Lycopodium

Endeavour Mining Corporation

Kalana Gold Project, Republic of Mali

Pre-Feasibility Study

National Instrument 43-101 Technical Report

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

In August 2020 Endeavour Mining Corporation (Endeavour) engaged various consultants to undertake a Pre-Feasibility Study (PFS) for their Kalana Project, located approximately 250 km south of the capital of Mali, Bamako, with access by sealed highway. The deposit was discovered by Malian survey companies between 1962 and 1982, with a small underground mine commencing in 1985. By August 1991, when funding from the Soviet Union stopped, this mine produced 81,000 oz of gold from 227,000 tonnes of ore at an average grade of 13 Au g/t. Somika was granted the Permit and between 2004 and 2015 produced 185,116 oz of gold from 628,615 tonnes at an average grade of 11.6 Au g/t. Underground and historic open pit mine workings, a gravity processing plant, offices, stores and accommodation adjacent to Kalana town remain at the site.

The climate is sub-tropical with a wet season from May to September, with most rain (250 mm) falling in August. The Project site is within one kilometre of Kalana town (population 14,000), which has electric power from the national grid, a potable water supply system, medical and schooling facilities. Water is available from underground sources for mining operations with excess water from the underground workings discharged. Water for community consumption is supplied by community pumps and boreholes.

1.2 Geology and Ore Resources

The Kalana Project comprises two orogenic gold deposits, Kalana and Kalanako, and two tailing storage facilities (TSFs).

The Kalana Project area is covered by an extensive lateritic profile, characteristic of the Guinea Savannah climatic zone, obscuring the bedrock. The depth of weathering ranges from 30 m to 130 m. The Kalana and Kalanako gold deposits are hosted by volcano-sedimentary rocks of the lower part of the Upper Birimian Group.

1.2.1 Kalana Deposit Geology

The Kalana deposit is composed of mesothermal gold mineralisation hosted mainly in quartz veins following shallow dipping and vertical dykes adjacent to small intrusions emplaced in a sub-volcanic environment. The Kalana deposit can be subdivided into four geographical areas.

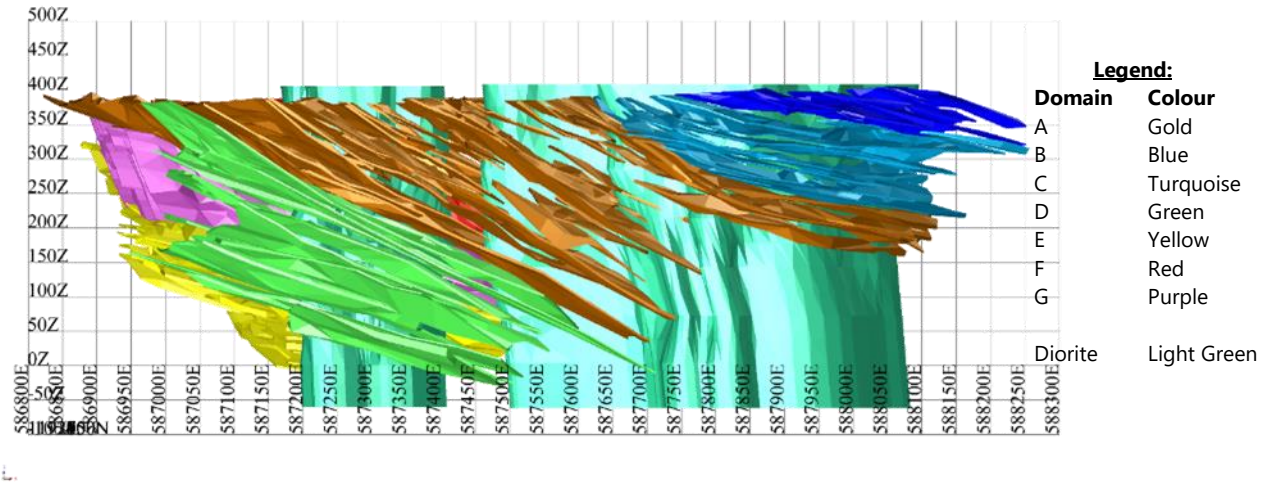
- Dynamite Trend – Located northeast of the Kalana deposit. Does not host significant gold mineralisation.
- Kalana North – Hosts 1st, 2nd and 3rd Order Veins.
- Kalana Southwest – Hosts 1st Order Veins only and the majority of the historical underground workings.
- Kalana Southeast - Hosts 1st Order Veins only.

The Kalana gold mineralisation occurs within stacked quartz veins and stockworks hosted by the Birimian age metasediments and the intrusive diorite stock, with four types of gold mineralisation:

- 1st Order Veins - Flat dipping (an average dip of less than 25°) quartz veins and vein sets in the metasediments and diorite stock.
- 2nd Order Veins – Dipping (typically 45° south) quartz veins and vein zones striking N60 to N40 in Kalana North and Kalana Northeast.
- 3rd Order Veins – Sub-vertical thin quartz veins in Kalana North only. Very minor.
- Stockwork mineralisation at the contact of the diorite.

The majority of the gold in the quartz veins is present as free gold with the gold commonly occurring as grains and small nuggets. Fine grained gold is also associated with sulphides in the quartz veins, as well as in the meta-sedimentary and diorite wall rocks immediately adjacent to the quartz veins. The long section N-S in Figure 1.2.1 indicates the complexity of the veins within the deposit in relation to the intrusives.

Figure 1.2.1 Kalana Deposit Mineralised Veins by Domain (Long Section N-S)



1.2.2 Kalanako Deposit Geology

Kalanako is located approximately three kilometres northeast of Kalana. Several mineralised trends have been established that form a northwest-southeast explored corridor of 1,500 m by 250 m. The sub-parallel mineralised zones are typically five to 15 m wide, with strike lengths of 250 m to 500 m, and are steeply dipping to the east. Kalanako typically has high-grade intercepts in the oxide part of the deposit. The deposit occurs along a mineralised geophysical structure (aeromagnetic and ground IP), which remains open along strike.

The depth of saprolite and saprock is between 70 m and 130 m, much deeper than that observed at Kalana. Diamond drilling at Kalanako intersected numerous high strain zones, packets of densely laminated quartz veins with sulphides and locally highly altered and mineralised felsic intrusive rocks.

Mineralisation is associated with the felsic intrusive rocks and quartz stockworks; thirty-four felsic-associated quartz veins have been modelled, six of which account for two-thirds of the mineralised volumes (referred to as 'major veins'). The veins have an average thickness of 7.2 m down-the-hole (DTH).

1.2.3 Tailing Storage Facilities (TSFs)

Two Tailing Storage Facilities (TSFs) exist within the Kalana Project: the TSF proper (TSF) located halfway between Kalana and Kalanako, and the small TSF (Shaft No. 2 TSF) within the fence close to the plant, see Figure 1.2.2.

The TSF is located outside the Kalana UG Mine fence, 600 m northeast or approximately halfway to Kalanako. It was originally constructed in the 1990s by SOGEMORK (the Russians) as a single impoundment in the shape of an elongated horseshoe. SOGEMORK subsequently sub-divided it to retain approximately 200,000 t of underground tailings and 50,000 t of oxide tailings in separate zones.

The TSF was re-opened in 2004 with a three-metre lift, and extended in 2006, 2012 and 2016 by SOMIKA (Avnel) and closed (i.e. no new material added, not covered or rehabilitated) in December 2017 by SOMIKA (Endeavour). It is of an upstream construction, side-hill impoundment design and has a similar footprint to that created by SOGEMORK with a number of lifts.

The Shaft No. 2 TSF is located within the Kalana UG Mine fenced perimeter, to the west of the old ventilation shaft. It was the original open-pit excavated by SOGEMORK in the 1980s and was subsequently infilled with plant tailings from underground workings generated by SOMIKA (Avnel) between 2004 and 2009.

Figure 1.2.2 Kalana TSF Locations (Source: Google Earth Pro. Image date: January 2016)



1.2.4 Informing Data

Exploration has been conducted on the Kalana Project since the 1960's and a significant amount of data generated, including soil samples, geophysics, and regional exploration drill holes. Digital terrain models (DTMs) for the topographic elevations have been generated for the Project.

Up until 2013, drill samples were routinely assayed by 50 g fire assay. Between 2010 and 2013, various gold deportment studies were undertaken and LeachWELL assays were introduced in an attempt to reduce the effect of coarse gold on the accuracy and precision of the gold estimate. A systematic re-assaying programme was undertaken in three (depth constrained) phases, during which the leaching tails were systematically assayed using Fire Assay to ensure that the leaching process was complete if the LeachWELL grade was over 0.1 Au g/t. Samples from Kalanako and Dijrila were also re-assayed.

Kalana

The Kalana Resource Drill Hole Database, with an effective date of 12 July 2018, contains nearly 1,300 holes for approximately 210,000 m, see Table 1.2-1. No drilling has occurred since July 2018.

Table 1.2-1 Kalana Deposit Resource Drill Hole Database, as of 12 July 2018

Company	Campaign	No. Holes	Type	Total Metres
Ashanti-JCI	1996	1	RCDD	294
SOMIKA	2008	41	RC	1,401
*IAMGOLD & SOMIKA	2010 - 2013	567	RC	79,388
		226	DD	53,004
	Subtotal	**834**		**133,793**
SOMIKA (Avnel)	2014	17	AC-RC	1,523
	2015	137	RC	21,405
		16	RCDD	3,950
		19	DD	2,811
		9	DDTT	1,976
	Subtotal	**198**		**31,665**
SOMIKA (Endeavour)	2017 - 2018	165	RC	24,914
		77	RCDD	25,841
		4	DD	1,347
	Subtotal	**246**		**52,102**
Total		**1,279**		**217,854**
*Includes 19 "H" holes which were UG DD GC holes drilled by SOMIKA in 2010.				

Kalana Underground Mine drill hole, channel and grab samples have not been used in the grade estimate. Geological and mineralisation qualitative data have been used to aid the interpretations, as have the surveyed underground workings.

Kalanako

The Kalanako Resource Drill Hole Database, see Table 1.2-2, does not include the results of the 2019 Kalanako North Extension drilling campaign (27 holes) and reconnaissance exploration holes. It has an effective date of 7 February 2018.

Table 1.2-2 Kalanako Resource Drill Hole Database, as of 7 February 2018

Company	Campaign	No. Holes	Type	Total Metres
SOMIKA (Avnel)	2010 – 12	29	DD	7,409
		206	RC	22,145
	Dec 2016	82	RC	8,635
SOMIKA (Endeavour)	2017 – 2018	63	RC	7,842
		1	DD	189
		2	DDTT	350
Total		**383**		**46,570**

TSFs

The TSF volume has been estimated with consideration of the SOGEMORK estimated volume in 2004 by Snowden, preliminary dam design reports, the Denny Jones 2015 estimate, production plant discharge data and various topographic surveys. It is estimated to be 653,000 t as of June 2020.

The TSF was sampled in May 2004 on a 10 m by 10 m grid. A NI 43-101 (CIM) compliant Measured Resource of 234 kt at 1.9 Au g/t for 14 koz of gold was reported by Snowden in 2005.

The volume of the Shaft No. 2 TSF as of June 2020 has been modelled from drill hole logs, current topographic surfaces and historical partial survey data to be 128,000 m3. It includes the depletion of 54,000 t reprocessed in 2016 – 2017 by SOMIKA.

Both TSFs were sampled by auger in 2012, but the holes did not always reach the dam base. The results are summarised in Table 1.2-3.

Table 1.2-3 IAMGOLD TSF Auger Sampling Summary

	Shaft No. 2 TSF	TSF
Hole Name Prefix	KA-SEK-WBS	KA-SEK-WBN
No. Auger Holes	165	137
Average Depth (m)	3.6	3.5
No. Assays	587	485
Average Grade (Au g/t)	2.3	2.1
Min. Grade (Au g/t)	0.5	0.2
Max. Grade (Au g/t)	13.0	16.1

1.2.5 Resource Estimation

The grade estimation of the mineralised veins and vein packages at Kalana and Kalanako are hampered by the very coarse nature of the gold; the drill hole databases have numerous instances of barren assays with a number of visible gold grains logged in a known mineralised vein / lode. The resource estimation methodology applied uses a combination of Categorical and Ordinary Kriging.

An appraisal of downhole sample length demonstrated that over 99% of the drillhole data had sample lengths of one metre or less. Consequently, the drillhole data was composited to a target length of one metre within the constraints provided by the vein package, weathering and lithology interpretations. This composited data was used for all further analysis.

Mineralisation Distribution

The Kalana vein package (VP) and Kalanako mineralisation interpretations were compiled based on gold grade, presence of quartz, presence of visible gold and structural and lithological factors. The geometry of the Kalana VPs led to the adoption of a projection (coordinate transformation) methodology as part of all estimation processes.

Block Model Creation

A block model was created with topography, weathering, lithology, vein package and TSF wireframes for the Kalana deposit. The wireframes were represented by 10 mE by 10 mN by 5 mRL parent blocks with 5 mE by 5 mN by 1.25 mRL subcells at boundaries. A rotated block model was created for the Kalanako deposit from topography, weathering, oxidation, and mineralisation wireframes. These wireframes were represented by 5 mX by 20 mY by 10 mRL parent blocks with 1.25 mX by 5 mY by 2.5 mRL subcells at boundaries.

Density

Density measurements were taken from diamond core samples using a water displacement (Archimedes) method.

The Kalana database exhibits a distinct bimodality. The most common lower density is around 1.5 t/m3 and the most common higher density is around 2.67 t/m3. Preliminary analysis showed that lithology-based density contrasts were very small, and this led to utilising only the weathering subdivisions for further analysis.

The Kalanako database also exhibits a distinct bimodality. The most common lower density is around 1.8 t/m3 and the most common higher density is around 2.68 t/m3.

The saprock categories exhibits a greater degree of variability than the other weathering categories. This is caused by the density values within this category retaining some bimodality in their distribution, which is consistent with the mixed characteristics of this domain.

Density values were assigned to the resource models based on the interpreted weathering domain as per Table 1.2-4.

Table 1.2-4 Block Model Assigned Densities

Weathering Type	Kalana Density (t/m^3)	Kalanako Density (t/m^3)
Laterite / Mottled Zone / Regolith	1.67	1.70
Saprolite	1.64	1.76
Sap-rock	2.15	2.09
Fresh	2.68	2.64

Underground Depletion

The Kalana model has been depleted for ore extracted by historical underground mining using a stope and development wireframe model at a resolution of 5 mE by 5 mN by 1.25 mRL. Blocks identified as being inside an underground void have been coded within the block model and assigned a density value of zero.

TSF Estimation

In 2004, the SOGEMORK tailings in the TSF were evaluated by Snowden and a NI 43-101 (CIM) compliant Measured Resource of 234 kt at 1.9 Au g/t for 14 koz of gold reported (Snowden, 2005).

Endeavour estimates the TSF proper as of June 2020 to contain 653 kt at 1.8 Au g/t containing 37.3 koz Au. The Shaft No. 2 TSF is estimated by its volume (128,000 m3) multiplied by a dry bulk density (1.5 t/m3) at a grade of 1.7 Au g/t to contain 192 kt containing 10.4 koz Au.

The Total TSF Indicated Resource is estimated to be 845 kt @ 1.8 Au g/t containing 48 koz Au.

1.2.6 Mineral Resource Classification and Reporting

The Mineral Resource classification system used is consistent with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards 2014. Resources are classified into Measured, Indicated and Inferred categories based upon increasing geological confidence. Resource classification within the mineralised vein wireframes are generally based on drill hole spacing, grade continuity and overall geological continuity.

The Kalana Mineral Resource has been classified using drillhole spacing as the primary criterion (Table 1.2-5). No Measured category material was assigned, largely because of the coarse gold character of the deposit, the high nugget effect component and the relatively poor grade continuity definition provided by the drilling data. A 'reasonable prospect of eventual economic extraction' (RPEEE) limit was developed using pit optimisation methods to define a pit shell.

Table 1.2-5 Kalana MRE, June 2020 By Weathering

Rock Type	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Gold (koz)	Tonnes (kt)	Grade (Au g/t)	Gold (koz)
Regolith/Laterite	665	0.91	20	1	0.62	0
Saprolite	7,720	1.67	415	12	0.96	0
Transition	3,212	1.62	168	8	1.38	0
Fresh	31,931	1.50	1,540	4,188	1.64	221
Total	**43,527**	**1.54**	**2,158**	**4,210**	**1.64**	**222**

Kalanako has been classified using drillhole spacing and mineralisation lode size (i.e. overall geological confidence) as the primary criteria (Table 1.2-6). As with Kalana, no Measured category material was assigned, largely because of the relatively poor grade continuity definition provided by the drilling data and its relatively wide spacing. A RPEEE limit was developed using pit optimisation methods to define a pit shell.

Table 1.2-6 Kalanako MRE, June 2020 By Weathering

Rock Type	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Gold (koz)	Tonnes (kt)	Grade (Au g/t)	Gold (koz)
Regolith/Mottled Zone/Laterite	16	1.80	1	13	1.75	1
Saprolite	1,219	2.23	87	314	1.61	16
Transition	318	1.85	19	20	8.54	5
Fresh	61	2.18	4	4	0.83	0
Total	**1,614**	**2.15**	**112**	**351**	**1.99**	**23**

1.2.7 Statement

The Kalana Project comprises two primary deposits, Kalana and Kalanako, and two TSFs. The Kalana Deposit has an Indicated Mineral Resource of 43.5 Mt at a grade of 1.54 Au g/t containing 2.16 Moz gold and an Inferred Mineral Resource of 4.2 Mt at a grade of 1.64 Au g/t containing 222 koz gold.

Kalanako has an Indicated Mineral Resource of 1.6 Mt at a grade of 2.15 Au g/t containing 112 koz gold and an Inferred Mineral Resource of 351 kt at a grade of 1.99 Au g/t containing 23 koz gold.

The two TSFs have a combined Indicated Mineral Resource of 845 kt at a grade of 1.8 Au g/t containing 48 koz gold. No cut-off grade has been applied and the entire Resource would be mined.

Table 1.2-7 Kalana Project Mineral Resource Estimate, as of June 30, 2020

Deposit	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Gold (koz)	Tonnes (kt)	Grade (Au g/t)	Gold (koz)
Kalana	43,527	1.54	2,158	4,210	1.64	222
Kalanako	1,614	2.15	112	351	1.99	23
TSFs	845	1.8	48	-	-	-
Total	**45,986**	**1.57**	**2,318**	**4,561**	**1.67**	**245**

Notes:

Mineral Resource estimates follow the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition standards for Mineral Resources and Reserves and have been completed in accordance with the Standards of Disclosure for Mineral Projects as defined by National Instrument 43-101.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Kalana and Kalanako Mineral Resources are constrained by MII $1,500/oz Pit Shell and based on a cut-off of 0.5 Au g/t.

The TSF Mineral Resource does not have a cut-off grade applied.

Reported tonnage and grade figures have been rounded from raw estimates to reflect the relative accuracy of the estimate. Minor variations may occur during the addition of rounded numbers.

The Qualified Person for the exploration data and geological interpretation is Helen Oliver MSc FGS CGeol, Endeavour Mining Group Resource Geologist. The Qualified Person for the Tailings Mineral Resource Estimate is Ms. Oliver.

The Qualified Person for the Kalana and Kalanako Mineral Resource Estimate is Paul Blackney BSc Hons MAusIMM MAIG, Optiro Pty. Ltd.

The cut-off date for the Kalana drill hole database is 1 July 2018 and 7 February 2018 for the Kalanako drill hole database. The effective date of the MRE is 30 June 2020.

1.3 Mining

1.3.1 Pit Optimisation

Inputs and Parameters

Block models were developed by Optiro for both pits: Kalana used 10 mE x 10 mN x 5 mRL parent blocks using top cut ordinary kriging with a secondary step using local uniform conditioning into 5 mE x 5 mN x 1.25 mRL block size (or subcells), while for Kalanako 2.5 mE x 20 mN x 10 mRL parent blocks were used with a secondary step using local uniform conditioning into 1.25 mE x 5 mN x 2.5 mRL block size. No additional dilution or ore loss factors were applied to either pit – these items are accounted for in the mining models.Pit wall slope angles used are summarised in Table 1.3-1, based on design parameters from Snowden; the steeper wall angles reflect a lower reduction for ramps.

Table 1.3-1 Pit Optimisation Slope Angles by Weathering

Weathering Type	Base		Steeper	
	Footwall (°)	Hangingwall (°)	Footwall (°)	Hangingwall (°)
Laterite	26.5	26.5	28.5	28.5
Saprolite	26.5	26.5	28.5	28.5
Saprock	42.0	45.0	42.0	45.0
Fresh	46.5	49.5	48.0	52.0

The processing rate used in the optimisation varied by weathering type, as shown in Table 1.3-2.

Table 1.3-2 Pit Optimisation Processing Rate by Weathering

Weathering Type	Processing Rate (Mdt/a)
Laterite	4.0
Saprolite	4.0
Saprock	3.6
Fresh	3.0

Mining costs used varied by pit, depth and weathering type using algorithms taking into account the components of the cost. A rehabilitation cost of $0.06/dmt was included for closure.

A royalty of 3% and stamp duty of 0.6% was applied to all sales based on the price. A transport and refining charge of $4/oz was also applied. Time costs were included as variable costs in the mining and ore costs.

A gold price of $1,500/oz was used.

A discount rate of 5% was applied with revenue and cost being discounted by depth using a vertical advance rate of 40 m per year. With the exception of mining mobile equipment, which was applied as part of the mining operating cost, no capital or taxation costs were included.

Using the parameters and modifying factors outlined above, Table 1.3-3 shows the marginal cut-off grade by weathering.

