinea, 1:250,000 Geological Series with Explanatory Notes.
- Thomas, M., 2007. Irumafimpa Resource Estimate, December 2007. Barrick internal report.

Thomas, M., 2010. Irumafimpa & Kora Resource Estimation & Mining Evaluation, October 2010. Barrick internal report.

Thomas, M., 2011. Kora Resource Evaluation Update, December 2011. Barrick internal presentation.

Thomas, M., 2012. Kora Financial Model V12.1 update.xlsx, November 2012. Barrick internal Excel worksheet.

Thomas, M., 2012. Revised Kora Block Model Update, October 2012. Barrick internal memorandum.

Tosdal, R., 2012. Observations and thoughts regarding the Tankuanan, Timpa, A1, and Breccia Hill prospects, Kaimun (Kainantu) area, Papua New Guinea. August 13, 2012. Unpublished Consultants Report.

Williamson, A., & Hancock, G., 2005. Geology and Mineral Potential of Papua New Guinea. Papua New Guinea Department of Mining, 152p.

Woodward, A.J., 2015. Independent Technical Report and Resource Estimate, Kainantu Project, Papua New Guinea. 06 March 2015.

Woodward, A.J., 2015. Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea. 01 May 2015.

Woodward, A.J., 2016. Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea. 15 April 2016.

28 CERTIFICATE OF QUALIFIED PERSON

ANTHONY JAMES WOODWARD

I, Anthony James Woodward hereby certify that:

I am a Consulting Geologist and Professional Geoscientist residing at 14 Carlia Street, Wynnum West, Queensland 4178, Australia (Telephone +61-7-3396 9584). I am independent of the issuer as independence is described in Section 1.5 of NI 43-101.

I graduated from the University of Nottingham, UK in 1968 with a B.Sc. (Hons) in Geology and from James Cook University, Townsville, Australia in 1976 with a M.Sc in Exploration and Mining Geology.

I have over 35 years' experience in the minerals industry as a Geologist in the fields of mineral exploration, mine geology and mineral resource estimation. I have had senior exploration roles with Buka Gold, Niugini Mining, Eltin Minerals and Oakbridge Ltd. I have conducted evaluation of advanced exploration and mining projects in Australia, Brazil, Fiji, Indonesia, Kazakhstan, New Zealand, and Turkey. I worked as Technical Services Manager and Chief Geologist at the Vatukoula Gold Mine in Fiji (Emperor Mines Ltd) from 1995 to 2005 and as Technical Services Manager for Anvil Mining Congo at the Kinsevere copper mine, DRC from 2007 to 2008. At these mines I was responsible for mine and exploration geology, surveying, mine planning, environment, drilling, and assay laboratory. At both operations I spent time as Acting General Manager of Operations. In this role I supervised multiple disciplines and integrated their work into operational mine plans. Most recently, I have been an exploration consultant in the Philippines involved with total exploration program management on tenements prospective for both epithermal gold-molybdenum and porphyry copper-gold deposits including regional exploration targeting through to deposit resource drilling.

Applicable to the Kainantu Project is my extensive experience in mineral deposits in volcanic terrains, specifically the Vatukoula and Tuvatu epithermal gold deposits in Fiji. I have also worked on epithermal/hydrothermal and porphyry-style mineralization in similar environments in Papua New Guinea, Fiji, New Zealand, Philippines, Indonesia, Brazil and Turkey as well as Australia.

I am a Member of the Australian Institute of Geoscientists (Member No. 2668).

For the purposes of the Technical Report entitled: "INDEPENDENT TECHNICAL REPORT, MINERAL RESOURCE ESTIMATE UPDATE AND PRELIMINARY ECONOMIC ASSESSMENT OF IRUMAFIMPA AND KORA GOLD DEPOSITS, KAINANTU PROJECT, PAPUA NEW GUINEA", 2nd March 2017, of which I am a part author and responsible person. I am a Qualified Person as defined in National Instrument 43-101 ("the Rule").

I am responsible for the preparation of Sections 1 to 15, 18 to 20, and 23 to 27 of the technical report.

I visited the Kainantu Project on the 12th and 13th of November, 2014 and 21st to 25th November, 2016 and have had no prior involvement with the Kainantu property.

I have read the Rule and this technical report is prepared in compliance with its provisions. I have read the definition of "qualified person" set out in the Rule and certify that by reason of my education, affiliation with a professional association (as defined in the Rule) and past relevant work experience, I fulfil the requirement to be a "qualified person" for the purposes of the Rule.

To the best of my knowledge, information and belief the technical report contains all scientific and technical information that is required to be disclosed in order to make this report not misleading.