Table 1.3-3 Pit Optimisation Cut-offs

Weathering Type	Kalana (A g/t)	Kalanako (Au g/t)
Laterite	0.41	0.42
Saprolite	0.41	0.42
Saprock	0.52	0.53
Fresh	0.60	0.61

1.3.2 Pit Design

The weathered horizon (base of saprolite) is typically 70 m below the surface at the pit perimeter. The Kalana design is based on Pit Shell 28 from the kal16 optimisation. The minimum width for all stages was 40 m with a minimum mining width of 20 m.

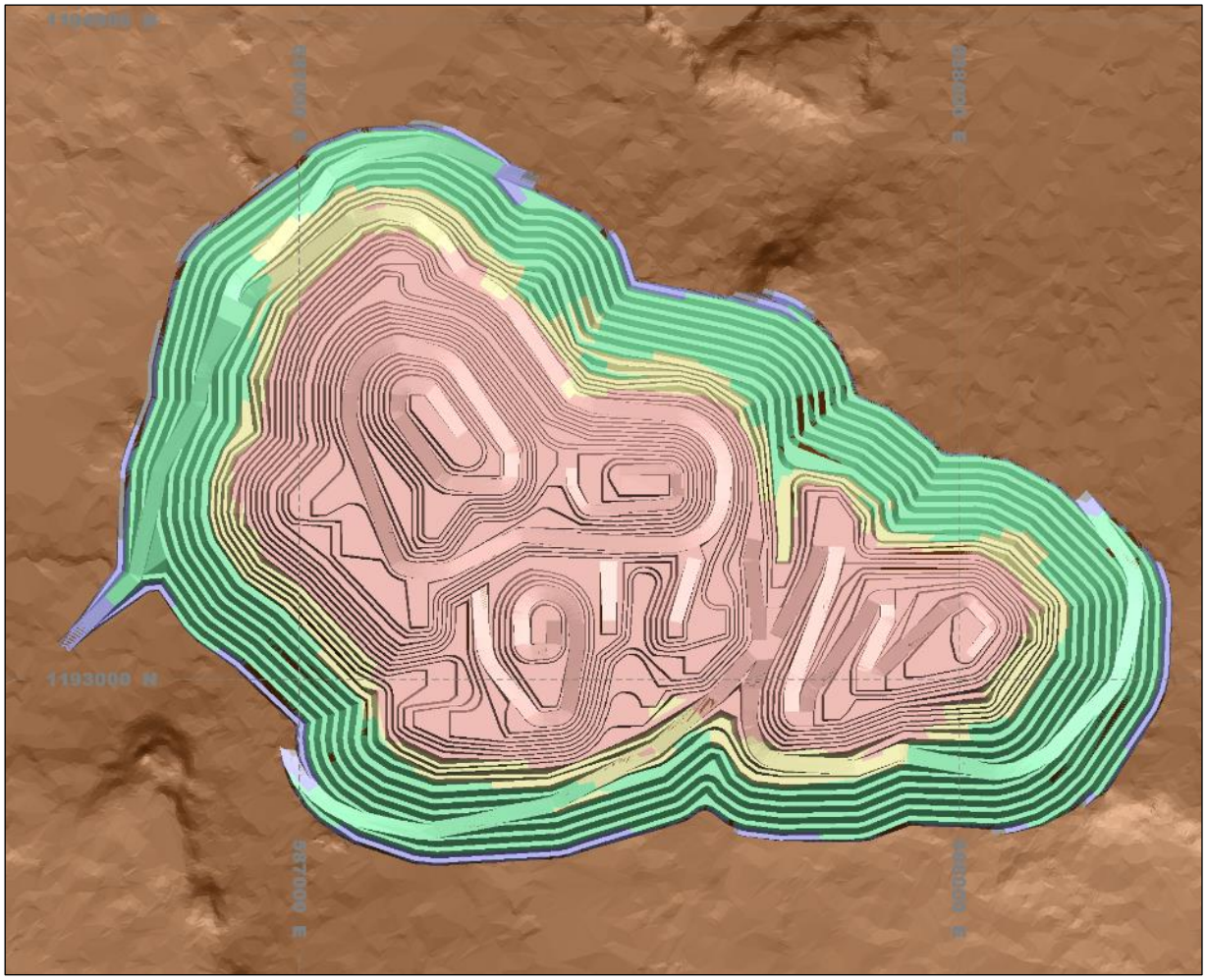
Table 1.3-4 Pit Design Parameters by Weathering

Parameter	Laterite / Saprolite		Saprock		Fresh	
	Footwall	Hangingwall	Footwall	Hangingwall	Footwall	Hangingwall
Batter angle (°)	45	45	65	75	70	80
Batter height (vertical m)	10	10	10	10	10	10
Batter berm interval (vertical m)	10	10	10	10	10	10
Berm width (m)	7	7	5	5	5	5
Inter-ramp slope (toe to toe, no ramp) (°)	30.5	30.5	46.0	52.5	49.2	55.9
Geotechnical berm interval (vertical m)	Base of saprolite	Base of saprolite	100	100	100	100
Geotechnical berm width (m)	10	10	10	10	10	10
Overall slope (toe to crest, ramps) (°)	26.7	26.6	42.0	46.6	45.0	49.5

The orebody was previously mined underground and most of the workings are inside the pit shell and may intersect the final wall in the southwest. Void management will be required over much of the mine life.

The ultimate pit design for Kalana is shown in Figure 1.3.1 with the weathering types exposed on the walls (laterite, saprolite, saprock, fresh). The pit will be developed in six stages, which will progressively impact on Kalana township from Stage 1, as well as other existing infrastructure.

Figure 1.3.1 Ultimate Kalana Pit Design (blue: laterite, green: saprolite, yellow: transition, red: fresh rock)



The ultimate Kalanako pit design is shown in Figure 1.3.2. It is based on Pit Shell 28 from the kko10 optimisation and forms two separate pits. These pits are almost entirely within the saprolite weathering horizon. Stage 1 reaches a maximum depth of 80 m at the 300 level, while Stage 2 reaches a maximum depth of 100 m at the 290 level.

Figure 1.3.2 Ultimate Kalanako Pit Design (plan view - blue: laterite, green: saprolite, yellow: transition)

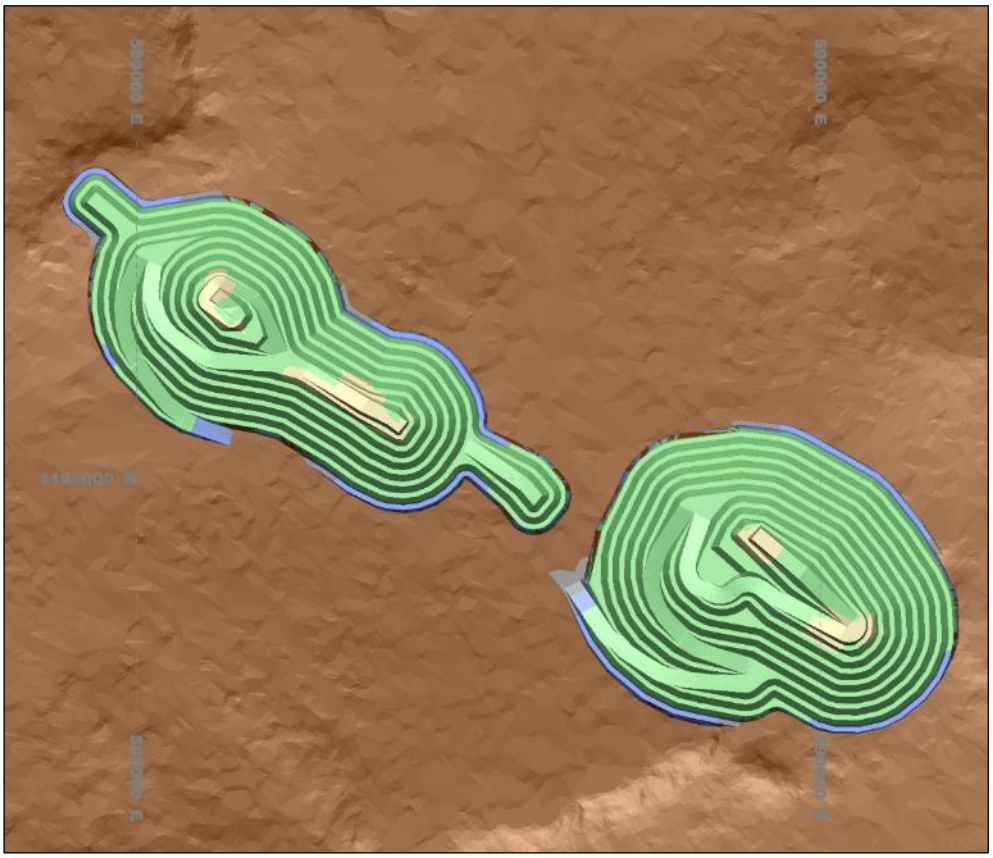


Table 1.3-5 summarises the pit design by stage. Nearly 97% of the project ore tonnes are from Kalana, the remainder is from Kalanako which is higher grade at a higher strip ratio. Underground voids are predominantly in the later Kalana stages with Stage 4 having the largest void volume.

Table 1.3-5 Pit Design Summary

Deposit	Stage	Volume (Mbcm)	Ore (Mdmt)	Gold (g/t)	Waste (Mdmt)	Total (Mdmt)	Strip ratio (waste:ore)	Pit bottom (mRL)	Maximum depth (m)	Underground void volume (kbcm)
Kalana	1	5.1	1.5	2.27	7.3	8.8	5.0	290	90	-
	2	8.5	3.9	2.03	11.6	15.5	3.0	270	110	13
	3	14.5	2.7	1.63	23.1	25.8	8.6	260	120	-
	4	17.9	7.9	1.41	29.3	37.2	3.7	180	200	91
	5	31.4	7.0	1.72	63.1	70.1	9.0	180	200	68
	6	41.3	10.7	1.33	83.6	94.3	7.8	80	300	65
	Subtotal	**118.7**	**33.6**	**1.58**	**218.1**	**251.7**	**6.5**	**80**	**300**	**237**
Kalanako	1	5.5	0.5	2.32	9.2	9.7	20.3	300	80	-
	2	7.9	0.7	2.13	13.1	13.9	18.7	290	100	-
	Subtotal	**13.4**	**1.2**	**2.21**	**22.4**	**23.5**	**19.3**	**290**	**100**	**-**
Total		**132.1**	**34.8**	**1.60**	**240.5**	**275.2**	**6.9**	**80**	**300**	**237**

1.3.3 Mineral Reserve

The Kalana and Kalanako open pits contains 34.8 Mt at 1.60 Au g/t, inclusive of dilution and mining loss, derived from an Indicated Mineral Resource which is classified as a Probable Mineral Reserve as summarised in Table 1.3-6.

Table 1.3-6 Kalana and Kalanako Mineral Reserve, as of 31 December 2020

Deposit	Item	Probable	Total Mineral Reserve
Kalana	Ore (Mt)	33.6	33.6
	Gold (g/t)	1.58	1.58
	Gold (Moz)	1.71	1.71
Kalanako	Ore (Mt)	1.2	1.2
	Gold (g/t)	2.21	2.21
	Gold (Moz)	0.09	0.09
Old Tailings	Ore (Mt)	0.82	0.82
	Gold (g/t)	1.67	1.67
	Gold (Moz)	0.04	0.04
Total	**Ore (Mt)**	**35.6**	**35.6**
	Gold (g/t)	**1.60**	**1.60**
	Gold (Moz)	**1.83**	**1.83**

The Kalana and Kalanako open pits also contain 0.1 Mt and 0.2 Mt, respectively, of Inferred mineralisation which has been scheduled as waste.

1.3.4 Overall Site Layout

The overall site layout with the Kalana township to the east and south of the Kalana deposit is shown in Figure 1.3.3.

Figure 1.3.3 Site Layout

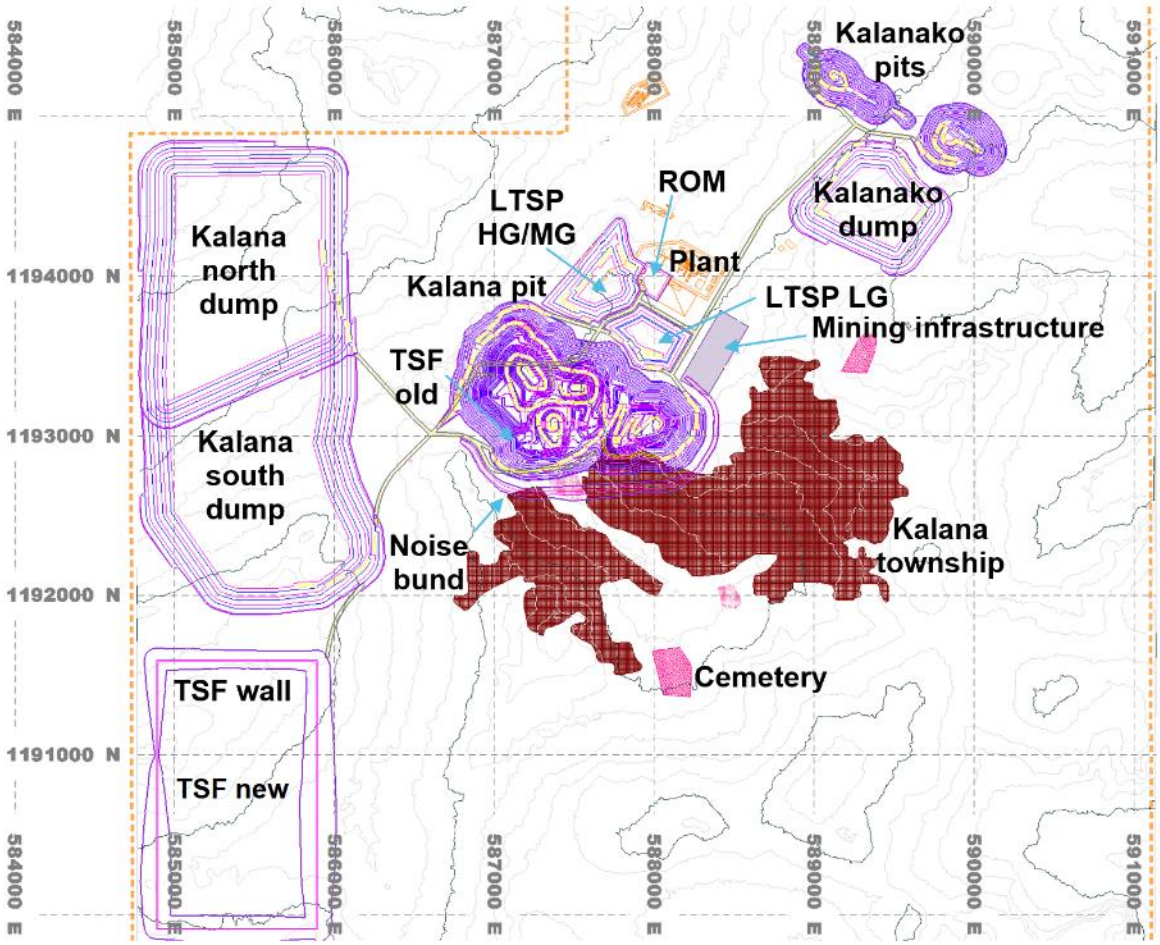


Table 1.3-7 summarises the waste dump capacities. This capacity is higher than the expected requirement of 152 Mlcm and allows for swell variations, variable dump top level (allowing optimised hauls and improved rehabilitation profile) or footprint changes due to exploration potential.

Table 1.3-7 Waste Dump Capacities

Waste Dump	Volume (Mlcm)
Kalana north	91.2
Kalana south	113.6
Kalanako	14.7
ROM base	0.2
Noise bund	1.1
TSF wall	10.1
Total	**230.9**

1.3.5 Schedule

Mine scheduling divided the material into four categories:

- Waste
- Low Grade: ≥Cut-off and <0.9 Au g/t
- Medium Grade: ≥0.9 and <1.3 Au g/t and
- High Grade: ≥1.3 Au g/t

Figure 1.3.4 shows the ex-pit movement by stage.

Figure 1.3.4 Total Ex-Pit Movement by Stage

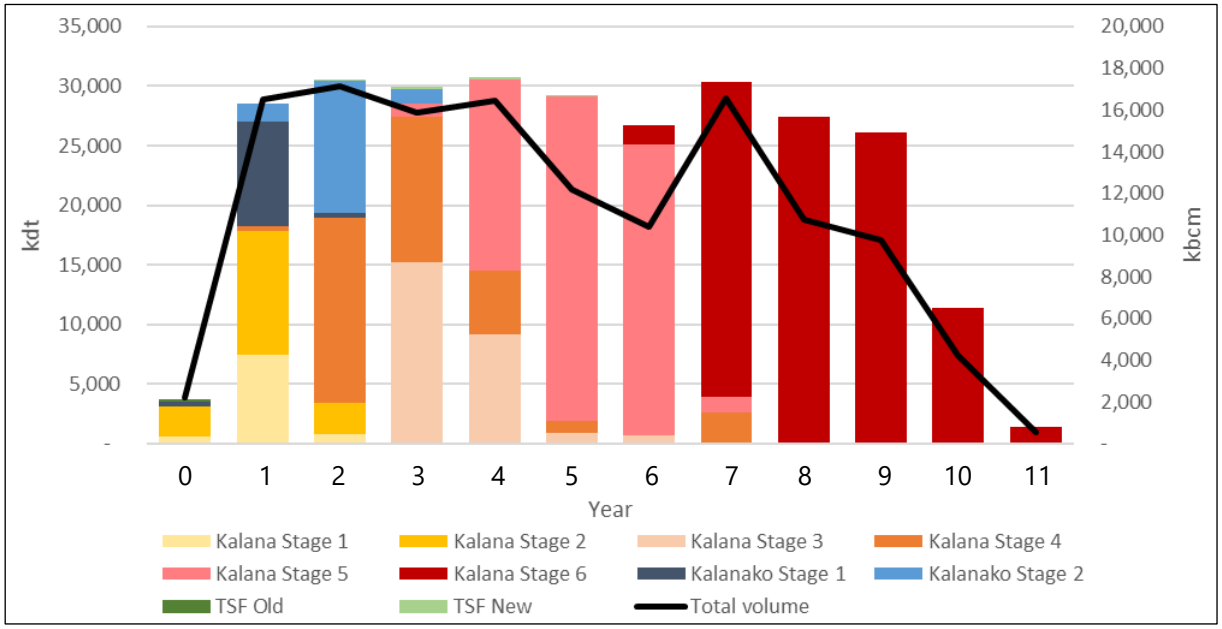
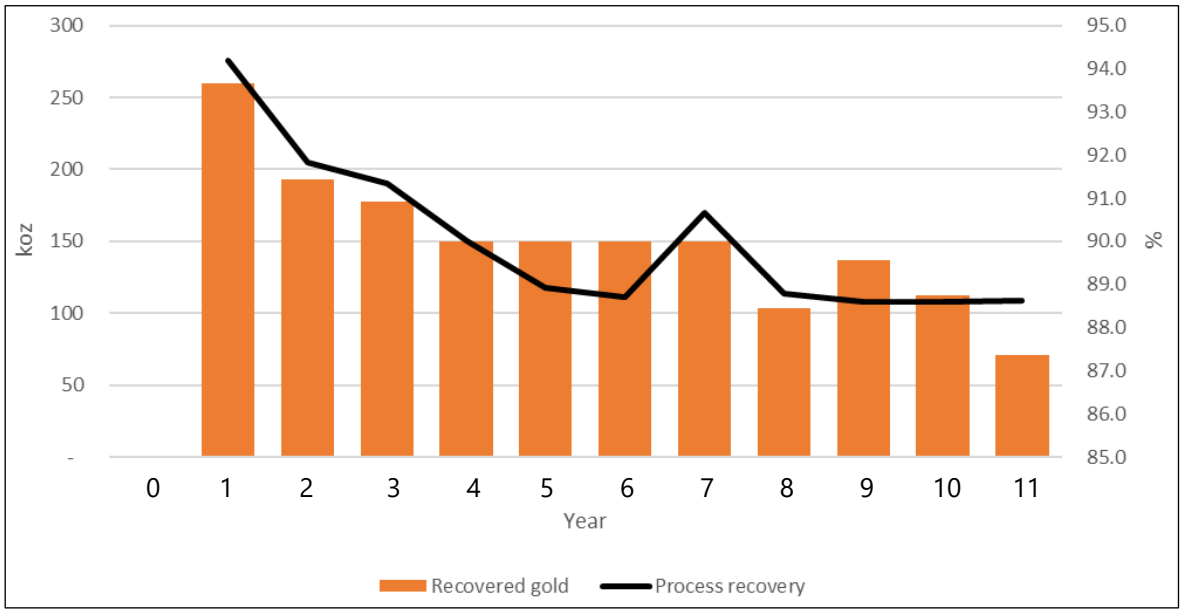


Figure 1.3.5 shows the metallurgical recovery and gold produced. Due to higher throughputs, above average grades and high metallurgical recoveries, the first three years of production are above 150 koz/a. This is followed by four years at 150 koz/a, with the remaining years below this level.

Figure 1.3.5 Gold Production and Metallurgical Recovery



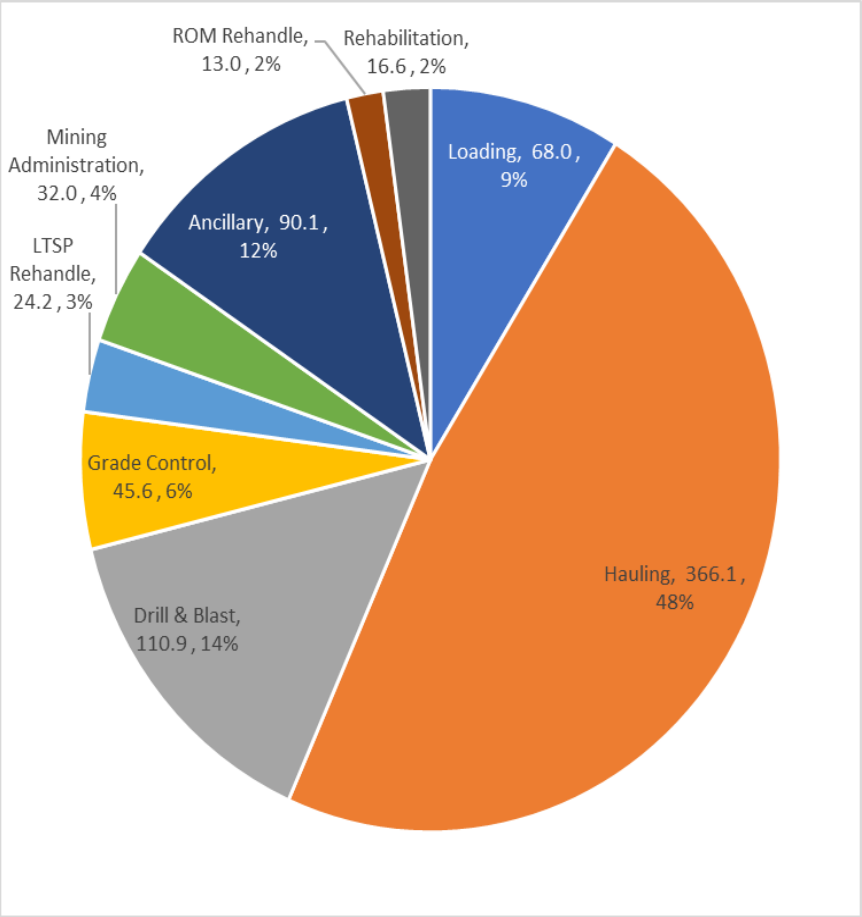
1.3.6 Mining Costs

Endeavour calculated mining costs based on haulage paths provided by Snowden. Figure 1.3.6 shows the life of mine operating costs by area, while Table 1.3-8 shows the LOM mining costs by activity.

Table 1.3-8 Total Mining Cost ($M)

Activity	Mining Cost ($M)
Loading	68.0
Hauling	366.1
Drill and blast	110.9
Grade control	45.6
Stockpile	24.2
Admin	32.0
Ancillary	90.1
ROM	13.0
Rehabilitation	16.6
Total Mining	**749.7**

Figure 1.3.6 Mining Operating Cost ($M) – LOM



1.4 Metallurgy and Flowsheet Development

1.4.1 Introduction

Lycopodium developed a comprehensive metallurgical testwork programme in 2018 and selected appropriate core samples from the ore body that will be treated through the processing facility. The majority of the testwork was carried out between January and July 2018, with some additional tests performed between November 2018 and January 2019. No additional testwork has been carried out since that time.

The following provides a summary of the testwork undertaken:

- Comminution tests performed on 14 samples representing a range of lithologies across the three main weathered rock types (saprolite, saprock, and primary) intended to produce a range of comminution data (SMC values, Bond Ball Mill Work indices, and Abrasion indices) for input into equipment sizing.

- Initial variability tests performed on 32 samples representing a range of lithologies across the three main weathered rock types, under standard cyanide leaching conditions and at a fixed particle size. The intention of these tests was to determine the relative gold recovery from the various lithologies and identify any anomalies prior to preparation of the Master Composites.

- LeachWELL tests performed on 56 primary ore samples to establish overall gold recoveries across the resource. This was necessary as the previous initial variability tests demonstrated no discernible correlation between gold recovery and lithology and thus could not facilitate the selection of appropriate samples to generate the Master Primary Ore Composites. The LeachWELL results were used to select appropriate samples, based on overall gold recovery, to establish two Primary Ore Master Composites based on 'high' and 'low' gold extraction.

- Optimisation tests performed on five Master Composites, to establish optimum leaching conditions (grind size, pulp density, and cyanide concentration) and to establish CIL, cyanide detoxification, and arsenic precipitation parameters.

- Oxygen uptake rate tests, and rheology tests were undertaken for each of the five Master Composites.

- Final Variability tests performed on 20 samples representing a range of lithologies across the three main weathered rock types, under optimised grind, pulp density, and cyanide leaching conditions.

Comminution Testwork – Results Interpretation

The results of the comminution testwork programme conducted on 14 ore samples across all predominant lithologies and weathered rock types were forwarded to Orway Mineral Consultants (WA) Pty Ltd (OMC) for interpretation. Based on this information and Endeavour's request to process 3 Mtpa of fresh ore to a target P_{80} of 90 µm, OMC has proposed a primary crush followed by SAG and Ball mill circuit (1C SABC) design to provide sufficient flexibility to accommodate all ore types likely to be treated at Kalana. Modelling of the oxide ore, however, suggests that the selected mills are likely to overgrind the oxide ore to some extent, even at maximum turndown. While this will likely be beneficial for CIL leaching, there may be some viscosity impact on thickening and dewatering processes.

None of the material tested in the comminution testwork programme was classified as 'Abrasive', 'Highly Abrasive', or 'Extremely Abrasive'.

Initial Variability Testwork

The objective of the initial metallurgical variability tests was to compare the gold extraction for each sample under the same set of standard cyanidation leach conditions for each of the various lithologies and weathered rock types across the Kalana deposit. The results of this initial variability programme would then be used as a basis upon which to establish the Master Composites for the subsequent optimisation testwork programmes.

The following provides a summary of the results:

- Gold extraction varied significantly across the samples, with 10 of the 32 initial variability tests returning gold extractions >95%, while six of the tests yielded gold extractions <80%.
- The highest tails gold assays corresponded with the highest head assays.
- Gravity recoverable gold (GRG) from the 32 samples varied significantly, returning maximum and minimum values of 84.6% and 8.4%, respectively, with an average GRG of ~49%.
- Overall gold recovery from the eight Oxide ore samples tested was very high (93.4% minimum, 98.2% maximum, 95.7% average).
- Overall gold recovery from the three Transition ore samples was also high (82.7% minimum, 95.0% maximum, 88.5% average).
- Overall gold recovery from the 21 Primary ore samples was highly variable (19.0% minimum, 98.7% maximum, 78.2% average).
- Comparison of the gold recoveries based on primary ore lithology showed no discernible correlation. Consequently, compilation of a Primary ore Master Composite, based on primary ore lithology, for the subsequent testwork phase was not possible based on these results alone.

Subsequently, a series of high intensity cyanide leach (LeachWELL) tests was performed on 56 primary ore samples across 21 diamond drill holes. Based on the Au recovery results, the samples have been categorised into high (>90%) and low (<90%) Au recovery and used to establish the two Primary ore Master Composites.

Optimisation Testwork Programme

Following completion of the comminution and initial variability testwork programmes, a separate optimisation testwork programme was implemented to optimise the grind size and subsequent cyanide leach conditions for the proposed CIL circuit. This optimisation programme also included gravity recovery, oxygen uptake rate, preg-robbing index, standard carbon loading (kinetic), cyanide detoxification, arsenic removal, and slurry rheology tests.

The results of the tests performed on each of the Master Composites indicate a significant proportion of gravity recoverable gold for both the Kalana oxide (Composite #1) and Kalanako oxide (Composite #3) ores, with a lower percentage of gravity recoverable gold from the Kalana transitional (Composite #2). Preg-robbing is unlikely to occur but, if required, the plant design can incorporate a pre-leach stage.

The results of the grind optimisation study indicate:

- Maximum revenue is realised for the high recovery primary ore at a grind size P_{80} of 90 µm, while maximum revenue is generated for the low recovery primary ore at a grind size P_{80} of 125 µm.
- The increase in gold revenue (recovery) with fineness of grind is offset by the increase in operating costs to achieve the finer grind sizes.
- Given the significantly greater net revenue delta for the high recovery primary ore when comparing the P_{80} grind sizes of 90 µm and 125 µm (~$15.8M/year) versus a similar comparison for the low recovery primary ore (~$9.6M/year), the optimum grind size P_{80} for the Kalana primary ore is 90 µm.

This grind size was selected for the remainder of the testwork.

The results of the pulp density testwork indicated no distinct benefit with respect to gold recovery, cyanide consumption, and lime consumption between pulp densities of 40% w/w and 50% w/w solids. Consequently, the midpoint pulp density of 45% w/w solids was adopted as the optimum condition for all subsequent testwork.

The results indicate:

- The initial leaching rates of the samples increase with increasing initial cyanide dosage at the range of concentrations tested.
- There is a trend of decreasing tails gold grade with increased cyanide dosage rate.

- Overall cyanide consumed by the samples at different initial cyanide dosage rates increases with increased starting cyanide concentration.

- The lime (60% available CaO) requirements by the Perth tap water ranged from 1.6 to 2.0 kg/t for Kalana Oxide and Transition ores, decreasing to ~0.7 kg/t for Kalanako Oxide, and ~0.4 kg/t for Kalana Primary ore.

The optimised leach conditions were selected as: Grind size of 90 µm, pulp density of 45% and initial cyanide concentration of 350 ppm.

Diagnostic leach tests conducted on Master Composite #5 (Kalana Primary, Low recovery) indicate that approximately two thirds of the unleached gold is locked within sulphide minerals, confirming the refractory nature of this primary ore sample and the need to adopt a different processing route for this particular ore type if gold recoveries are to be improved. The remaining one third of the unleached gold is locked in silicates.

The oxygen uptake tests demonstrated that the oxygen uptake demand is low (i.e. significantly less than 0.15 mg/L.min for all values recorded), the addition of oxygen instead of air to the CIL tanks is proposed as it offers a number of other significant benefits, such as reduced CN consumption, improved leach kinetics, oxidation of reactive sulphides and reduced potential for HCN generation and CN stripping.

Rheology Tests

The results indicate:

- Master Composite # 1 (Kalana Oxide) exhibited the highest viscosity, closely followed by Master Composite # 3 (Kalanako Oxide). Both indicate that operation of a CIL circuit at a pulp density greater than 50% w/w solids would not be recommended.

- The Kalana Transition ore (Master Composite # 2) displayed slightly lower pulp viscosity characteristics than the oxide material, indicating that operation of a CIL circuit at a 50%w/w solids pulp density should not be problematical.

- Both the Kalana Primary ore composites (Master Composite # 4 and Master Composite # 5) exhibited low apparent viscosity at low pulp densities, indicating that 'sanding' of the CIL tanks may pose a problem if the circuit is operated at low pulp densities. The inclusion of a pre-leach thickener ahead of the CIL circuit will allow the CIL to operate at densities of 50%w/w solids or greater which will improve slurry mixing characteristics.

- Pumping of solids at densities up to ~60% w/w solids using centrifugal pumps should not be problematical for all of the composites tested.

Bulk Cyanidation and Carbon Adsorption

Bulk gravity separation and cyanidation testwork was conducted on each of the five Master Composites under the optimised conditions. A summary of the bulk cyanidation testwork results is provided in Table 1.4-1.

Table 1.4-1 Bulk Cyanidation Leach Results at Optimised Conditions

Master Composite	Grind Size P_{80} (µm)	% Solids (w/w)	Initial NaCN (ppm)	Calc. Head (g/t)	GRG (%)	Overall Recovery (%)		Reagent Consumption	
								NaCN (kg/t)	Lime (kg/t)
				Au	Au	Au	Ag		
Composite #1 [Kalana Oxide]	90	45	350	3.89	72.8	98.1	71.0	0.15	1.87
Composite #2 [Kalana Trans]	90	45	350	2.54	35.7	95.5	78.5	0.23	1.90
Composite #3 [Kalanako Oxide]	90	45	350	0.69	50.3	93.7	66.1	0.21	0.53
Composite #4 [Kalana Primary]	90	45	350	1.38	9.35	92.0	72.9	0.23	0.50
Composite #5 [Kalana Primary]	90	45	350	1.36	10.2	76.2	68.7	0.21	0.37

Carbon adsorption testwork results indicate good gold adsorption characteristics for most composites except the Kalanako Oxide composite which had a relatively low equilibrium carbon loading.

Cyanide Detoxification and Arsenic Precipitation

The cyanide detoxification testwork indicates the following:

- CN_{WAD} concentrations in the discharge liquor following cyanide detoxification for each of the Master Composites may be reduced to <3 ppm CN_{WAD} using either milk of lime or sodium hydroxide solution for pH control.
- The target CN_{WAD} value of <5 mg/L in the cyanide detoxification discharge liquor is able to be achieved with the addition of ~200% stoichiometric addition of SMBS for each of the Master Composites.
- It is required to supplement the cyanide detoxification reactions with copper sulphate solution for each of the Master Composites as there is insufficient soluble copper to catalyse the reaction following cyanide leaching.

The results of the arsenic precipitation tests demonstrated the ability to achieve low arsenic concentration (≤1 mg/L) in the final effluent using the Scorodite process. Although these final arsenic concentrations still exceed the target value of <0.5 mg/L for the individual composites, it is likely that the commercial facility will be exposed to a blend of composites which, it is expected, will reduce the final solution arsenic concentration to <0.5 mg/L. Average sulphuric acid consumption for the four composites tested at pH 6 is 3.3 kg/t.

Variability Testwork

Twenty individual samples from across the orebody were tested through the optimised process, with the results comparing favourably with those from the Master Composite testwork, indicating relatively consistent metallurgical response throughout the orebody.

1.4.2 Process Design Summary

The metallurgical treatment route selected has been based on the results of the metallurgical testwork programmes and may be summarised as follows:

- Single stage primary crushing.
- SABC (SAG mill, Ball mill, recycle pebble crushing) comminution circuit.
- Gravity concentration.
- CIL feed thickening.
- Carbon-in-leach (CIL).
- Split AARL stripping circuit.
- Air / SO_2 cyanide detoxification circuit.
- Arsenic precipitation circuit.

The key process design parameters derived from the comminution and metallurgical testwork are summarised in Table 1.4-2.

Table 1.4-2 Key Process Design Parameters

Milling Circuit Design	**Parameter**
SMC A x b	26.7 (85th percentile)
Bond Ball Mill Work Index (BBWi)	18.7 kWh/t (85th percentile)
Crushing Work Index (CWi)	27.7 kWh/t (85th percentile)
Abrasion Index (Ai)	0.151 (85th percentile)
Primary Crushing Circuit Product Size, P_{80}	130 mm
Milling Circuit Product Size, P_{80}	90 µm
Gravity Gold Recovery	65%
Gravity Silver Recovery	15%
CIL Design	
Pre-leach Thickener Flux	1.0 t/m^2.h
Flocculant Addition	30 g/t
CIL Residence Time	36 hours (primary ore)
CIL Pulp Density	50% w/w solids (primary ore)
Cyanide Addition (nominal)	0.13 kg/t (oxide ore), 0.15 kg/t (primary ore)
Lime Addition (nominal)	1.27 kg/t (oxide ore), 0.67 kg/t (primary ore)
Aeration	Oxygen sparging
Operating pH	10.5
Carbon loading kinetic parameters, k and n values	182, 0.75 (gold); 108, 0.68 (silver)
Cyanide Destruction Design	
Cyanide Destruction Process	Air/SO_2
Residence Time	2 hours
SMBS Stoichiometric Requirement	200%
Operating pH	9.0
pH Control Reagent	Sodium Hydroxide Solution
Aeration	Oxygen sparging
Arsenic Precipitation Design	
Residence Time	1 hour
Iron Sulphate Addition, Fe:As	15:1
Operating pH	Natural (as received)
Aeration	Oxygen sparging

Gold and Silver Recovery

Analysis of the variability leach test results indicated that a linear regression relationship to predict gold and silver recoveries from head grades was not favoured. An alternative approach took the average extractions obtained from the leach variability tests and applied these to the mine schedule for the appropriate ore type to yield a weighted recovery value for each year. Using this information, and assuming soluble gold and silver losses of 0.01 mg/L and 0.03 mg/L, respectively and a CIL pulp density of 50%w/w solids, it is possible to predict the gold and silver recoveries for each year of the life-of-mine. These recoveries are summarised in Table 1.4-3.

Table 1.4-3 Predicted Life of Mine Metallurgical Recoveries

Mine Year	1	2	3	4	5	6	7	8	9	10	11
Au Extraction, %	94.1	92.8	91.7	90.4	89.0	88.9	92.1	88.8	88.6	88.6	88.6
Ag Extraction, %	46.4	46.2	46.0	45.9	45.6	45.5	46.0	45.5	45.5	45.5	45.5

Reagent Consumption

A summary of the life-of-mine sodium cyanide and lime consumptions is provided in Table 1.4-4.

Table 1.4-4 CIL Lime and Cyanide Consumption

Composite	NaCN Consumption kg/t	Lime[1] Consumption kg/t
Composite # 1 (Kalana Oxide)	0.14	1.80
Composite # 2 (Kalana Trans)	0.22	1.85
Composite # 3 (Kalanako Oxide)	0.10	0.70
Composite # 4 (Kalana Primary)	0.17	0.40
Composite # 5 (Kalana Primary)	0.14	0.30

Note: 1. Lime consumption based on 90% available CaO.

Refractory Composite

Grind sensitivity leach tests on Master Composite # 5 (Kalana Primary Ore – Low Gold Recovery) indicated that there is little improvement in gold extraction unless much fine grinding to a P_{80} of 15 µm is performed, which is unlikely to be economic.

Thickening Testwork

The results of the thickening testwork performed on the five Master Composite samples concluded that each of the samples can be successfully thickened to high densities over a range of flux rates. The results have enabled a series of preliminary thickener sizing criteria to be developed for each Master Composite.

1.5 Process Plant

The key project and ore specific design criteria that the plant design must meet are as follows:

- 3,000,000 t/y of primary ore.
- Crushing plant mechanical availability of 80% (7,008 h/y).
- Mechanical availability for the remainder of the plant of 91.3% (8,000 h/y) supported by crushed ore storage and standby equipment in critical areas.
- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

The selected milling circuit has been sized to accommodate the 85th percentile in terms of competence of the ores to be treated.

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet as shown in Figure 1.5.1.

1.5.1 Water Supply

The raw water supply to the processing plant will be sourced from the Kalana pit. The quality and treatment requirements for this water has not yet been clearly defined and a dedicated raw water treatment facility is currently excluded from the project scope.

1.5.2 Electrical Design

The Project, including all mining and processing facilities, accommodation and infrastructure is estimated to have a maximum demand of 17.5 MW, and an average load of 16.4 MW. The largest single loads will be the SAG mill and ball mill, each with an installed motor power of 6.0 MW. The SAG mill will be provided with a VVVF drive which will minimise the impact on the power system during starting, while the ball mill will have a liquid resistance starter.

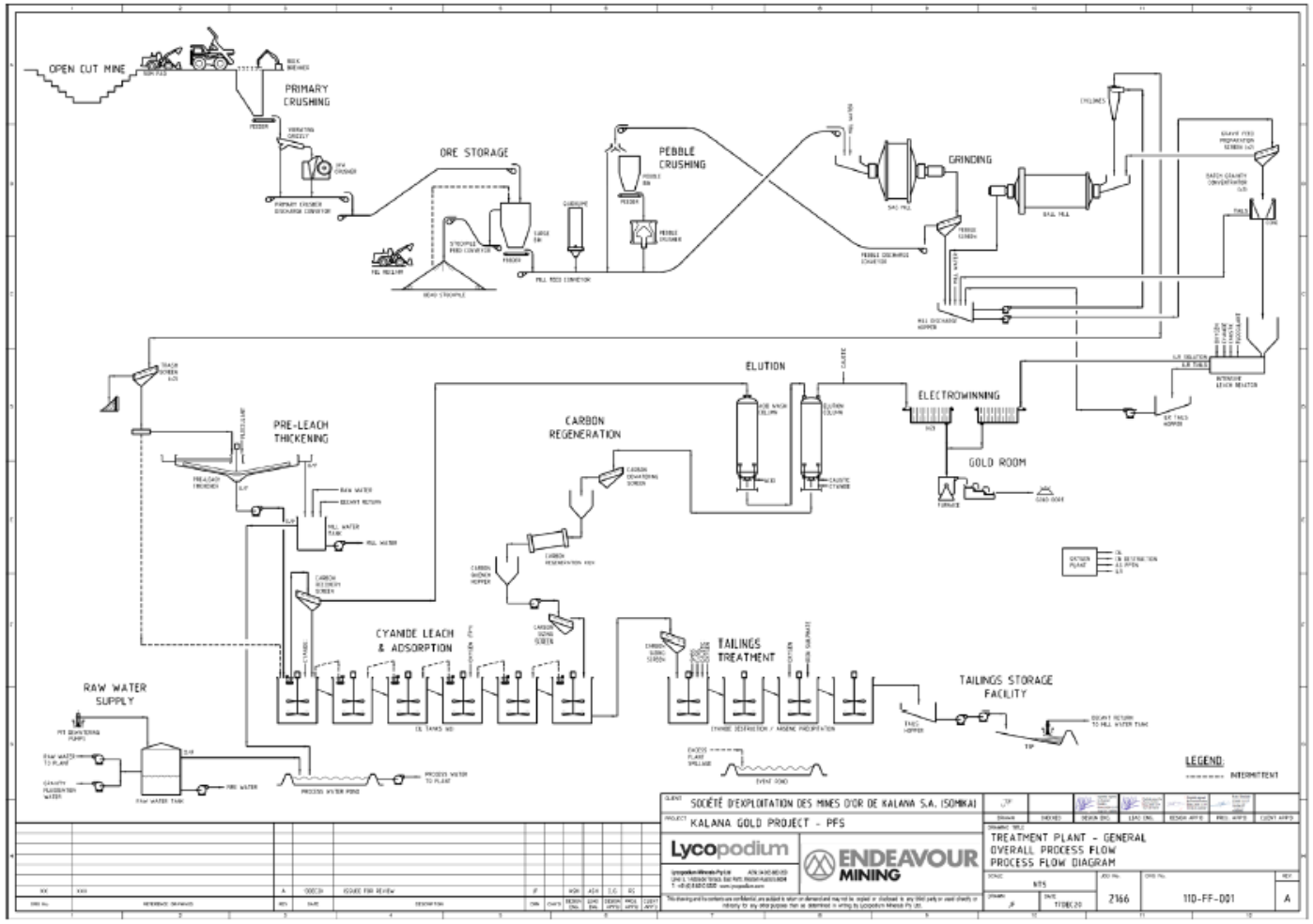
1.5.3 Communications

The onsite communications network is designed around a site-wide fibre optic backbone which will be shared by all services.