I have no direct or indirect interest in the properties which are the subject of this report and I have had no prior involvement with the Property. I do not hold, directly or indirectly, any shares in K92ML, K92PNG, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92PNG, K92 Holdings, K92, and, the Property, as independence is described by Section 1.5 of NI 43-101.

I will receive only normal consulting fees for the preparation of this report.

Dated at Brisbane this 2nd March 2017.

Respectfully submitted

(signed) "*Anthony James Woodward*"

Anthony James Woodward, BSc Hons, M.Sc., MAIG
Qualified Person

CERTIFICATE OF QUALIFIED PERSON

CHRISTOPHER GABOR DESOE

I, Christopher Gabor Desoe of Brisbane, Australia do hereby certify:

1. That I am Manager - Mining with Australian Mine Design and Development Pty Ltd with a business address at Level 4, 46 Edward Street Brisbane, Queensland 4000 Australia.
2. This certificate applies to the technical report titled Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment of Irumafimpa and Kora Gold Deposits, Kainantu Project, Papua New Guinea dated 2 March 2017 with an effective date of 2 March 2017 (the "Technical Report").
3. That I am a Fellow and Chartered Professional (Mining) of the Australasian Institute of Mining and Metallurgy, and a member of the Mining Industry Consultants Association.
4. That I graduated from the University of New South Wales, Australia, in 1983, with a B.E. (Min)(Hons).
5. I have 33 years of experience in the mining industry of which more than 15 years is in hard rock underground mining. Applicable to the Kainantu Project is my considerable experience in narrow underground operations and planning including Imwauna Gold Project in PNG, a confidential narrow vein gold project in Columbia, Ban Phuc Nickel Mine in Vietnam, the Mount Colin, Reward, West 45 and Selwyn 257 Copper Gold Mines, Merlin Molybdenum Rhenium deposit and Mount Isa Lead Zinc Mine all in North Queensland, Australia, and Renison Tin Mine in Tasmania, Australia.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that I am a "qualified person" for the purposes of NI 43-101.
7. That I Christopher Gabor Desoe last visited the Kainantu mine site during 2016 for eight days.
8. I am responsible for section 16 of the Technical Report, and for the conceptual costs and cashflows presented in sections 21 and 22 based on economic and processing assumptions provided by K92ML.
9. I am independent of K92ML as described in Section 1.5 of NI 43-101.
10. I have had no involvement with the property that is subject to the Technical Report prior to 2016.
11. I, or any affiliated entity of mine, have not earned the majority of our income during the preceding three years from K92ML, or any associated or affiliated companies.
12. I have no interest in the subject property, either directly or indirectly.
13. I, or any affiliated entity of mine, do not own, directly or indirectly, nor expect to receive, any interest in the properties or securities of K92ML or any associated or affiliated companies.
14. I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
15. That as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 2nd day of March, 2017 at Brisbane, Queensland, Australia.

Christopher Gabor Desoe, BE (Min)(Hons), FAusIMM, RPEQ, MMICA
Manager – Mining, Australian Mine Design & Development Pty Ltd

CERTIFICATE OF QUALIFIED PERSON

LISA JANE PARK

I hereby state:

1. My name is Lisa Jane Park and I am the principal of the firm Process Engineering Options of 13 William St Hawthorn, VIC 3122, Australia.
2. I am a practising process engineer registered as a Fellow with the Australasian Institute of Mining and Metallurgy. My membership number is 112751.
3. I graduated with a B Eng degree in Chemical Engineering in 1994 from the University of Melbourne, Australia. I also hold a Master degree in Applied Finance from the Queensland University of Technology, Australia.
4. I have practised my profession for 22 years, since 1994. I have experience in project development, operations and construction. My previous experience in copper-gold projects includes the Silangan project (Philippines), Pebble project (Alaska, USA), Waisoi project (Fiji), Didipio project (Philippines) and many other projects in various capacities over the years.
5. I am a "qualified person" as that term is defined in National Instrument NI 43-101 (Standards of Disclosure for Mining Studies) (the "Instrument").
6. I have not visited the K92 Mining Ltd project area as at 02 March 2017.
7. I assisted with the K92 Mining Ltd scrubber project, for Mincore Pty Ltd in 2016.
8. I have assisted in the preparation of the study by Mincore Pty Ltd dated 5 February 2017.
9. For the purposes of the Technical Report entitled "Independent Technical Report, Mineral Resource Estimate update and preliminary economic assessment of Irumafimpa and Kora gold deposits, Kainantu project, Papua New Guinea" (the "Report"), prepared by Nolidan Mineral Consultants, Australian Mine Design and Development, and Mincore, effective date 02 March 2017, I am a part author and responsible person.
10. Specifically, I am responsible for the preparation of section 17 Recovery methods; section 25.2.3 Treatment plant upgrade; and section 26.3 Treatment plant.
11. I am a Qualified Person as defined in National Instrument 43-101 ("the Rule").
12. At the effective date of the technical report, to the best of my knowledge, information, and belief, the above parts contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
13. I am Independent of the K92 Mining Ltd project pursuant to Section 1.5 of the Instrument.
14. I have read the National Instrument and Form 43-101 F1 (the "Form") and parts of the study for which I have assisted to ensure it has been prepared in compliance with the Instrument and the Form.
15. I do not have nor do I expect to receive a direct or indirect interest in the K92 Mining Ltd project, and I do not beneficially own, directly or indirectly, securities of K92 Mining Ltd.