A microwave internet link will provide external network connectivity to the site. Camp entertainment services will be provided via satellite TV.

A site-wide private mobile radio system will be installed for voice communications between vehicles and plant operations.

Figure 1.5.1 Overall Process Flow Diagram



1.6 Tailings and Water Management

1.6.1 Tailings Design Summary

The TSF will comprise a two-cell paddock storage facility formed by multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 77 ha for the Stage 1 TSF, increasing to 239 ha for the final TSF. For Stage 1, only Cell 1 will be constructed. Cell 2 will be commissioned in Year 3 of operation.

The TSF is designed to accommodate a total of 35.6 Mt of tailings. The Stage 1 TSF is designed for 18 months storage capacity. Subsequently, the TSF will be constructed in annual raises to suit storage requirements; however, this may be adjusted to biennial raises to suit mine scheduling during the operation.

Downstream raise construction methods will be utilised for all TSF embankment raises of the perimeter embankments.

The perimeter embankment upstream face will be lined with textured HDPE geomembrane liner, to allow safe egress from the TSF. A downstream seepage collection system will be installed within and downstream of the TSF perimeter embankments, to capture and return seepage from the TSF.

The TSF basin area will have a 200 mm thick compacted soil liner with a 1.5 mm HDPE geomembrane liner overlying the compacted soil liner. An underdrainage system will be installed to reduce pressure head acting on the basin liner system, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. A leakage collection and recovery system (LCRS) will be installed beneath the basin composite liner, to reduce water pressure build-up on the HDPE liner.

Supernatant water will be removed from the TSF via submersible pumps located within a decant tower in each cell. The supernatant pond will be maintained in the centre of the TSF. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit. An operational emergency spillway will be available at all times for each cell during TSF operation.

A pipeline containment trench will be constructed during Stage 1 to contain both the tailings delivery pipeline and decant return pipeline between the TSF and Plant Site, to reduce environmental impact if the pipeline bursts.

1.6.2 Tailings Testing

The results of the geochemistry testing are summarised below:

- The test results indicate that the tailings solids are Non-Acid Forming (NAF).
- Only arsenic was found to be highly enriched across all samples. Antimony was found to be slightly to significantly enriched in all samples, with four samples slightly to significantly enriched in sulphur and bismuth.

- Comparison of the multi-element results to soil quality screening guidelines indicated that the tailings samples did not meet the guidelines for human health, ecology or site contamination due to elevated concentrations of metals and metalloids.

- The supernatant samples did not meet water quality standards for release of water from mining operations, with exceedances recorded in the pre-detox tailings sample for fluoride and sulphate.

- The four arsenic precipitate tailings samples recorded a greater number of exceedances, with total dissolved solids (TDS), arsenic, mercury and sulphate elevated in all samples. Total cyanide was found to elevated above the guidelines in two samples.

On the basis of the tailings solids multi-element results and the elevated arsenic and post detox cyanide concentrations, the TSF has been designed to prevent the loss of tailings solids.

1.6.3 Closure Summary

At the end of the TSF operation, the downstream faces of the embankments will have an overall slope profile of approximately 3.5H:1V, which will be inherently stable under both normal and seismic loading conditions.

Rehabilitation of the tailings surface will commence upon termination of deposition into the TSF. The final soil cover for the tailings surface subsequent to decommissioning will be confirmed during operation based on ongoing operational tailings geochemistry testing results. The following cover design for the tailings beach has been adopted at this stage but is subject to ongoing testing and review:

- Mine waste capillary break (500 mm).

- Low permeability fill layer (300 mm).

- Topsoil growth medium layer (200 mm).

- The finished surface will be shallow ripped and seeded with shrubs and grasses

1.6.4 Water Management

The water balance modelling included the TSF with a view to determining site water storage requirements. Design wet conditions were modelled to ensure that the TSF is designed with sufficient storage capacities to comply with design criteria. A constant pit dewatering rate of 605 m^3/h from the Kalana Pit as predicted by hydrogeological modelling, was used for the water balance modelling and will be available to provide make up and raw water to the process plant. It was assumed that any excess water generated from pit dewatering will be discharged directly off site.

The modelling predicts that the Kalana Pit dewatering supply (605 m^3/h) is sufficient to supply all raw and make-up water requirements for average and design dry climatic conditions, while the maximum annual discharge to the environment of excess water from the Kalana pit dewatering (after abstraction of raw water requirements) under design wet conditions is 3.6 Mm^3/year.

A preliminary groundwater desktop assessment was conducted for the Kalanako Pit. With limited hydrogeological data available, conclusions were based on data from the Kalana area assuming similar hydrogeological conditions. On this basis the preliminary, estimated total Kalanako inflow for both pits ranges between 3,110 m^3/day (129 m^3/h) and 4,147 m^3/day (173 m^3/h).

Surface water diversion structures around the TSF, waste dumps and Kalana pit have been designed on the basis of a 1:100 Annual Return Interval storm event.

1.7 Site Infrastructure

1.7.1 Site Development

The existing Kalana gold plant was designed and constructed by the Soviets for a throughput of 3,600 tpm (5 t/h). The plant will be demolished and the site made safe prior to starting construction for the new plant, which will therefore be a greenfield site.

The process plant and tailings storage facility will be located on the north eastern side of the open pit, outside of the 350 m blast zone. The ROM pad will be close to the pits as well as the mine services area (MSA). The accommodation camp will be located north of the process plant.

The 4.9 km main access road will approach the site from the south east and connect to RN8, part of the regional road network between Yanfolila and Bougouni.

1.7.2 Roads and Airstrip

Kalana Bypass Road

The existing Kalana village access road will be decommissioned and realigned to the northeast around the Kalana Pit. The Kalana Bypass Road (4.7 km long) will run from the Kalana village to the existing highway with two 3.5 m width running lanes and a one metre shoulder each side, for a total formation width of nine metres. The running surface will be sealed and have linemarking, kerbs and other appurtenances as required. The culvert crossings are designed to convey 100-year Average Recurrence Interval (ARI) storm events without overtopping the road.

Site Access and Plant Roads

The total length of the site access road is 13.8 km, with two 3.5 m width running lanes and a one metre shoulder each side, and a laterite running surface. Culvert crossings will convey all runoff resulting from a 20-year ARI storm event.

Minor tracks and roads will service the TSF, accommodation and security camps.

Mine haul roads will be constructed progressively as the mine develops, with two 12 m width running lanes and a 1.5 m high safety bund on each side of the road, for a total formation width of 30 m. Culvert crossings will accommodate all runoff from a 20-year ARI storm event.

Airstrip

The Project airstrip will be located four kilometres north of the process plant and will be an unsealed strip 800 m long and 18 m wide to accommodate a Pilatus PC-12 aircraft. A fully automated weather station will be included, together with refuelling facilities. No site investigation has been completed at this stage.

1.7.3 Electric Power

The installed load and maximum demand for the site is shown in Table 1.7-1.

Table 1.7-1 Plant Power Demand

Area	Plant Installed Load	Plant Maximum Demand	Plant Average Continuous Load
Process Plant	24,615 kW	16,400 kW	14,924 kW
Infrastructure	4,000 kW	3,200 kW	2,560 kW
Total	**28,615 kW**	**19,200 kW**	**17,484 kW**

The existing mine is supplied by a 66 kV overhead transmission line from Yanfolila, 51 km away. A new 66 kV bay and transformer will be installed there and a new 66 kV transmission line run to a new 66 / 11 kV substation to be installed adjacent to the process plant, where the voltage will be stepped down to 11 kV by a 25 / 33 MVA step down transformer.

Power will be distributed at 11 kV with 415 V working voltage. Emergency power generators will be supplied at the main plant (2,000 kVA).

1.7.4 Potable Water

The potable water demand has been estimated at 85 m^3/day and will be sourced from boreholes around the site and in a modular potable water treatment plant and stored in plant potable water tank. Potable water for the accommodation camp will be pumped from the water treatment plant at the plant site to a water storage tank at the camp.

1.7.5 Buildings

Layouts for site buildings were developed for scope definition purposes and include the mine services area, general administration facilities, and plant area buildings.

A 100-bed accommodation camp will be located approximately one kilometre north of the process plant and will provide accommodation for salaried and security staff not originating from the local area.

The existing operations camp supplemented with containerised accommodation units will be used for pioneer accommodation pending early completion of the new permanent camp. These containerised units will be moved to the permanent camp once is it operational. Construction contractors will provide their own accommodation.

1.8 Environment and Community

1.8.1 Permitting

The Project comprises an exploitation permit (Permis d'Exploitation) registered to SOMIKA, which was transferred from Avnel on 23 July 2003. The Permit is a unique 30-year exploitation permit that is derived from the legislation originally passed to enable SOGEMORK to develop the Kalana Mine.

A Final Environmental and Social Impact Assessment (ESIA) was submitted on 21 March 2016 for approval by the Minister of Environment. The ESIA for the Kalana Project was approved and the environmental permit issued on 28 April 2016 by the 'Ministere de l'Environnement, de l'Assainissement et du Developpement Durable'. The ESIA was accompanied by a Resettlement Action Plan (RAP) for the families to be resettled that are located within the 350 m buffer zone from the Kalana Open Pit.

According to the new design, including the new Kalanako pit, the exclusion zone will be extended. The extension of the pit size inside Kalana town will result in a change in the perimeter of the part of the village to be relocated and will increase the number of people that will be affected by the resettlement. There has also been construction on existing properties in the former impacted areas that have been approved by the Mediation Committee. In addition, the national road that was sealed by the Government in 2019 will require a sealed deviation. These changes and additions will affect the current ESIA and RAP, which will need to be addressed in amendments to be submitted to the authorities.

A permit is required for the discharge of water from the pit to ensure a safe working environment and to ensure compliance with the relevant standards. This is included in the ESIA report submitted and has been approved. Due to the changes to the Project since this approval was granted, this will also be captured in the amendment.

Over and above the local Malian laws and guidelines, additional best practice guidelines considered for the Project are described below:

- The Equator Principles (EPs) developed in conjunction with the International Finance Corporation (IFC).
- The IFC's sustainability framework, which includes the Policy and Performance Standards on Social and Environmental Sustainability.
- The IFC's Guidance Notes: Performance Standards on Environmental and Social Sustainability, which are technical reference documents with general and industry-specific examples of GIIP.
- The International Cyanide Management Code (ICMC).

1.8.2 Public Consultation

A series of public consultation meetings were held for the Project within Kalana town and the surrounding communities in accordance with decree N°2013/0256/MEA-MADAT-SG of 29 January 2013, which stipulates the methods for conducting public participation. An issues register was developed to record issues that were raised during the public meetings. These grievances were responded to and handed back to the Sub-prefect of Kalana, the Mayor of Kalana town and the village chiefs and their representatives for their approval and/or amendments.

As with permitting, a new public consultation will be conducted according to the new Decree and record all additional issues that will be raised during the Definitive Feasibility Study.

1.8.3 Resettlement

The Project requires the partial resettlement of Kalana town due to the placement of Project infrastructure and the need to ensure a safe buffer zone between the mining activities and the community. The development will also result in the economic displacement and compensation of various farmers in the area due to the establishment of mining infrastructure and associated activities on farms carrying subsistence and cash crops.

Resettlement and compensation will be undertaken in terms of the relevant Malian legislation.

1.8.4 Environmental and Social Management

The recommendations from the ESIA will be used to compile an Environmental and Social Management Plan (ESMP) to assist Endeavour in reducing potential impacts and risks and achieving its environmental objectives as well as fulfilling its commitment to the environment. The ESMP will be used to ensure compliance with environmental specifications, monitoring and management measures and will be implemented from site preparation through to decommissioning and closure. Monitoring plans will be based directly on the ESMPs.

1.8.5 Opportunities, Risks and Recommendations

These can be summarised as follows:

- The Project water balance was based on available hydrogeological and climate information. Actual flows will need to be confirmed and may require the establishment of additional boreholes and a review and update of the water balance once the testwork has been completed.
- The expected water quality and ultimate salt balance will need additional testwork for confirmation. A budget has been allocated for this work.
- A separate ESIA submission for the road realignment of the Yanfolila-Kalana road was requested by the DNACPN, this will be amended according to the new design.
- The Project's revised RAP and ESIA will be submitted to DNACPN and for approval by the authorities. The implementation of the RAP will be a complex process requiring negotiation with individuals and communities.
- The 2016 ESIA was completed based on an engineered US$1,400/oz Au pit design and associated infrastructure. A review of the impacts associated with the revised 2020 pit design found that the Project impact will be materially different due to an increase in the waste rock dump (WRD) footprint, additional cultivated fields affected by the mining infrastructure, as well as the need for the diversion of the Kalanako stream (and its associated impact on the aquatic environment). The ESIA will have to be amended together with the CAPEX and OPEX costs associated with the environmental and social mitigation measures required. The Definitive Feasibility Study budget has made provision for this additional capital expenditure. Although the 2016 ESIA for the Kalana Project was approved and the environmental permit issued on 28 April 2016, allowing Endeavour / SOMIKA to commence construction and mining, this permit is no longer valid. A new environmental permit will be required to commence construction and mining. The application for a new environmental permit will be initiated when the ESIA amendment in finalised.

1.8.6 Conclusion

Endeavour considers the Project is being undertaken with due consideration of biophysical, social and economic factors, as well as the relevant Mali legislative requirements, EPs and IFC Performance Standards. The economic benefits of such a development are numerous; however, as in any mining project of this nature, there are also negative impacts which will require planning, monitoring and mitigation during construction, operation and decommissioning phases. None of the impacts identified are considered as fatal flaws and as indicated, high significance impacts, after implemented mitigation measures, if implemented, will be of low to medium significance.

1.9 Permits, Royalties and Land Tenure

The Kalana Exploitation Permit (the Permit) is located in the Sikasso Region of southwestern Mali, and covers a surface area of 387.4 km^2, of which the Kalana deposit accounts for an area of approximately 2 km^2. The Kalana Deposit Mineral Resource is located near the centre of the northern part of the Permit, and is within one kilometre of Kalana town.

Kalana Holdings (formerly known as Avnel Gold Ltd.), a wholly-owned subsidiary of Endeavour Mining, has an 80% interest in SOMIKA, the holder of the Permit. The Government of Mali holds the remaining 20% beneficial interest in SOMIKA.

The Project comprises one Exploitation Permit (Permis d'Exploitation), registered to SOMIKA that grants the exclusive right of exploitation and exploration. The Permit confers the right to exploit and explore for gold and silver for a period of 30 years. According to Article 43 of the 1999 Mining Code, an exploitation permit is granted by decree for a 30-year period, renewable for a further 10-year period until depletion of the reserves contained within the boundaries of the permit.

During exploration, the construction of the proposed Kalana Mine and for three years following commercial production, SOMIKA will be exempt from VAT and certain custom duties related to the Project. This exemption will apply to any one or more further mines discovered and developed on the Permit.

Following the three-year period from the commencement of commercial production, an 18% value added tax (VAT), or 'TVA', will be applicable on goods and services, which is fully reimbursable, and customs duty will be applicable on imported goods.

As SOMIKA elected to stay under the old CIT regime in force before Loi de Finance 2012, the tax structure in Mali applicable to its operations consists of a general corporate income tax rate of 35% of profits after deductions for expenses and allowable depreciation. SOMIKA is also subject to a minimum tax rate of 1.0% of net revenue, even in years with a deficit.

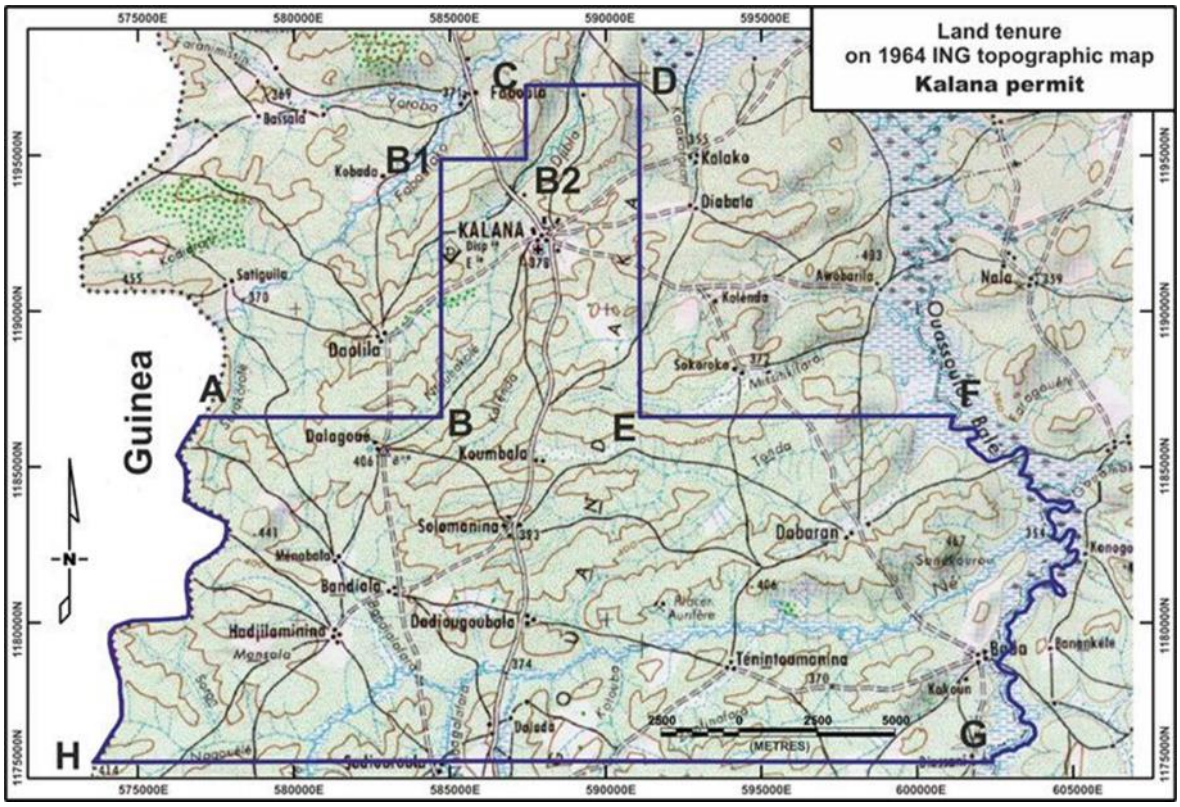
SOMIKA is subject to a royalty which is a special tax on mining products (Impôt Spécial sur Certains Produits or ISCP) at a rate of 3% and is calculated on the basis of turnover before tax. Pursuant to the stability clauses in the Company's Foundation Agreement, the 3% Ad Valorem Net Smelter Return Tax, introduced as part of the 2012 Mining Code, is not applicable to SOMIKA.

All the agreements signed with the Republic of Mali and SOMIKA or Kalana Holdings contain an arbitration clause governed by the Canadian International Resources and Development Institute (CIRDI). The details of the Agreements are included in Section 10.

1.9.1 Location of Property Boundaries

The perimeter of the Permit is presented in Figure 1.9.1. The western margins of the Permit boundaries adjoin the Guinea / Mali border and the eastern margin of the permit boundary is the contact with the Ouassoulou-Bale River.

Figure 1.9.1 Land Tenure for the Permit



1.9.2 Environmental Permitting

The area within the Permit, and more specifically, the Kalana Mine operation, is subject to a long history of exploration, mining and mineral processing, including informal mining activities.

The QPs understand that the only significant permit approvals to be required to develop the proposed Kalana Open Pit are the approval of an Environmental and Social Impact Assessment (ESIA) and its associated Environmental and Social Management Plan (ESMP).

1.10 Capital Cost Estimate

The Project capital cost estimate (CCE) was compiled by Lycopodium with input from Knight Piésold on the tailings storage facility, water infrastructure, site access roads and airstrip. Endeavour have provided project specific portions of mine establishment and facilities.

All costs are expressed in US dollars ($) unless otherwise stated and are based on 3Q20 pricing. The estimate is deemed to have an accuracy of +20/-10%.

The CCE reflects the Project scope as previously described in this study report and is summarised in Table 1.10-1.

Table 1.10-1 Capital Cost Estimate Summary ($, 3Q20, +20/-10%)

Main Area	Capital ($M)
Treatment Plant	68.3
Reagents & Plant Services	16.1
Infrastructure	43.1
Mining	16.9
Construction Distributables	18.6
Subtotal	**163**
Management Costs	21.7
Owners Project Costs	67.5
Working Capital	5.7
Subtotal	**257.9**
*Contingency	35.8
Definitive Feasibility Study	3.3
Project Total	**297.0**

Contingency for Pre-strip are included in the estimated subtotals and have not been included in the stated contingency value.

A contingency analysis has been applied to the estimate that considers scope definition, materials / equipment pricing and installation costs. Contingency applicable to various Owners inputs have been specified by Endeavour. The resultant contingency for the Project is 12.2%.

The following items are excluded from the overall project capital cost estimate:

- Duties / taxes / fees.
- Permits and licences.
- Project sunk costs.
- Exchange rate variations.

- Project escalation.

Preproduction costs that include first fills, opening stocks, preproduction labour and vendor representative costs have been included in the estimate, as well as an allowance for working capital costs to cover the first six weeks of plant operation (in Owners cost) and mining (included in pre-strip allowance) while full operating costs are being incurred with no revenue.

1.11 Process Operating Cost Estimate

Processing operating costs have been developed by Lycopodium for fresh, transition, and oxide ore and a life of mine (LOM) blend of 63% fresh, 9% transition, and 28% oxide material. It is expected that the plant will operate on a range of ore blends. The LOM processing costs are a weighted average of the various ore type processing costs based on the LOM blend and are summarised in Table 1.11-1.

Table 1.11-1 Process Plant Operating Cost Summary ($, 4Q20, ±25%)

	LOM Blend	
Proportion of LOM	**100%**	
Plant Feed t/y	**3,233,100**	
Cost Centre	**$/y**	**$/t**
Power	17,744,016	5.49
Operating Consumables	14,965,302	4.63
Maintenance	4,144,389	1.28
Laboratory	1,342,572	0.42
Process Plant Labour	4,785,318	1.48
Total Processing	**42,976,879**	**13.29**
Fixed Cost US$/y	15,397,553	
Variable Cost US$/t		8.53
Administration Labour	5,905,813	1.83
General & Administration Costs	6,929,399	2.14
Subtotal G&A	**12,835,212**	**3.97**
Total Plant Including G&A	**55,816,808**	**17.26**
Fixed Cost US$/y	28,232,765	
Variable Cost US$/t		8.53

1.12 Financial Analysis

A financial model has been built by Endeavour to include the relevant study results in order to estimate and evaluate project cash flows and economic viability. The evaluation method takes into account mill feed tonnages and grades (including dilution) for the ore and the associated recoveries, gold price, operating costs, bullion transport and refining charges, government royalties and capital expenditures (both initial and sustaining). The project has been evaluated on a 100% ownership basis, with no debt financing.

The economic model shows robust results. Applying a long term gold price of $1,500/oz on a flat line basis from the commencement of production, the pre-tax NAV 5% (Net Average Value ('NAV') at a 5% discount rate) is $498M and the pre-tax IRR is 59%. The life of mine average cash cost per ounce is $785.

The life of mine average all-in Sustaining Cost (ASIC) is $901/oz.

1.13 Risk Assessment

The Project risk assessment was carried out in a workshop setting with attendees from the Endeavour Corporate and Operations, Lycopodium, Snowden and Knight Piésold study teams. Risks identified were rated according to their likelihood and consequence, with mitigating actions identified where necessary.

The top eight risks following mitigation are summarised in Table 1.13-1 below.

Table 1.13-1 Top 8 Risks After Mitigating Actions

No.	Risk No. - Risk Description	Main Mitigation Strategy	Risk Rating
1.	G04 - Lost mineralisation	Grade control drilling. Geologist to drive mining.	17 (High)
2.	M04 - Fly rock outside of blast exclusion zone into populated area	350 m buffer which is considered adequate for safety. Drill and blast program tailored to surrounding area and pit.	15 (High)
3.	M06 - Excessive dilution causing lower grade material fed through the mill, and lower production of gold	GC drilling program to be designed accordingly. Implement adequate mining practices. Selecting appropriate fleet, contractor selection. Ensure geologist input into mining.	14 (High)
4.	GT S02 - Quality is unsuitable from identified sources	Quality to be confirmed. Aggregate will need to be sourced elsewhere. Increased costs.	12 (High)
5.	TSF01 - An event larger than the design event occurs and embankment fails	Not typically seismic area. Review TSF design.	15 (High)
6.	TSF05 - Some damage to embankment and tailings are released	Management, audits, annual reports, design standards, internal annual reviews.	15 (High)
7.	IF03 - Multi vehicle accident with fatality	Design and accident prevention plan, street lighting.	19 (Extreme)
8.	IF04 - Multiple power failures due to poor grid availability	• Solar plant & batteries as 'back-up' power has been considered. • Possibility to add additional back-up power with diesel gen-set.	12 (High)

1.14 Project Implementation Schedule

A summary of key schedule activities is listed in Table 1.14-1.

Table 1.14-1 Development Schedule

Activity	Month Start	Duration (Months)	Month Complete
DFS/43-101	Apr 2021	6	Sep 2021
FEED	Jul 2021	3	Sep 2021
Full Funding Commitment	Oct 2021	2	Nov 2021
Place Long Lead Orders	Dec 2021	9	Aug 2022
Detail Engineering, Design and Drafting	Dec 2021	10	Sep 2022
Bulk Earthworks	Apr 2022	12	Apr 2023
Process Plant Civils, SMP & E/I	Jun 2022	14	Aug 2023
Commissioning	Jul 2023	3	Oct 2023
First Gold Pour		-	Oct 2023

Key critical path activities / project risks are as shown in Table 1.14-2.

Table 1.14-2 Key Milestones

Item	Target Date	Duration from Award	
		Weeks	Months
Funding Approval	30-Nov-21	-	-
Proceed with Detailed Engineering & Drafting and Procurement	01-Dec-21	0	0
Award SAG & Ball Mill Supply	07-Dec-21	1	0
Start Bulk Earthworks	19-Apr-22	20	4.5
Start Civils Construction	21-Jun-22	29	6.5
Start SMP Installation	27-Oct-22	47	10.5
Start Electrical Installation	20-Dec-22	55	12.5
Start Plant Commissioning	26-Jul-23	86	20
First Gold Pour	31-Oct-23	100	23

The primary critical path runs through:

- Funding approval to proceed with design and procurement.
- SAG and ball mill order, procurement and delivery.
- SAG and ball mill installation.
- Installation of associated milling area mechanical, piping and electrical works.
- Subsequent commissioning through to gold pour.

Additionally, the supply of permanent power for commissioning should be considered a risk.

1.15 Recommendations for Future Work

Work undertaken to date is sufficient to support PFS level design and cost estimating; however, a number of recommendations for future work have been identified which require further evaluation prior to, or as part of any subsequent feasibility study phase. The additional work covers a number of areas and can be summarised as:

- Resource definition drilling for Kalana 3rd order veins and Kalanako close spaced drilling to improve grade continuity.
- Additional deposit exploration to the southwest of Kalanako, in the Kalanako northwest area, and in the Bandiala, Tonda and Djirila deposits.

- Regional exploration should continue on the Kalana Project permits.
- Establishment of additional pit dewatering boreholes and a review and update of the water balance once the testwork has been completed.
- Geochemical testwork on the pit water to establish the salt and water balance.
- Measures to minimise and manage the current and LoM rehabilitation and closure liabilities.
- An update of the entire Resettlement Action Plan must be completed during the DFS Phase, and prior to making a full investment decision on the Project. Items to consider include:
  - relocation of three cemeteries
  - resettlement / relocation of part of the Kalana Village
  - loss of cultivated land.
- Preparation of a ground control management plan.
- Pit geotechnical design review.
- Maintaining the basis for the Mineral Reserve Estimate.
- Detailed short-term mine planning completed as soon as practically possible following a Project development decision.
- Designing and implementing best practise ore control procedures in the Kalana open pit.
- Develop a system of mine to mill reconciliation.
- Engage suitably qualified and experienced open pit blasting personnel.
- Investigate 'hot-seat-change' procedures for equipment operators.
- Optimise pit ramp widths.
- Exploration drilling for a potential northern extension of the Kalana open pit.
- Include a comminution and metallurgical infill testwork programme in a subsequent DFS to confirm the metallurgical response and/or physical characteristics of the primary ore at deeper levels (between ~200 m and ~320 m).
- Conduct a detailed site geotechnical program for the PFS plant site and infrastructure design.

- Complete site sterilisation drilling for process plant and infrastructure.
- Review plant and infrastructure design based on the results of additional testwork outlined above.
- Conduct a groundwater investigation for the Kalanako Pit area, to be incorporated into the hydrogeological study for the project.
- Confirm the Malian Government design requirements for the Kalana Bypass Road.
- Incorporate the findings from the geotechnical investigation into the seepage and stability assessments for the TSF Dam Breach Assessment.
- Complete a population census for settlements in proximity to the project site (Kalana and Daraolia) to determine the population and areal extents of the settlements, to inform the dam breach assessment and ANCOLD consequence classification of the TSF.
- Continue to update the Project Implementation schedule to include permit approvals and lead times for equipment and design and construction of facilities.

2.0 INTRODUCTION

In August 2020 Endeavour Mining Limited (Endeavour) engaged various consultants to undertake the scope of work for a Pre-Feasibility Study (PFS) for their Kalana Project, located approximately 250 km south of the capital of Mali, Bamako.

2.1 Terms of Reference

Endeavour Mining Limited (Endeavour) is developing the Kalana and Kalanako deposits situated within the Kalana Exploitation Permit, located in Mali. Endeavour engaged Lycopodium Minerals Canada Pty Ltd (Lycopodium) to coordinate and prepare an independent Technical Report on the Kalana Gold Project (the Project) to the standard of the Canadian National Instrument 43-101 'Standards of Disclosure for Mineral Projects' following the completion of a Pre-Feasibility Study (PFS) covering the Project in January 2021.

2.2 Sources of Information

This Report relies on historic and recent data generated by Endeavour. Endeavour has engaged several specialist consultants and information from reports prepared by previous independent consultants has been utilised in the compilation of this report. Section 27 provides a list of references relied upon in preparation of this report.

2.3 Cautionary Notes

This Report has been compiled based on information available up to and including the date of the Report. The status of agreements, royalties or tenement standing pertaining to the assets, have not been investigated by contributing consultants and were not required to be. All matters relating to ownership are to be directed to Endeavour for clarification if required.

2.4 Contributing Consultants

Lycopodium has provided engineering and project management services to the international mining industry for over 20 years.

Sections of this report were authored or co-authored by Qualified Persons from Optiro, Knight Piésold, Snowden Mining and Endeavour Mining.

Optiro is a resources consulting and advisory group based in Perth, Western Australia. Optiro provided the mineral resource estimate for Kalana and Kalanako.

Snowden Mining is a mining and geological consulting group based in Perth, Western Australia. Snowden provided the mining design and the mineral reserve estimate.

Knight Piésold is an international firm of consulting engineers and scientists with over 90 years of experience in the mining and power sectors. The Australian offices of Knight Piésold conducted studies for tailings management, water supply, surface water management and groundwater evaluations for the Project.

2.5 Qualified Persons

At the request of Endeavour, Lycopodium has been mandated to prepare a NI 43-101 Report for the Project with the participation of specialised consultants. Table 2.5-1 provides a detailed list of qualified persons as defined in Section 1.5 of NI 43-101 and their respective sections of responsibility.

Table 2.5-1 Qualified Persons / Responsibility

Section	Title of Section	Qualified Person
1	Summary	All Related QPs
2	Introduction	David Gordon
3	Reliance on Other Experts	David Gordon
4	Property Description and Location	Helen Oliver
5	Accessibility, Climate, Local Resources, Infrastructure and Physiology	Helen Oliver
6	History	Helen Oliver
7	Geological Setting and Mineralisation	Helen Oliver
8	Deposit Types	Helen Oliver
9	Exploration	Helen Oliver
10	Drilling	Helen Oliver
11	Sampling Preparation, Analysis and Security	Helen Oliver
12	Data Verification	Helen Oliver
13	Mineral Processing and Metallurgical Testing	David Gordon
14	Mineral Resource Estimates	Paul Blackney & Helen Oliver
15	Mineral Reserve Estimates	Allan Earl
16	Mining Methods	Allan Earl
17	Recovery Methods	David Gordon
18	Project Infrastructure	David Gordon & David Morgan
19	Market Studies and Contracts	Patrick Pérez
20	Environmental Studies, Permitting and Social or Community Impact	Patrick Pérez
21	Capital and Operating Costs	David Gordon & Patrick Pérez
22	Economic Analysis	Patrick Pérez
23	Adjacent Properties	Helen Oliver
24	Other Relevant Data and Information	All related QPs
25	Interpretation and Conclusions	All related QPs
26	Recommendations	All related QPs
27	References	All related QPs

2.6 Effective Date and Declaration

The effective date of this Technical Report is 31 December, 2020. There were no material changes to the information on the Project between the effective date and the signature date of the Technical Report.

2.7 Site Visits and Inspections

Due to the onset of the COVID-19 global pandemic in early 2020, and prior to the commencement of the PFS, travel restrictions were put in place, that have prevented travel to the Project area to date.

Despite the requirements for preparation of a NI 43-101 Technical Report, not all competent persons (CPs) have visited the project site at this point and have relied on the evidence of other contributors to the report, including Lycopodium, whose representatives have not visited the site prior to travel restriction being put in place. Lycopodium is of the opinion that the pandemic qualifies as "exceptional circumstances" under NI 43-101 site visit requirements.

Helen Oliver, Endeavour Mining Group Resource Geologist regulary visited Kalana from the autumn of 2017 until late 2019.. (including in October 2017, February 2018, April 2018, September 2018, November 2018, February 2019, May 2019 and October 2019).

Patrick Pérez, Endeavour Mining Director of Technical Studies, has not visited the site, but he has previously completed studies and projects in Mali and is familiar with the country and region.

Lycopodium QP Mr David Gordon (Lycopodium) has not visited site; however, Mr Alastair Holden, Principal Process Engineer with Lycopodium and a representative of Mr David Gordon undertook a site visit to Kalana (22 October 2019 to 2 November 2019) to supervise the metallurgical sample selection and packing for shipment. Lycopodium has previously completed studies and projects in Mali and is familiar with the country and region.

Mr David Morgan of Knight Piésold has undertaken a site visit, he has previously completed studies and projects in Mali and is familiar with the country and region and has routinely travelled to other Endeavour assets.

Mr Allan Earl of Snowden Mining has visited the site, and had previously completed studies and projects in Mali.

2.8 Units and Currency

Unless stated otherwise, Le Système International d'Unités (SI) units have been used throughout the Report (note, that some more commonly used non-metric units have been retained for ease of understanding, e.g. gold tenors are reported in troy ounces in some instances).

Currencies used in the report are US dollars, unless noted otherwise. Conversion rates from local or other currencies to US dollars used in cost estimates or financial analyses are reported in the relevant sections.

2.9 List of Abbreviations

The following terms and abbreviations are used:

ºC	degrees Celsius
>	greater than
≥	greater than or equal to
<	less than
≤	less than or equal to
'	minute
"	second
%	percent
®	Registered trademark
AAS	Atomic Absorption Spectrometry
ABA	Acid base accounting
AC	Acid Consuming (tailings)
AEP	Annual Exceedance Probability
Ag	Silver
Ai	Abrasion Index
AIA	Archaeological Impact Assessment
ANC	Acid neutralising capacity
ARI	Annual Return Interval
As	Arsenic
AS	Acid Soluble (Cu analysis method)
ASCII	American Standard Code for Information Interchange
ASL	Above Sea Level
batter	The slope on the side of cuttings or on dump or on walls
BBWi	Bond ball mill work index
BLEG	Bulk Leach Extractable Gold
BOCO	Base Of Complete Oxidation
BOO	Build, Own and Operate
BRWi	Bond rod mill work index
BTRANS	Base of Transitional
burden	The distance between drill rows within a blast pattern
C1 Costs	direct costs, which include costs incurred in mining and processing (labour, power, reagents, materials) plus local G&A, freight and realisation and selling costs. Any by-product revenue is credited against cost.

CAPEX	Capital expenditure
Cat	Caterpillar
CCTV	Closed circuit television
CIF	Carriage and Insurance Free
CRM	Certified Reference Material
CSL	Compacted Soil Liner
CSR	Corporate Social Responsibility
CSS	Closed Side Setting
CSV	Comma Separated Values (Spreadsheet)
Cu	Copper
CWi	Crushing Work Index
D&C	Design and construct
DDH	Diamond Drill Hole
DMT	Dry Metric Tonne
DTM	Digital Terrain Model
EBITDA	Earnings Before Interest Taxes Depreciation Amortization
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EOI	Expression of interest
EPC	Engineer, Procure, and Construct
ESIA	Environmental and Social Impact Assessment
F_{80}	Feed 80% passing size
FEED	Front end engineering design
FEL	Front end loader
FPIC	Free, prior and informed consent
FR	Fresh Rock
FS	Feasibility study
FY	Financial Year
GA	General Arrangement
G&A Costs	General and Administration Costs
GET	Ground Engaging Tools
GHG	Greenhouse Gas
GMP	Groundwater Management Plan
GPO	General Purpose outlet
GPS	Global Positioning System
GST	Goods and services tax
HAZAN	Hazard analysis
HAZID	Hazard identification study
HAZOP	Hazard and operability study

HDPE	High-density polyethylene
HG	High Grade
HIA	Health Impact Assessment
HQ	Diamond drill core with core diameter of 63.5 mm
ID	Internal Diameter
IFC	International Finance Corporation
In situ	In the natural or original position
IRR	Internal Rate of Return
IWL	Integrated Waste Landform
JORC	Joint Ore Reserves Committee
JSA	Job safety analysis
JSO	Job safety observation
KP	Knight Piésold
Kt	Thousand tonnes
Koz	Thousand ounces
LIBOR	London Inter-Bank Offered Rate
LOM	Life of mine
Magazine	Building used to store explosives
MAR	Mean Annual Rainfall
masl	Metres above Sea Level
mbgl	Metres Below Ground Level
MBM	Mining Block Model
MCAF	Mining Cost adjusted with Factors
MCC	Motor control centre
MCE	Maximum Credible Earthquake
MIA	Mine Industrial Area
MG	Medium Grade
MPA	Maximum potential acidity
MRE	Maiden Resource Estimate
Mt	Million tonnes
Mlb	Million pounds
MMW	Minimum Mining Width
MRM	Mineral Resource Model
MW	Mineralised waste
MWD	Mineralised Waste Dump
MW-HW	Moderately to Highly Weathered (rock)
MWS	Mineralised waste stockpile
NGO	Non-government organisations
NPV	Net present value
NQ	Diamond drill core with core diameter of 47.6 mm

OEM	Original equipment manufacturer
OHRA	Occupational Health Risk Assessment
OIS	Operator interface system
OP	Output
OPEX	Operations expenditure
OSA	Overall Slope Angle
P_{80}	Product 80% passing size
P&ID	Piping and Instrumentation Diagram
PCAF	Processing Cost Adjusted with Factors
PDS	Project definition statement
PFD	Process Flow Diagram
PFS	Pre-feasibility study
PGA	Peak ground acceleration
PLC	Programmable logic controller
PMP	Project Management Plan
PMP	Probable Maximum Precipitation
PPE	Personal Protection Equipment
PV	Process Variable
QA	Quality Assurance
QC	Quality Control
QKNA	Quantitative Kriging Neighbourhood Analysis
RAD	Remote Area Dwellers
RC	Reverse Circulation
RF	Revenue Factor
RFP	Request for Pricing
RL	Relative Level to sea level
RoM	Run of mine
rpm	Revolutions per minute
RQD	Rock Quality Designation
RTK GPS	Real Time Kinematic GPS
SABC	SAG and ball mill with pebble crusher
SAG	Semi autogeneous grinding
SCADA	Supervisory control and data acquisition
SD	Standard Deviation (statistical)
SG	Specific Gravity
SLD	Single Line Diagram
SMU	Selective Mining Unit
SOP	Standard operating procedure
SP	Set point
SUBS	Subsoil Stockpile

SWL	Static Water Level
TDS	Total Dissolved Solids
TMM	Total Material Movements
ToR	Terms of reference
TSF	Tailings storage facility
TSF-C	Tailings Storage Facility zone C
TSP	Total suspended particles
UCS	Unconfined compressive strength
UPS	Uninterruptable Power Supply
UTM	Universal Transverse Mercator
UV	Ultraviolet
VTC	Vocational Training Centre
VWP	Vibrating Wire Piezometer
WAB	Water Apportionment Board
WAP	Work area pack
WBS	Work breakdown structure
ww	Wedge wire screen mesh (parallel aperture)
WRD	Waste rock dump
XRF	X-ray Fluorescence
US$	United States Dollar

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

While information provided by Endeavour relating to mineral rights, surface rights and permitting has been reviewed, no opinion is offered in these areas. The Qualified Person is not expert in land, legal, permitting, and related matters and therefore has relied upon, and is satisfied, there is a reasonable basis for this reliance on the information provided by the Endeavour regarding mineral rights, surface rights and permitting in Section 4 of this Technical Report. The authors of this Technical Report, state that they are Qualified Persons for the areas as identified in the Certificate of Qualified Person attached to this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Kalana Property

The Kalana Project is located 250 km south of Bamako, in southwest Mali, see Figure 4.1.1. Mali is a land-locked country in West Africa, and is bordered to the east by Niger, to the north by Algeria, to the west by Mauritania and Senegal, and to the south by Guinea, Cote d'Ivoire and Burkina Faso. The Kalana Project area is located in southwest Mali and is bound to the west by the Mali-Guinea border and to the east by the Ouassoulou-Bale River. It is situated south of the Kodieran Gold Mine and proximal to Kalana town.

Figure 4.1.1 Location of the Kalana Project Area



The Kalana Project area was explored for gold by two Malian National companies: Société Nationale de Recherche et d'Exploitation Minière (SONAREM) and Société de Gestion et d'Exploration des Mines d'Or de Kalana (SOGEMORK), as part of the Soviet Union Technical Assistance Program (STAP) to Mali between 1962 and 1982. In 1982, a decision was taken to develop the Kalana Mine as a small, fresh rock only, underground mine using a recovery process based on crushing, milling, and gravity concentration. In 1984, a Permit was created in the favour of SOGEMORK for a period of 15 years, renewable at five-year intervals thereafter up to a maximum of a 30-year term.

Production from the Kalana Mine commenced in 1985 and a total of 227 thousand tonnes (kt) of ore was treated at an average head grade of 13 Au g/t to produce 81,000 oz of gold by August 1991. In 1991, technical and financial resources were withdrawn from the Kalana Mine following the dissolution of the Soviet Union. This resulted in the Kalana Mine being placed on care and maintenance. The Permit reverted to the Malian Government following the dissolution of SOGEMORK in February 1992.

Investigations of the mineralisation in the Permit area have been subsequently undertaken by various companies, including Ashanti Gold Fields Company Limited (Ashanti) (1995 to 1997). In 2003, Avnel was granted the Permit.