Dated at Perth, Western Australia, on 2nd March 2017.

Signed

Lisa Jane Park
Principal
Process Engineering Options

29 APPENDIX 1: GLOSSARY OF TECHNICAL TERMS AND ABBREVIATIONS

This glossary comprises a general list of common technical terms that are typically used by geologists. The list has been edited to conform in general to actual usage in the body of this report. However, the inclusion of a technical term in this glossary does not necessarily mean that it appears in the body of this report, and no imputation should be drawn. Investors should refer to more comprehensive dictionaries of geology in printed form or available in the internet for a complete glossary.

“2D”	Two dimensional space, typically Y and Z planes
“3D”	Three dimensional space, Y, X, Z planes
“200 mesh”	the number of openings (200) in one linear inch of screen mesh (200 mesh approximately equals 75 microns)
“AAS”	Atomic Absorption Spectroscopy
“Ag”	chemical symbol for silver
“Au”	chemical symbol for gold
“AuEq”	Gold equivalent, assumptions include metal prices and assumed metallurgical recoveries.
“BLA”	Billimoian Landowners Association
BSc (Hons)	Bachelor of Science with Honours
“block model”	A block model is a computer based representation of a deposit in which geological zones are defined and filled with blocks which are assigned estimated values of grade and other attributes. The purpose of the block model (BM) is to associate grades with the volume model. The blocks in the BM are basically cubes with the size defined according to certain parameters.
“bulk density” “BD”	The dry in-situ tonnage factor used to convert volumes to tonnage. Bulk density testwork is carried out on site and is relatively comprehensive, although samples of the more friable and broken portions of the mineralized zones are often unable to be measured with any degree of confidence, therefore caution is used when using the data.
“°C”	Degrees Celsius
“Cu”	Chemical symbol for copper
“DDH” “diamond drilling, diamond core”	Rotary drilling technique using diamond set or impregnated bits, to cut a solid, continuous core sample of the rock. The core sample is retrieved to the surface, in a core barrel, by a wireline.
“down-hole survey”	Drillhole deviation as surveyed down-hole by using a conventional single-shot camera and readings taken at regular depth intervals, usually every 50 metres.
“drill-hole database”	The drilling, surveying, geological and analyses database is produced by qualified personnel and is compiled, validated and maintained in digital and hardcopy formats..
“EL”	Exploration Lease
“FA”	Fire Assay
“g.m”	Grams x metres, metal accumulations across the width of the vein
“grade cap, also called top cut”	The maximum value assigned to individual informing sample composites to reduce bias in the resource estimate. They are capped to prevent over estimation of the total resource as they exert an undue statistical weight. Capped samples may represent “outliers” or a small high-grade portion that is volumetrically too small to be separately domained.
“g/t”	grams per tonne, equivalent to parts per million
“g/t Au”	grams of gold per tonne
“HGL”	Highlands Gold Limited
“HPL”	Highlands Pacific Limited
“ID” “inverse distance estimation”	It asserts that samples closer to the point of estimation are more likely to be similar to the sample at the estimation point than samples further away. Samples closer to the point of estimation are collected and weighted according to the inverse of their separation from the point of estimation, so samples closer to the point of estimation receive a higher weight than samples further away.