In 2004, SOMIKA (Avnel) recommenced underground gold production at Kalana Mine. From 2004 to 2015, Kalana Mine produced 0.17 million ounces (Moz) of gold from 531 kt at an average grade of 11.5 Au g/t with 86% average recovery.

The Project area contains the developed Kalana Mine site, which has underground and historic open pit mine workings, a gravity processing plant, offices, stores and accommodation for non-local mine employees.

The mining industry is well developed in Mali, with several gold mining operations, and one of the latest developments – Yanfolila – is located only 50 to 60 km away from Kalana.

4.2 Issuer's Interest

Kalana Holdings (formerly known as Avnel Gold Ltd.), a wholly-owned subsidiary of Endeavour Mining (as noted in Section 2), has an 80% interest in SOMIKA, the holder of the Permit. The Government of Mali holds the remaining 20% beneficial interest in SOMIKA.

4.3 Mineral Tenure

The Project comprises one Exploitation Permit (Permis d'Exploitation), registered to SOMIKA that grants the exclusive right of exploitation and exploration.

The permit derived from legislation passed to enable the Soviet Union aided state company SOGEMORK, to develop the Kalana underground mine.

The Permit PE 001/84 Bis (decree n°305 PG-RM dated of 17 December 1984) was granted to SOGEMORK, a Malian national company, in 1984 for a period of 15 years, renewable at five-year intervals thereafter up to a maximum of a 30-year term. SOGEMORK was liquidated in February 1992 (decree n°92-049 PM-RM).

The Permit was then transferred to Ashanti-JCI on 30 November 1995 by decree n°95-419 PM-RM and the duration was extended to 30 years from 1995. As Ashanti-JCI had failed to meet its contractual commitment to put the Soviet built Kalana underground mine back into production within the prescribed three-year term, the Malian Government issued a decree (n°99-055 PM-RM dated of 18 March 1999) cancelling the transfer of the Permit to Ashanti-JCI.

On 18 March 1999, the Malian Government re-issued an international invitation to tender for the Permit in July 1999. The successful company did not meet its financial commitments, as a result of which the tender was cancelled and the Permit was not issued.

In response to an international invitation to tender of the Malian government, in October 2001, Avnel Group submitted a technical bid and a financial bid for the Permit. The Avnel Group originally lost the tender to another company. Management of Avnel Group was subsequently informed by the Malian government, that the successful bidder had defaulted on its financial obligation and that its tender rights were cancelled. As a result, the Malian Minister of State Domain formally awarded the tender to Avnel Group on 23 December 2002 by letter n°272/MDEAFH-SG.

In April 2003, the Permit PE 001/84 Bis was renewed and transferred to Kalana Holdings. according to decree n°03147/PM-RM dated 7 April 2003 and was simultaneously reinstated for a new term of 30 years.

SOMIKA was formed on 5 August 2003 and the Permit then transferred by Kalana Holdings to SOMIKA by Decree n°03-579/PR-RM dated 30 December 2003.

The Permit confers the right to exploit and explore for gold and silver for a period of 30 years (Table 4.3-1). According to Article 43 of the 1999 Mining Code, an exploitation permit is granted by decree for a 30-year period, renewable for a further 10-year period until depletion of the reserves contained within the boundaries of the permit.

According to the provisions of the 1999 Mining Code an Exploitation Permit may be transferred or leased. The transfer or lease of a mining permit may only come into effect if it has been authorised by Decree. In the absence of a provision restricting the division of the permit (and there are no such provisions in the Permit), then having regard to the size of the Permit and the different deposits it may contain, it may be possible to divide it into several separate permits, provided that the provisions of the 1999 Mining Code regarding the assignee are respected.

Table 4.3-1 Mineral Tenement Information

Type of Mineral Tenure	Name	Number	Registered To	Grant Date	Expiration Date	Area (km^2)
Permis d'Exploitation	Kalana	PE 001/84 Bis	SOMIKA	7 April 2003	7 April 2033	387.4

Source: Avnel, 2014

The Permit is unique to Mali as it is a combined exploration and exploitation permit that is grandfathered under the 1999 Mining Code. Under current Malian law, exploitation permits are normally granted exclusively to exploration companies in respect of areas covered by a complete and approved Feasibility Study. Additionally, exploration permits are normally granted for a limited time (usually three years, twice renewable on three-year terms), with a set amount of minimum required spending and on the basis that the area covered by the exploration permit halves each time it is renewed.

During exploration, the construction of the proposed Kalana Main Mine and for three years following commercial production, SOMIKA will be exempt from VAT and certain custom duties related to the Project. This exemption will apply to any one or more further mines discovered and developed on the Permit.

Following the three-year period from the commencement of commercial production, an 18% value added tax (VAT), or 'TVA', will be applicable on goods and services, which is fully reimbursable, and customs duty will be applicable on imported goods.

As SOMIKA elected to stay under the old CIT regime in force before Loi de Finance 2012, the tax structure in Mali applicable to its operations consists of a general corporate income tax rate of 35% of profits after deductions for expenses and allowable depreciation.

4.4 Agreements

All the agreements signed with the Republic of Mali and SOMIKA or Kalana Holdings contain an arbitration clause governed by the Canadian International Resources and Development Institute (CIRDI).

4.4.1 Foundation agreement

On 14 February 2003, Kalana Holdings and the Malian Government entered into the Foundation Agreement. The Foundation Agreement sets out a three-stage framework for activities in relation to the Permit and the concession:

- Recommencement of mining of Kalana deposit (Stage 1) including rehabilitation of the Kalana mine and infrastructure and recommencement of mining operations.
- Development of the Kalana deposit (Stage 2) in order to exploit resources and reserves.
- Research, development and mining all economically viable deposits on the surface area of the entire concession, other than the Kalana Main deposit, as defined by the Direction Nationale de la Géologie et des Mines (DNGM) in 2015 (Stage 3).

The Foundation Agreement, governed by the 1999 Mining Code, includes the following key terms:

- The Malian Government's 20% shareholding in SOMIKA has a priority dividend right in respect of profits and available for distribution.
- SOMIKA is granted a stability guarantee in respect of tax and custom duties, and is also required to pay specified annual royalties to the Malian Government.

The Foundation Agreement provides for various tax exemptions and provides sovereign protection from adverse changes in the fiscal regime.

On 17 February 2015, Kalana Holdings. and the Malian Government entered into an Avenant to the Foundation Agreement by adding Articles 10-2 and 10-3, in order to segregate the exploration expenses and the exploitation expenses made on the Permit. The fiscal regime applicable to exploration works, following the 1999 Mining Code and the Foundation Agreement, is different from the fiscal regime applicable to exploitation works.

Consequently, a branch of SOMIKA (with the same tax number): 'SOMIKA Exploration' dedicated to all the exploration works made on the Permit, was formed to manage all the exploration expenses and enabling SOMIKA, according to the Foundation Agreement and the 1999 Mining Code, to benefit from the special fiscal regime applicable to exploration works.

4.4.2 SOMIKA's Shareholders' Agreement

Under the terms of the Foundation Agreement, the Malian Government is entitled to a 20% stake in SOMIKA. On 28 July 2003, the Shareholders' Agreement between the shareholders of SOMIKA was approved and executed by Kalana Holdings and State of Mali, represented by the Minister of State domains of Mali.

Key features of the Shareholders' Agreement include the following:

- The Board of SOMIKA consists of seven directors, of which five are nominated by Kalana Holdings and two are nominated by the Government.
- All expenditures by Kalana Holdings from the tender date to the incorporation of SOMIKA and the total amount in respect of the entry fee, tangible assets fee and reserve fee (totalling US$2.5 million) are treated as a loan by Kalana Holdings to SOMIKA.
- Loans among shareholders which are not the subject of a specific agreement reached at Board level are to bear interest at the London interbank offered rate (LIBOR) plus 2%. This is applied to Kalana Holdings shareholder loan to SOMIKA, with initial loans set at the LIBOR rate prevailing at the time of signing.
- Loan accounts which are the subject of specific agreements between SOMIKA and third parties are to bear interest at full commercial rates for loans to Mali.
- Assignment by either of Kalana Holdings or the Malian Government may not be made without the prior written consent of the other shareholder.
- The distribution of dividends is fixed by the Board and is subject to the laws of Mali and the Foundation Agreement, which provide for a priority distribution right in favour of the Malian Government.

4.4.3 Location of property boundaries

The perimeter of the Permit is presented in Figure 1.9.1 and defined in Figure 4.4.1. The western margins of the Permit boundaries adjoin the Guinea / Mali border and the eastern margin of the permit boundary is the contact with the Ouassoulou-Bale River.

Figure 4.4.1 Land Tenure for the Permit

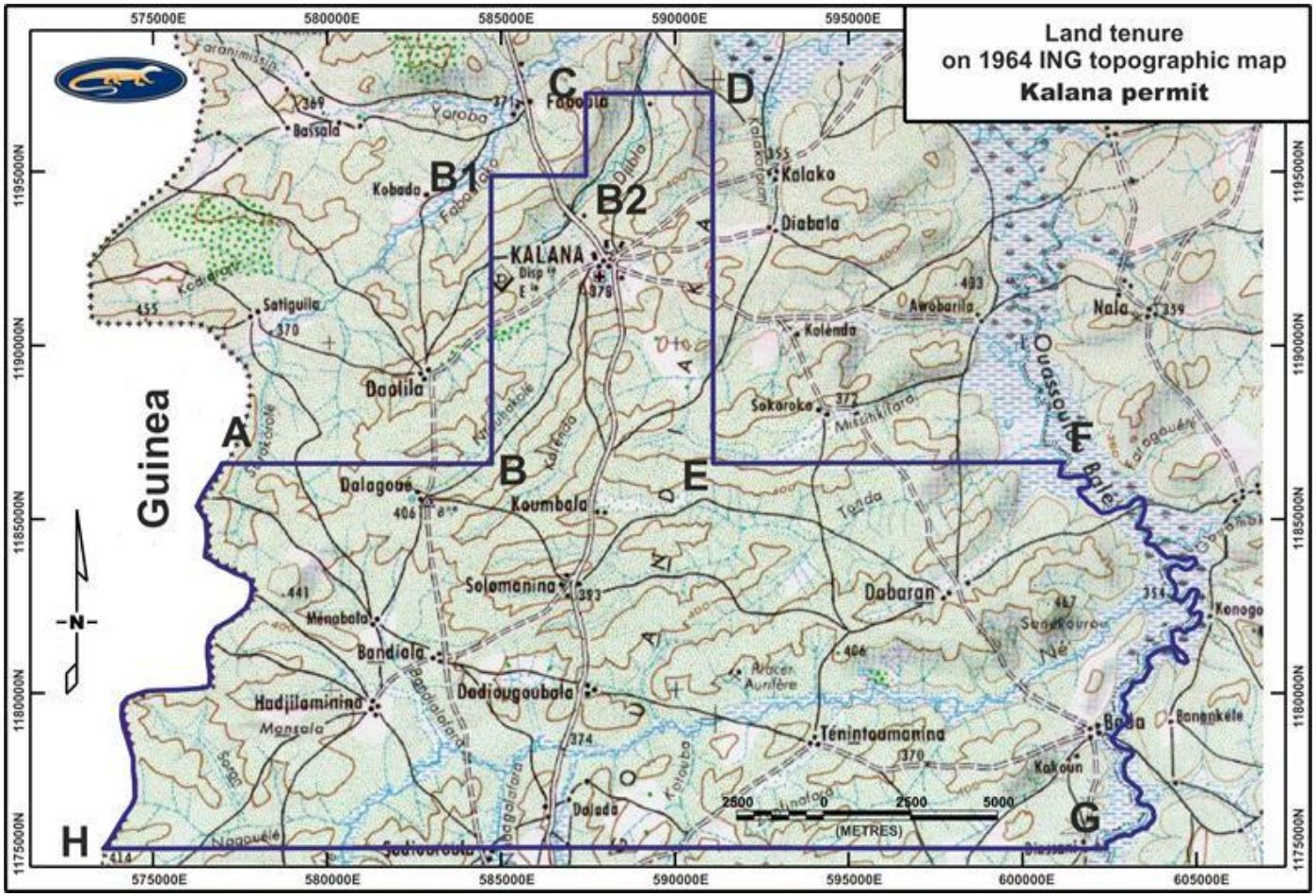


Table 4.4-1 Definition of the Perimeter of the Permit

Vertex	Location of point
Point A:	Intersection de la frontière Guinéo-Malienne avec le parallèle 10°44'00''N. Du point A au point B suivant parallèle 10°44'00''N.
Point B:	Intersection du parallèle 10°44'00''N avec le méridien 8°13'30''W. Du point B au point B1 suivant le méridien 8°13'30''W.
Point B1:	Intersection du méridien 8°13'30'' W avec le parallèle 10°48'30'' N. Du point B1 au point B2 suivant le parallèle 10°48'30''N.
Point B2:	Intersection du parallèle 10°48'30 N avec le méridien 8°12'00''W. Du point B2 au point C suivant le méridien 8°12'00''W.
Point C:	Intersection du méridien 8°12'00''W avec le parallèle 10°49'47'' N. Du point C au D suivant du parallèle 10°49'47''N.
Point D:	Intersection du parallèle 10°49'47'' N avec le méridien 8°10'00''W. Du point D au E suivant le méridien 8°10'00''W.
Point E:	Intersection du méridien 8°10'00''W avec le parallèle 10°44'00''N. Du point E au F suivant du parallèle 10°44'00''N.
Point F:	Intersection du parallèle 10°44'00''N avec la rivière du Ouassoulou-Balé. Du point F au G suivant le Ouassoulou-Balé.
Point G:	Intersection de la rivière Ouassoulou-Balé avec le parallèle 10°37'47''N. Du point G au H suivant du parallèle 10°37'47''N.
Point H:	Intersection du parallèle 10°38'00'' N avec la frontière Guinéo-Malienne. Du point H au A suivant la frontière Guinéo-Malienne.

Note: Coordinates in Lat-Long (WGS84)
Source: Avnel, 2014

4.5 Royalties, Back-in Rights, Payments, Agreements, Encumbrances

The following is a general description of mining rights in Mali not withstanding grandfathered or otherwise special regulations.

In the Republic of Mali, all mineral rights are held by the State and are administered by the DNGM on behalf of the Department of Mines Energy and Water.

Mali's mineral law is based on French civil law. The Mining Code was revised in 1991,1999, 2012 and 2019. For the purpose of SOMIKA, Mining Code is the Ordinance n° 99-032 P-RM dated 19 August 1999 as amended by Ordinance n°00-013/P-RM dated 10 February 2000 and Regulations Decree n°99-255/P-RM dated 15 September 1999 (the Mining Law). The Kalana Main Project was established under the 1999 Mining Code. Standard agreements are available. A scale of fees based on area is applied to mineral licences.

A Foundation Agreement (Convention d'Etablissement) is signed between an investing company (whether foreign or domestic) and the Malian government before exploration or mining commences. The agreement, negotiated between the parties, comprehensively details all the conditions that will apply to exploration activities and, in the event of a discovery, mineral exploitation activities. The conditions include work obligations, reporting, taxes, duties, duty-free arrangements and state equity participation of up to 20%.

According to 2019 Mining Code, an exploitation permit (permis d'exploitation) is typically granted by decree for a maximum 12-year period and can be renewed by consecutive 10-year periods. The area of an exploitation permit cannot exceed that of the exploration permit from which it is derived. Fees are payable when an exploitation mining permit is granted or renewed. There are also annual rents due based on the area of the permit.

During the construction of the proposed Kalana Main Mine and for three years following commercial production, SOMIKA will be exempt from VAT and certain custom duties related to the Project. Following the three-year period from the commencement of commercial production, an 18% VAT will be applicable on goods and services, which is fully reimbursable, and customs duty will be applicable on most imported goods.

As SOMIKA elected to stay under the old CIT regime in force before Loi de Finance 2012, the tax structure in Mali applicable to its operations consists of a general corporate income tax rate of 35% of profits after deductions for expenses and allowable depreciation. SOMIKA is also subject to a minimum tax rate of 1.0% of net revenue, even in years with a deficit.

SOMIKA is subject to a royalty which is a special tax on mining products (Impôt Spécial sur Certains Produits or ISCP) at a rate of 3% and is calculated on the basis of turnover before tax. Pursuant to the stability clauses in the Company's Foundation Agreement, the 3% Ad Valorem Net Smelter Return Tax, introduced as part of the 2012 Mining Code, is not applicable to SOMIKA.

4.6 Environmental Permitting

The area within the Permit, and more specifically, the Kalana Mine operation, is subject to a long history of exploration, mining and mineral processing, including informal mining activities.

The QPs understand that the only significant permit approvals to be required to develop the proposed Kalana Open Pit are the approval of an Environmental and Social Impact Assessment (ESIA) and its associated Environmental and Social Management Plan (ESMP).

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

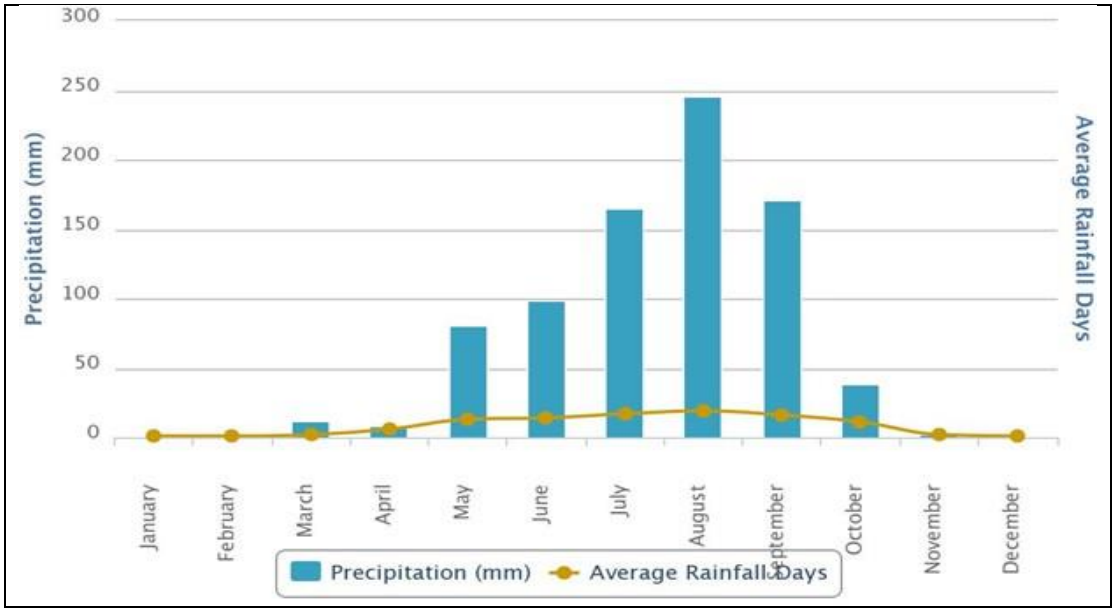
The Kalana Project is located 250 km south of Bamako, the capital of Mali. Access to the Project from Bamako is via sealed highway all year round. There is an international airport at Bamako with daily direct flights to many European and West African countries.

5.2 Climate

The Kalana Project is located in a sub-tropical climate. The dry season occurs between October and April with most of the rainfall during July, August and September (Figure 5.2.1). Annual rainfall at Kalana is between 1,000 mm and 1,200 mm. Temperatures range from 20°C to 28°C in the cooler months and 15°C to 37°C in the hotter months (Figure 5.2.2).

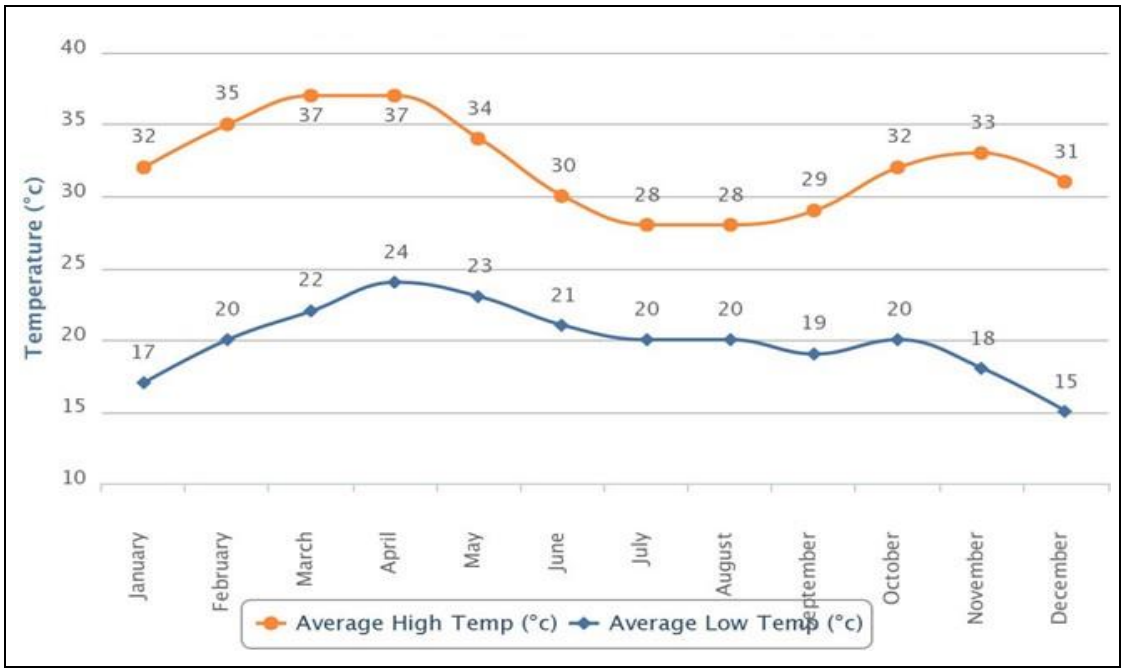
The operating season for the Kalana Project is all year round (subject to an average of 12 days lost per year due to rain and weather-related delays).

Figure 5.2.1 Average Rainfall in Kalana Project Area



Source: Snowden, 2014

Figure 5.2.2 Average Temperature in Kalana Project Area



Source: Snowden, 2014

5.3 Local Resources and Infrastructures

The Project site is within one kilometre of Kalana town, where approximately 14,000 people reside. Kalana town has access to electric grid power, a potable water supply system, medical and schooling facilities. Water is available from underground sources for mining operations with excess water from the underground workings discharged. Water for community consumption is supplied by community pumps and boreholes. Power is supplied through Mali's national power grid along an overhead line (OHL).

5.4 Physiography, Topography, Elevation and Vegetation

Mali contains primarily savannah in the south and flat to rolling plains or high plateaus (200 m to 500 m in elevation) in the north. There are rugged hills in the north-east with elevations of up to 1,000 m. The topography within the Project area undulates between 350 m above mean sea level (mamsl) and 440 mamsl.

Vegetation within the Project area is characterised as sub-tropical, (the Sudanian Savannah) and characterised by the coexistence of trees and grasses indigenous to the area.

6.0 HISTORY

The historical milestones of the Kalana Project can be summarised as:

- Underground gold mining of the Kalana deposit with gravity-only concentration by the SONAREM-SOGEMORK between 1982 and 1991, producing approximately 81,800 gold ounces from 227,000 tonnes mined at an average grade of 13 Au g/t with a recovery of 86%.

- Underground mining recommenced in 2004 by SOMIKA (of which Avnel was the majority shareholder). Mining stopped in September 2017 and processing ceased in December 2017; 165.9 koz of gold ounces at an average grade of 11.7 Au g/t were produced by gravity concentration only.

- A Joint Venture with IAMGOLD between 2009 and 2013 invigorated the exploration activities on the Project with a significant amount of work undertaken.

- The introduction of the LeachWELL assay protocol and the subsequent up-grade in the measurable gold content in 2013 significantly enhanced the viability of the Project.

- Publication of a Feasibility Study Technical Report by Avnel to develop the Project as a conventional open-pit mine using trucks and excavators (developed in 12 stages at Kalana, Kalanako was not included). The Study was based on an average mining rate of 12 Mtpa with a Life-of-Mine of 18 years, including six months pre-stripping and processing of stockpile material for three months. Processing was to involve a gravity and Carbon-in-Leach (CIL) plant with an average throughput of 1.26 Mtpa producing 1.8 Moz gold over an 18 year Project Life.

- Purchase of Avnel, and therefore SOMIKA and the Kalana Project, by Endeavour Mining in 2017.

6.1 Historical Ownership

The following is a summary of the major ownership events in the history of the Kalana Project. Further details can be found in the 2016 Technical Report.

- 1967 to 1982 - Exploration by two Malian national companies: Société Nationale de Recherche et d'Exploitation Minière (SONAREM) and Société de Gestion et d'Exploration des Mines d'Or de Kalana (SOGEMORK), as part of the Soviet Union Technical Assistance Programme (STAP) to Mali.

- 1982 - Kalana underground mine opened with gravity recovery only.

- 1991 - Kalana underground mine closed and placed on care and maintenance.

- 1992 - SOGEMORK dissolved and the permit reverted to the State.

- 1994 - Malian Government launch a Kalana privatisation tender.
- 1995 - Ashanti and JCI formed a joint venture (Ashanti-JCI) to evaluate the Kalana Gold Mine and conduct regional exploration in the permit area.
- 1996 - JCI withdrawal from the joint venture.
- 1997 - Ashanti mandated Rothschild Natural Resources LLC to seek a suitable mining company to develop, operate and acquire a majority interest in the Permit. Nelson Gold Limited conducted a Feasibility Study which was not ratified by the Malian Government
- 1999 - The Malian Government terminated Ashanti-JCI's right to the Permit and re-issued the tender process.
- 2001 - The Malian Government re-issued the tender process.
- 2003 - Avnel Cayman (formed by Elliot Associates and the Fern Trust which held the intellectual property of Nelson Gold) awarded the Kalana Permit.
  - SOMIKA incorporated as an 80% subsidiary of Avnel with the Malian Government holding a 20% free carry interest.
  - Avnel transfer the Permit to SOMIKA.
- 2009 - IAMGOLD-Avnel joint venture agreement signed. The option agreement provided IAMGOLD the opportunity to earn an initial 51% interest in Avnel's 80% share of the Kalana Project by spending a minimum of US$11 million on exploration activities over three years and by delivering a NI 43-101 compliant Technical Report for a Mineral Resource of at least 2 Moz Au and a feasibility study.
- 2013 - IAMGOLD-Avnel joint venture lapsed. IAMGOLD spent approximately US$32.5 million on exploration activities at Kalana but did not deliver a Mineral Resource that met the 2 Moz Au vesting threshold. This resulted in the automatic lapse of the option agreement. The costs incurred by IAMGOLD were represented by quasi-equity loans to Avnel Cayman, which were lent to SOMIKA. On expiry of the option agreement, IAMGOLD's loans to Avnel Cayman were forfeited. IAMGOLD has no interest in the Permit or SOMIKA.
- 2017 - Purchase by Endeavour of Avnel Gold Mining Ltd under an all-share transaction for a consideration of approximately US$122 million in September 2017.

6.2 Exploration History

Artisanal mining exploiting alluvial gold in the region dates back to at least the 1300's under the reign of the Malian Emperor, Musa Keita I. The first modern report of gold within the Permit area was published in 1931 by a geologist conducting regional exploration during which a high grade quartz-gold vein was found in an artisanal working, which is now known as Vein 1.

The following is a summary of the exploration activities on the Kalana Permit. A significant enhancement for the Project was the introduction of the LeachWELL assay protocol in 2013.

6.2.1 SONAREM & SOGEMORK, 1967 to 1991

SONAREM-SOGEMORK undertook systematic exploration on the Kalana Permit between 1967 and 1991, including 871 core drill holes, see Table 6.2-1. Note: the Russians selectively sampled the visible gold veins.

Table 6.2-1 Russian Drilling Summary

Campaign	Company	No. Holes	Type	Total Metres
1967 – 1982	SONAREM & SOGEMORK	815	Core	81,524
1989 - 1991	SOGEMORK	56	Core	16,000
Total		**871**		**97,524**

SONAREM – SOGEMORK also conducted exploration (including 16 core holes) in the Kalanako area.

Exploration activities were also carried out on grids in the southern part of the Permit area (such as Solomanina, Tonda, Dabaran, East Dabaran, Nigrimbala, Zamirila, and Djirila I and II). Most of the work involved mapping, selective sampling and ground geophysics. No systematic soil geochemistry and only limited drilling was conducted.

6.2.2 United National Development Programme, 1980's

In the 1980s, the United Nations Development Programme funded the Or-Bagoé regional soil geochemical survey of Mali. The survey targeted the Kalana district in 1988 and covered an area of 1,076 km^2 situated east of the Guinean border, west of the Ouassoulou-Balé River and south of the 11th degree parallel. A total of 5,268 soil samples were collected from this area, which has a dimension of approximately 60 km (north-south) by 25 km to 30 km (east-west). The Kalana Permit is located approximately in the centre of the survey area.

The survey grid utilised east-west oriented lines spaced one kilometre apart and sampling was conducted at intervals of 200 m (east-west). The soil samples were collected at a depth of 30 cm, sieved through -80 mesh (screen aperture of 180 microns or µm) and analysed for eight elements (gold, copper, lead, zinc, chromium, nickel, lithium and molybdenum).

The Or-Bagoé data identified clustered gold anomalies that coincide with known large occurrences (e.g. Kalana, Kodieran, Kalanako, and Kolenda). Smaller gold occurrences (e.g. Djirila and Zamirila) could also be identified in the data. ("Batch problems" and "line effects" were apparent in the data, but SOMIKA (Avnel) did not consider these to be significant enough to compromise the usefulness of the data and utilised this data for target generation.)

A number of geological features, pertinent to gold mineralisation, can be interpreted from the Or-Bagoé survey. A preferred alignment of gold anomalies can be noted from the Or-Bagoé survey gold data. Most of these trends strike northwest and define a wide corridor in the Permit area.

6.2.3 Ashanti-JCI, 1995 to 1996

The Ashanti-JCI joint venture undertook two phases of regional exploration at Kalana: surface exploration to assess near surface mineralisation in the vicinity of the underground mine complex and underground mapping and sampling as summarised in Table 6.2-2.

Table 6.2-2 Ashanti-JCI Kalana Exploration Summary

Activity	Area	Comments
Soil sampling	Historical Permit	E-W lines,1 km line spacing, 200 m sample spacing. 1,766 soil samples analysed for Au and 577 for As.
	Follow-up Soil sampling	Multiple grids 200 m x 80 m.
Oxide and supergene enrichment assessment for open-pit working	UG Mine Complex	308 pits on a 50 m grid by 50 m grid, 329 samples.
		RC drill programme with 49 holes for 2,635 m on a 100 m x 100 m grid.
		Two twin DD holes for 609 m.
Drilling	600 m E of the Shaft & a N-S deformation corridor W of the Shaft	20 RC holes for 1,419 m. 2 DD holes for 645 m.
UG Mapping & Sampling		200 channels with 1,773 samples. 41 metallurgical samples. 2 UG DD holes.

The Ashanti-JCI DD holes were twinned with SOGEMORK holes and generally confirmed the correlation and continuity of mineralised zones and the repeatability of the quartz veins.

Ashanti-JCI also conducted exploration at Kalanako, as summarised in Table 6.2-3.

Table 6.2-3 Ashanti-JCI Kalanako Exploration Summary

Activity	Summary
Orientation & Regolith Study	14 pits excavated to 4 m to 8 m, sampled and assayed.
	1,278 soil samples.
Mineralogical Study	141 samples.
Follow-up pitting	41 excavations, 264 samples assayed.
RC drilling	26 holes for 2,781 m (23 vertical, 3 inclined).
DD drilling	1 inclined hole for 290.5 m.

Ashanti-JCI completed an airborne magnetic survey (2,000 line kilometres, orientated 112°, 200 m line spacing, nominal ground clearance of 100 m with tie lines flown at five kilometre spacing) flown by Aerodat Incorporated between December 1995 and January 1996. Magnetic, radiometric and very low frequency electromagnetic data were acquired and recorded.

6.2.4 SOMIKA (Avnel), 2003 - 2016

SOMIKA (Avnel) undertook rotary air blast (RAB), reverse circulation (RC) and diamond drill core (DD) drilling programmes in 2005 and 2006 during their Phase 3 and 4 exploration campaigns, see Table 6.2-4. Underground exploration DD and RC drill programs were completed in 2008 in the Kalana Mine. Note, all the samples were assayed by 50 g fire assay and records of the tailings dam auger holes cannot currently be located.

Table 6.2-4 SOMIKA (Avnel) Drilling Summary 2003 - 2009

Deposit	Programme	No. Holes	Type	Total Metres
Tailings Dam	2004	?	Auger	?
Djirila	2005	150	RAB	6,235
	2005	22	RC	2,872
	2005/06	25	RC	2,542
	2005/06	15	DD	2,223
Tonda - Dabaran	2005/06	30	RC	2,525
Kalana Underground	2008	19	DD	2,029
Kalana	2008	44	RC	1,400

6.2.5 IAMGOLD, 2010 - 2013

IAMGOLD completed 164,655 m of DD and RC drilling within the Project Area (Table 6.2-5), including 130,363 m in the immediate vicinity of the Kalana Mine.

Table 6.2-5 IAMGOLD Drilling Summary 2010 - 2013

Deposit	Type	2010		2011		2012		2013		Total No.	Total m
		No.	m	No.	m	No.	m	No.	m		
Kalana	DD	50	13,177	109	22,955	48	14,843	-	-	207	50,975
	RC	111	11,702	203	24,209	251	43,223	2	254	567	79,388
Kalanako	DD	-	-	29	7,408	-	-	-	-	29	7,408
	RC	139	14,462	27	3,441	40	4,244	-	-	206	22,147
Dadjan	RC	22	2,202	-	-	-	-	-	-	22	2,202
Djirila	RC	-	-	-	-	19	2,535	-	-	19	2,535
Total		**322**	**41,543**	**368**	**58,013**	**358**	**64,845**	**2**	**254**	**1,050**	**164,655**

Kalana

The 2010 Kalana DD campaign was characterised by the use of HQ-NQ diameter drillholes in two directions (N254° and N344°) with a planned depth of 230 m (maximum 487 m). One hole (KA-SEK-DD001) was drilled at N30° targeting the vertical (3rd Order) veins. The RC programme was characterised by relatively short holes (planned depth 108 m) in one direction (N254°) to test the northwest – southeast vertical structures (Dynamite Trend) and the main part of the Kalana deposit.

The second campaign in 2010-11 was characterised by larger diameter core (PQ-HQ) with a planned depth of 280 m to 330 m (maximum depth 718 m). A larger diameter was used to improve core recovery and to increase sample weight. The holes predominately targeted the main Kalana deposit.

In 2012, deeper RC drill holes were completed in the Kalana deposit to infill the DD lines where structural information was considered sufficient. The use of RC drill holes was increased due to sample size (30 kg to 40 kg rather than three or four kilogrammes in DD core), improved sample recovery and efficiency in regard to time and cost. RC drilling improved the average grade of a given mineralisation interval according to gold behaviour and distribution as observed in underground working (with cross-reference to underground grade control face sampling) and increased the accuracy of local grade estimations.

IAMGOLD evaluated the RC drilling with a focus on the issue of grade smearing and concluded that there was no noticeable smearing of grade downhole.

Kalana North had some 80 DD holes, approximately 40% by length of the drilling programme. The majority of the holes were drilled at a dip of 60° to the north on 50 m sections (50 m between holes), nine holes were drilled at 60° to the east. The infill RC holes drilled in 2012 to a depth of 150 m to 270 m resulting in a 25 m by 50 m grid spacing.

Kalana Southwest also had approximately 80 DD holes, approximately 40% to 45% by length. They consisted of angled holes, dipping to the west on 50 m sections and 19 inclined holes dipping to the north on 100 m sections. In addition, a programme of angled RC holes was completed, drilled to a depth between 100 m and 250 m with collars spaced 25 m apart to achieve a 25 m by 50 m grid spacing.

Kalana Southeast was drilled by 16 DD holes, including eight holes drilled at a dip of 60° to the north on 100 m sections and eight holes drilled to the west on 100 m sections. The holes were drilled to depths between 100 m and 250 m with collars spaced at 25 m or 50 m apart. The thicker weathered zone, the lack of underground development and the low dipping orientation of the mineralisation contributed to the rational of drilling in two directions to better constrain the structural interpretations.

Kalanako

The first IAMGOLD campaign at Kalanako consisted of two east-west orientated RC drill lines.

Between 2010 and 2012, 206 RC drill holes were completed totalling 22,147 m. The holes were angled at 55° with an average depth of 107 m with collars spaced 25 m apart. In 2011, 29 DD holes totalling 7,409 m were drilled at an angle of 55° to the west at an average depth of 255 m.

Other Deposits

Dadjan was drilled by IAMGOLD in 2010 with 22 RC drill holes (2,202 m at a dip of 55°). The average hole depth was 100 m with collars spaced 100 m apart. The results were considered to be poor and the deposit has not been further explored.

IAMGOLD undertook additional drilling in April 2012 at Djirila to supplement that done by SOMIKA (Avnel) in 2005/06. IAMGOLD RC drilled 2,535 m which confirmed the high potential of the prospect.

6.2.6 SOMIKA (Avnel), 2013 - 2016

In 2014, SOMIKA (Avnel) undertook a small drill programme composed of 17 RC holes for 1,523 m to the north of the Kalana deposit. They also drilled 33 AC and RC drill holes in three condemnation programmes (one to the south of Kalana Town, one to the east of Kalana Town and one west of the Kalan deposit straddling the mine perimeter fence).

In 2015, SOMIKA (Avnel) completed 181 RC and diamond drill holes for 30,143 m on the Kalana deposit to provide additional information for the Feasibility Study ongoing at the time and facilitate grade definition in some areas of the resource. The campaign included 137 RC drill holes, 28 DD holes and 16 RC/DD holes.

The drilling, sampling and Quality Control/Quality Assurance (QA/QC) protocols used were the same as IAMGOLD implemented.

6.2.7 LeachWELL Re-assaying Programme

LeachWELL is a patented cyanide leaching reagent. In essence, the LeachWELL assay method applied at Kalana subjects a two kilogram sample (RC or DD) to a 24-hour bottle roll with the cyanide reagent at a controlled alkalinity (i.e. pH). The tails (50 g sample residue) are fire-assayed to detect any residual gold. The LeachWELL grade (LW) is added to the tails grade (TFA) to result in the final LeachWELL grade (LWTFA).

In 2013 after the departure of IAMGOLD, SOMIKA with Dr. Simon Dominy (a world leading expert on coarse gold and high nugget effect who was working with Snowden at that time) undertook a review of sampling and assaying methods at Kalana. A re-sampling programme involving whole sampling processing, LeachWELL or screen fire assaying (SFA) were considered; the whole sample method was dismissed due to practical constraints. The SFA methodology was tested but was not a success due to major QA/QC concerns. The initial LeachWELL methodology testwork on approximately 1,000 samples was successful and Avnel concluded that the method was suitable and resulted in a consistent upgrade to the existing assays.

Subsequently, the LeachWELL re-sample and re-assay programme was undertaken in three tranches:

- The first programme focused on samples within the central, upper 100 m portion of the Kalana Deposit. The LeachWELL samples showed a significant average grade difference (higher) and greater consistency of grade.
- The second programme principally targeted samples to a depth of 250 m. These samples again showed a significant average increase in grade and better consistency.
- As a result, there were minor changes to the geological interpretation, the new data added to the dataset and a new Kalana mineral resource estimate compiled (Denny Jones, 2014).
- The third re-sampling and re-assaying tranche focused on the remaining Kalana samples to initially a depth of 350 m and then extended to all remaining samples after the results of the 2014 MRE became available and it transpired that the resource was limited at depth by the re-sampling.

All of the results were integrated into the March 2015 Kalana Mineral Resource Estimate (Denny Jones, 2015).

The Kalanako and Djirila RC and DD samples taken by IAMGOLD were re-assayed by SOMIKA (Avnel) using the LeachWELL protocol in 2015. The increase in grade was consistent with the previously obtained results at the Kalana Deposit. The Kalanako re-assay results were incorporated into the March 2015 Kalanako Mineral Resource Estimate (Denny Jones, 2015).

The LeachWELL assays have precedence over the fire assays in the Resource Estimate for the re-assayed samples.

6.3 Tailings Storage Facility

Two tailing storage facilities (TSFs) exist within the Kalana Project, the TSF proper ("TSF") is located halfway between Kalana and Kalanako, and the small TSF ("Shaft No. 2 TSF") within the fence close to the plant, see Figure 6.3.1.

The TSF is located 600 m northeast outside the Kalana UG Mine fence, approximately halfway to Kalanako. It was originally constructed in the 1980s by SOGEMORK as a single impoundment in the shape of an elongated horseshoe. SOGEMORK subsequently sub-divided it to retain approximately 200,000 t of underground tailings and 50,000 t of oxide tailings in separate zones.

The TSF was re-opened in 2004 with a three-metre lift, and extended in 2006, 2012 and 2016 by SOMIKA (Avnel) and closed (but not covered or rehabilitated) in December 2017 by SOMIKA (Endeavour). It is of an upstream construction, side-hill impoundment design and has a similar footprint to that created by SOGEMORK with a number of lifts.

The Shaft No. 2 TSF is located within the Kalana UG Mine fenced perimeter, to the west of the Main Shaft. It was the original open-pit excavated by SOGEMORK in the 1980's. The exact size, geometry and gold recovered from the pit is unknown (the pit was reportedly approximately 100 m long by 50 m wide and 5 m deep, with about 10 kt of mineralised material excavated). It was subsequently infilled with plant tailings from underground workings generated by SOMIKA (Avnel) between 2004 and 2009.

The TSF was sampled in May 2004 on a 10 m by 10 m grid which collected one metre vertical downhole samples from auger samples (this information cannot be located by Endeavour). The tonnage was estimated from historical production records and combined with records for the tailings resulting from ore milling for the period from January to June 2004.

In 2012, IAMGOLD sampled the Shaft No. 2 TSF and the TSF proper using a hand-held auger to an average depth of five metres (which therefore did not always reach the dam base). The TSF was drilled on a 20 m by 20 m grid and the Shaft No. 2 TSF on an approximate 10 m by 10 m grid (due to access constraints). IAMGOLD implemented a QA/QC programme. The assays were processed by SGS Kalana and were generally 2.5 kg to 3 kg in weight. Table 6.3-1 presents a summary of the IAMGOLD TSF auger drilling results. (Note - the auger data suggests the presence of a coherent area at the southern end of the Shaft No. 2 TSF that exhibit assay grades exceeding 3 Au g/t.)

The Shaft No. 2 TSF was also sampled by some of the 2015 SOMIKA (Avnel) and 2017-18 SOMIKA (Endeavour) Kalana Deposit resource delineation drill holes (64 intervals from 29 holes with a length weighted average grade of 1.7 Au g/t).

Endeavour reports that approximately 54 kt of tailings were treated from the southern end of the Shaft No. 2 TSF in 2016 and 2017, and realised a gravity-recovered grade of 7.6 Au g/t. Endeavour notes that some of this material was treated in conjunction with underground ore and suggests that some coarse gold may have been retained in the plant circuit which should not be attributed to the TSF material. Endeavour concludes that the claimed recovered grade for the TSF material should be reduced to 6.0 Au g/t.

Note, some information pertaining to the TSFs including the original topographic surveys, actual collar elevation data for the 2012 auger data and results from a 2004 auger drilling programme are currently unavailable.

Figure 6.3.1 Kalana TSF Locations

(Source: Google Earth Pro. Image date: January 2016.)