	The inverse distance weights can also be raised to a power, generally 2 (also called inverse distance squared, ID2). The higher the power, the more weight is assigned to the closer value. A power of 2 was used in the estimate used for comparison with the OK estimates.
"Inferred Resource"	An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to an Ore Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
"Indicated Resource"	An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes, and is sufficient to assume geological and grade (or quality) continuity between points of observation where data and samples are gathered. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Ore Reserve.
"IRG" or "IRGC"	Intrusion Related Gold or Intrusion Related Gold Copper
"JORC"	The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 (the 'JORC Code' or 'the Code') sets out minimum standards, recommendations and guidelines for Public Reporting in Australasia of Exploration Results, Mineral Resources and Ore Reserves. The Code is a required minimum standard for Public Reporting b. JORC also recommends its adoption as a minimum standard for other reporting. Companies are encouraged to provide information in their Public Reports that is as comprehensive as possible. The definitions in the JORC Code are either identical to, or not materially different from, those similar codes, guidelines and standards published and adopted by the relevant professional bodies in Australia, Canada, South Africa, USA, UK, Ireland and many countries in Europe.
"kriging neighbourhood analysis, or KNA"	The methodology for quantitatively assessing the suitability of a kriging neighbourhood involves some simple tests. It has been argued that KNA is a mandatory step in setting up any kriging estimate. Kriging is commonly described as a "minimum variance estimator" but this is only true when the block size and neighbourhood are properly defined. The objective of KNA is to determine the combination of search neighbourhood and block size that will result in conditional unbiasedness.
"km"	Kilometre Unit of Length = 1000 metres. km ² unit of area = 1km x 1 km
"kVa"	1000 volt-amperes
"lb"	Avoirdupois pound (= 453.59237 grams). Mlb = million avoirdupois pounds
"micron (μ)"	Unit of length (= one thousandth of a millimetre or one millionth of a metre).
"mm"	Millimetre (=1/1000 metre)
"LMP"	licence for mining purposes
"LOM"	Life of Mine
"LTC"	Land Titles Commission
"m"	Metric Metre
MAusIMM(CP)	Member of The Australian Institute of Mining and Metallurgists (Certified Professional)
MAIG	Member of The Australian Institute of Geoscientists
"Measured Resource"	A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes, and is sufficient to confirm geological and grade (or quality) continuity

	between points of observation where data and samples are gathered. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Ore Reserve or under certain circumstances to a Probable Ore Reserve.
“Mineral Resource”	A ‘Mineral Resource’ is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade (or quality), and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade (or quality), continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.
“ME”	Mining Easements
“ML”	Mining Lease
“MOA”	Memorandum of Agreement
“MRA”	Mineral Resources Authority of Papua New Guinea
“NN” “nearest neighbour estimation”	Nearest Neighbour assigns values to blocks in the model by assigning the values from the nearest sample point to the block attribute of interest.
“OH&S”	Occupational Health and Safety
“OK” “ordinary Kriging estimation”	Kriging is an inverse distance weighting technique where weights are selected via the variogram according to the samples distance and direction from the point of estimation. The weights are not only derived from the distance between samples and the block to be estimated, but also the distance between the samples themselves. This tends to give much lower weights to individual samples in an area where the samples are clustered. OK is known as the “best linear unbiased estimator. The kriging estimates are controlled by the variogram parameters. The variogram model parameters are interpreted from the data while the search parameters are optimised during kriging neighbourhood analysis.
“oz”	Troy ounce (= 31.103477 grams). Moz = million troy ounces
“PGK”	Papua New Guinea Currency, Kina.
“pH”	measure of the acidity or basicity of an aqueous solution (scale 1 to 14)
“PhD”	Doctorate of Philosophy
“PNG”	Papua New Guinea
“Portal”	Opening/access to the underground Mine, Adit
“QA/QC”	Quality Assurance (“QA”) concerns the establishment of measurement systems and procedures to provide adequate confidence that quality is adhered to. Quality Control (“QC”) is one aspect of QA and refers to the use of control checks of the measurements to ensure the systems are working as planned.
“RC drilling”	Reverse Circulation drilling. A method of rotary drilling in which the sample is returned to the surface, using compressed air, inside the inner-tube of the drill-rod. A face-sampling hammer is used to penetrate the rock and provide crushed and pulverised sample to the surface without contamination.
“ROM”	Run of Mine, usually referring to an ore stockpile near the crusher
“survey”	Comprehensive surveying of drillhole positions, topography, and other cadastral features is carried out by the Company’s surveyors using ‘total station’ instruments and independently verified on a regular basis. Locations are stored in both local drill grid and UTM coordinates.
“Stoping”	An underground excavation made by the mining of ore from steeply inclined or vertical veins
“t”	Metric Tonne (= 1 million grams) “kt” = thousand tonnes
“te”	Chemical symbol for tellurium
“t/h”	Tonnes per hour
t/m ³	Tonnes per metre cubed (density units)
“TSF”	Tails Storage Facility
“unfolded space”	Undulating 3D veins projected onto a 2D plane.

"variogram"	The variogram (or more accurately the Semi-variogram) is a method of displaying and modelling the difference in grade between two samples separated by a distance h , called the "lag" distance. It provides the mathematical model of variation with distance upon which the Kriging estimation method is based.
"wireframe"	This is created by using triangulation to produce an isometric projection of, for example, a rock type, mineralization envelope or an underground stope. Volumes can be determined directly of each solid.