Table 6.3-1 IAMGOLD TSF Auger Sampling Summary

	Shaft No. 2 TSF	TSF
Hole Name Prefix	KA-SEK-WBS	KA-SEK-WBN
No. Auger Holes	165	137
Average Depth (m)	3.6	3.5
No. Assays	587	485
Average Grade (g/t Au)	2.3	2.1
Min. Grade (g/t Au)	0.5	0.2
Max. Grade (g/t Au)	13.0	16.1

Figure 6.3.2 TSF Plan showing 2012 Auger Collars and Dam Layout

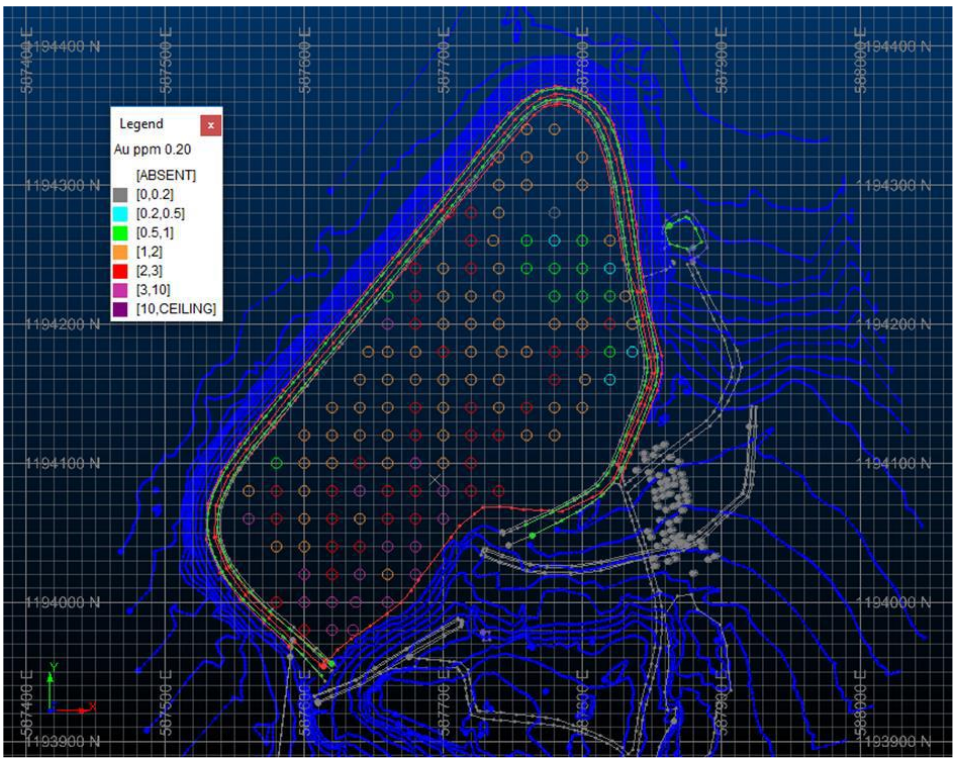
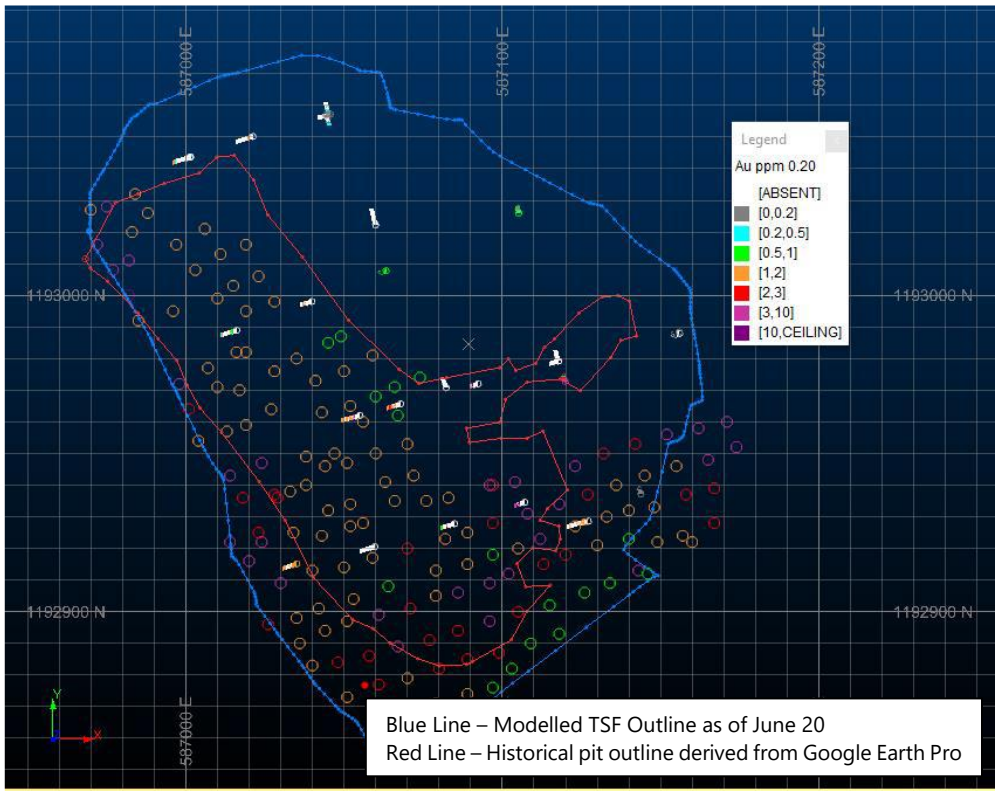


Figure 6.3.3 Shaft No. 2 TSF Plan showing 2012 Auger (circles) and Resource DH Collars



6.4 Historical Resource Estimates (Non-NI 43-101 Compliant)

The historical resource and reserve estimates in this Section were generated prior to the development and implementation of NI 43-101 and therefore should not be relied upon; they are solely included for the purposes of transparency. Further details can be found in the 2016 Technical Report. Table 6.4-1 summarises the Kalana resource and reserve estimates between 1985 and 1997.

Table 6.4-1 Summary of Historical Estimates of Mineral Resources and Mineral Reserves for Kalana Mine (1985 to 1997)

Company	Type	Mineral Resource/Reserve	Tonnes ('000 t)	Gold Grade (Au g/t)
SOGEMORK, 1985	UG	Mineral Resource above 150 m depth (start of mining)	1,737	15.10
SOGEMORK, 1991	UG	Mineral Resource above 150 m depth (end of mining)	1,644	14.70
SOGEMORK, 1991	UG	Mineral Resource from 150 m to 350 m depth	1,513	13.60
Clark, 1996	UG	Mineral Reserve	2,200	9.90
Geoconsult, 1996	OP & UG	Total Mineral Resource	6,957	5.70
Snowden, 1997	OP & UG	Total Mineral Resource	6,486	6.14

Whilst the Snowden 2004 Mineral Resource and Mineral Reserve estimate as presented in Tables 6.4.2 and 6.4.3 is not historical as defined by NI 43-101, it is prior to the modern work and therefore included in this section. It was prepared in conformity with Canadian Institute of Mining (CIM) Exploration Best Practices Guidelines for the Estimation of Mineral Resource and Mineral Reserves.

Table 6.4-2 Historical Mineral Resource Estimate for Kalana Mine, Snowden June 2004

Category	Tonnes (kt)	Gold Grade (Au g/t)	Ounces ('000 oz)
Measured	356	7.50	86
Indicated	2,692	10.8	934
Measured + Indicated	**3,047**	**10.4**	**1,020**
Inferred	2,261	3.4	249

Note: Mineral Resources are not Mineral Reserves and do not have a demonstrated economic viability. All values have been rounded to reflect the relative accuracy of the estimates. Mineral Resources are reported at unknown cut-off grade, gold price, and metallurgical recovery. This historical estimate is superseded by the current Mineral Resource in Section 14.

Table 6.4-3 Historical Mineral Reserve Estimate for Kalana Mine, Snowden June 2004

Category	Tonnes (kt)	Gold Grade (Au g/t)	Ounces ('000 oz)
Proven (Open pit)	10	5.0	2
Proven (Underground)	70	15.4	35
Probable	881	14.1	400

Note: All values have been rounded to reflect the relative accuracy of the estimates. Reserves are reported at unknown cut-off grade, gold price, and metallurgical recovery. This Historical Estimate is superseded by the current Mineral Reserve estimate in Section 15.

SONAREM-SOGEMORK reported a Historical Estimate at Kalanako of 600 kt at a grade of 3.6 Au g/t, unknown date.

A NI 43-101 (CIM) compliant Measured Resource of 234 kt at 1.9 Au g/t for 14 koz of gold was reported by Snowden in 2005 for the TSFs. The tonnage of the tailings dam was estimated from the historical production records with the addition of the tailings from ore milled in the period January to June 2004. No survey measure of the tailings dam was possible as there was no survey record of the original topographic base for the dam. The average sample gold grade for the tailings dam was used for the June 2004 resource estimate of the SOGEMORK tailings.

6.5 Previous Resource Estimates (NI 43-101 Compliant)

The Previous Mineral Resources reported herein were classified within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (November 2010) and were NI 43-101 compliant, as summarised in Table 6.5-1.

Denny Jones Pty. Ltd. (Denny Jones) prepared Mineral Resource Estimates (MREs) on the "Kalana Main Project" at the request of Avnel Gold Mining Ltd., the majority shareholder of SOMIKA in March 2014, September 2014, March 2015, September 2015 and March 2016. The "Kalana Main Project" encompassed the Kalana deposit, the two TSFs ("tailings"), the Kalana Exploitation Permit and the Kalana Underground Mine (with all of its associated infrastructure). The 2014 and 2015 MREs used a gold price of $1,100/oz and a cut-off grade of 0.9 Au g/t for the primary deposits. Following work on a Feasibility Study, the MRE was updated in March 2016 with a gold price of $1,400/oz.

The tailings MREs comprised tailings from the SOGEMORK processing (pre-1991) and tailings from processing of mineralisation by SOMIKA (post -2004). The overall tailings estimates were a simple weighted average of the SOGEMORK tailings estimate and the latest production data from June 2004. No cut-off grade was applied.

The Maiden Kalanako Inferred MRE by Denny Jones was published in September 2015 and updated in May 2017 after additional drilling in late 2016 to include Indicated Mineral Resources. A gold price of $1,100/oz in 2015 and $1,400/oz in 2016, and a cut-off grade of 0.9 Au g/t were used to report the Resources.

Table 6.5-1 Previous NI 43-101 Mineral Resource Estimates

Date	Deposit	Classification	Tonnes (Mt)	Grade (Au g/t)	Gold (Moz)
March 2014	Kalana	Indicated	8.54	4.53	1.25
		Inferred	2.09	3.76	0.25
September 2014	Kalana	Indicated	12.9	4.57	1.90
		Inferred	0.7	4.24	0.10
		Exploration Target	8.3 – 8.8	4.2 – 4.9	1.1 – 1.4
	Tailings	Indicated	0.66	1.80	0.04
March 2015	Kalana	Indicated	14.5	4.52	2.11
		Inferred	1.8	5.28	0.31
		Exploration Target	5.3 – 6.6	3.9 – 4.7	0.7 – 1.0
	Kalanako	Inferred	0.38	5.55	0.069
September 2015	Kalana	Measured	9.0	4.24	1.23
		Indicated	10.9	4.19	1.46
		Measured & Indicated	**19.9**	**4.21**	**2.69**
		Inferred	0.8	4.59	0.11
		Exploration Target	3.3 – 4.7	3.9 – 3.7	0.4 – 0.6
	Tailings	Indicated	0.73	1.75	0.04
March 2016	Kalana	Measured	9.5	4.20	1.28
		Indicated	13.5	4.10	1.77
		Measured & Indicated	**23.0**	**4.14**	**3.06**
		Inferred	1.7	4.51	0.24
May 2017	Kalanako	Indicated	0.77	4.61	0.11

Notes:

1. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues. The Mineral Resources in this Technical Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*

2. *The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.*

3. *Contained metal and tonnes figures in totals may differ due to rounding.*

6.6 Gold Production History

Kalana was worked from underground (and surface) by SOGEMORK between 1985 and 1991. Underground mining operations were resumed by SOMIKA in January 2004 until September 2017 when mining operations were stopped, and processing finished in December 2017.

6.6.1 SOGEMORK Surface Mining

Gold mineralisation was extracted initially in the central area of the Kalana deposit via a 20 m deep shaft with 80 m of lateral development along Vein 1 (Snowden, 2005). The mining method was later changed to an open pit approximately 100 m long by 50 m wide and 5 m deep. Some 10,000 tonnes of mineralised material was extracted from the open pit (Snowden, 2005).

6.6.2 SOGEMORK Underground Mining

Shafts No.1 and No. 2 were sunk to depths of 108 m and 103 m respectively to access the flat dipping quartz veins and to develop the 290 m level. Veins 1, 3, 14, 15 and a stockwork zone were exposed from the two kilometres of development on 290 m level. Vein 1 was mined along strike from No. 2 Shaft at the 315 m level (SRK, 2013). Vein 1 and Splays 1c and 1d were mined between the 315 m and 290 m levels and produced the majority of underground tonnes.

During their six-year tenure, SOGEMORK produced approximately 81,800 oz Au from 227,000 t mined grading an average of 13 g/t Au at a recovery of 86%.

6.6.3 SOMIKA Underground Mining

Mining operations were resumed by SOMIKA in January 2004 and continued until September 2017. The mining reserves were extended by deepening the No. 2 Shaft to 180 m below surface. Mining was conducted by room-and-pillar; ore was extracted from narrow stopes by drilling and blasting, with scraper winches removing the ore from the stopes. The mine production rate was approximately 50,000 tonnes per annum.

SOMIKA produced 185,116 oz Au from 628,615 t mined, grading an average of 11.6 g/t Au with a recovery of 83%. Table 6.6-1 presents SOMIKA's production by year.

Table 6.6-1 SOMIKA Production

Year	Tonnes Milled	Gold Grade (Au g/t)	Gold Recovery (%)	Gold Ounces
2004	35,667	9.4	68.6	7,396
2005	34,885	15.5	86.1	14,923
2006	27,743	28.2	90.1	22,638
2007	35,222	24.2	92.0	25,359
2008	45,348	15.7	82.0	21,407
2009	50,238	12.1	86.6	16,677
2010	50,238	7.7	86.1	10,727
2011	47,546	7.2	84.6	9,550
2012	36,608	7.3	87.3	10,090
2013	50,848	7.6	82.0	10,176
2014	49,513	7.4	91.2	9,549
2015	52,235	7.3	79.1	9,679
2016	54,043	7.2	76.9	9,633
2017	58,481	5.6	70.2	7,312
Total	**628,615**	**11.6**	**83.1**	**185,116**

Mine Geology

A dedicated Underground Exploration team was established in 2009 (until 2012) which focused on mapping, sampling and dedicated underground development to verify the concept of vein packages, examine the structural framework to verify the concept of vein packages, verify drillhole grade variability within vein packages exposed in existing stopes and galleries, and understand the distribution of gold in the vein packages in order to constrain the drilling pattern and the variability of grade at a sample scale. The results of this work formed the basis of the subsequent Mineral Resources and further studies.

It is reported that the underground sampling generally confirmed the mine grade control sampling results, with a less scattered statistical model (Snowden, 2016). Sampling and mapping of the underground development confirmed the consistency of drillhole grades and structural interpretations with that observed in the workings.

Mine Call Factor (MCF)

The following is modified after Avnel, 2013.

The Mine Call Factor (MCF) compares the gold received at the plant with the estimated gold grade of mineralisation delivered to the plant and is based on:

- Plant gold received was the total of gold recovered from the plant as bullion plus gold sent to tailings.
- The gold in tailings was based on the sampled tails grade multiplied by tonnes milled.
- The estimated gold grade delivered to the plant was based on the mine survey results multiplied by the face sample grades.
- The individual face sample grades were capped to 131 Au g/t.

The MCF always exceeded 100% (101% in 2012 and 121% in 2011) indicating the gold received at the plant exceeded the estimated gold delivered to the plant (as estimated from the underground sampling). Note, the gold estimated as received may be incorrect if the sampled tails grade was inaccurate.

This approach assumed that the grades from the tailings samples were accurate; IAMGOLD in 2012 sampled and assayed the tailings material by auger and pits. The assays indicated that the plant sampled grade under-called the tailings grade, which would result in a higher MCF. Hence, the evaluation of grade to the mill (whether it be a sampling issue, an assay issue, an evaluation or a combination of all three) under-estimated the grade delivered to the plant.

Endeavour notes that the assays were fire assays and therefore probably underestimated the grades due to the coarse nature of the gold.

Resource Reconciliation

Historically, the forecasting of mining grades based on historical exploration drill holes was difficult due to the high percentage of free milling visible gold. Experience showed drill hole fire assayed grades underestimated the actual mining grades between 60% and 300% (Avnel, 2013).

Snowden reconciled the 2004 MRE to the SOGEMORK 1982 - 1991 production and concluded that approximately 30% more gold was produced than was predicted (Snowden, 2004).

In 2013, SOMIKA reconciled the actual mined gold with the Snowden 2004 MRE (Avnel, 2013). The reconciliation compared the predicted gold production (from Reserve Estimates) against actual gold production; gold production exceeded reserve estimates due to higher grades and higher tonnes mined (due to the extension of mining blocks outside the reserve block model). Examples are:

- Vein 1 - 137% additional ounces due to 100% increase in grade.
- Vein 17 - 66% additional ounces due to 68% increase in grade.

7.0 GEOLOGICAL SETTING AND MINERALISATION

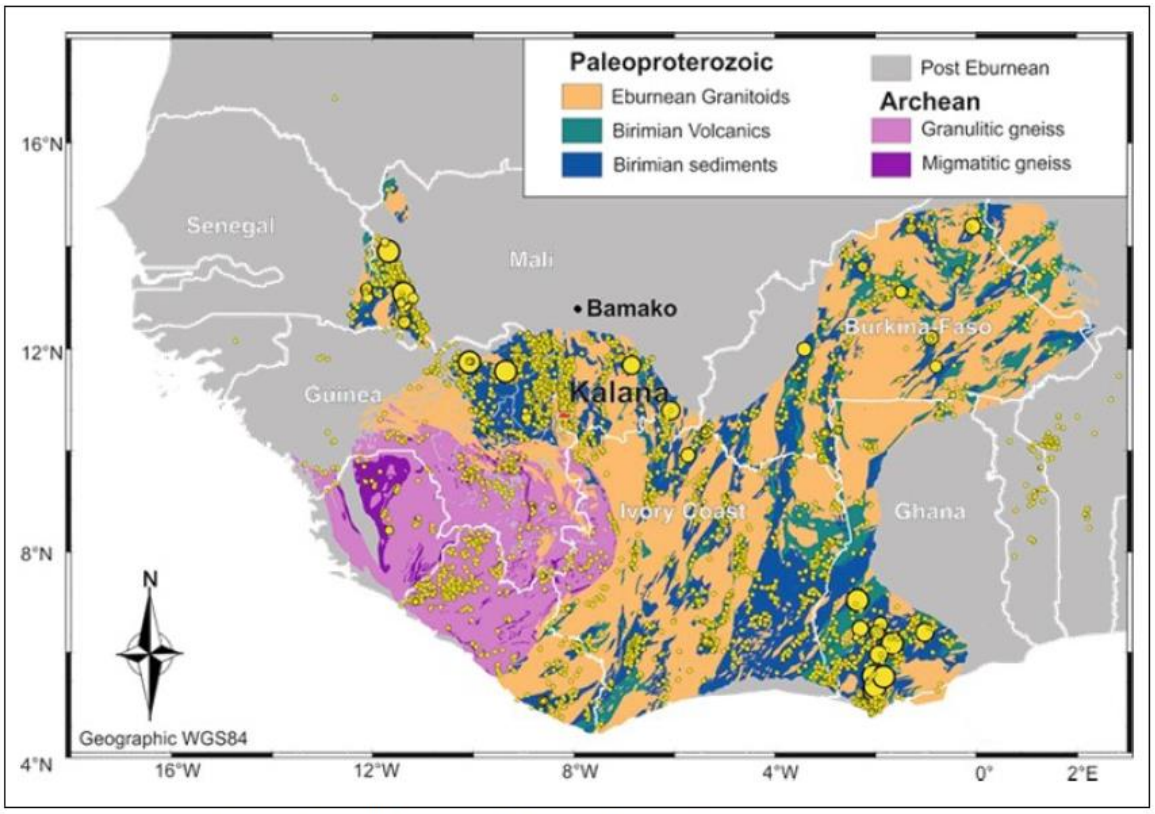
7.1 Regional Geology

Mali is underlain by two cratonic nuclei, extensions of the West African Craton and the Tuareg Shield, which were welded together during the Neoproterozoic Pan-African orogeny.

The Kalana Project is located in southern Mali, near the Guinean border and consists of Paleoproterozoic rocks that are part of the Dialo-Mossi Domain (the Leo-Man Rise) and the West African Craton. The Paleoproterozoic part of the Craton (Birimian volcano-sediments) were structurally deformed when Archean blocks merged between 2,200 Ma to 2,000 Ma in the Eburnean Orogeny.

The Kalana region is underlain by the Dialo-Mossi (or Baoule-Mossi) Domain composed of three principal Birimian-Eburnean lithological and structural units, covered easternmost and northernmost by Neoproterozoic sediments of the Taoudeni Basin, Figure 7.1.1. The first unit is composed of a Birimian volcano-sedimentary series (dacitic-andesitic compositions) of the Yanfolila-Kalana and Bagoé Basins. These were intruded by a younger granitic to monzogranitic suite of rocks (approximately 2,090 Ma) and thereafter by a late dioritic-grandioritic intrusive event (approximately 2,075 Ma) represented in the Yanfolila-Kalana area by local plugs and dykes at the Kalana and Kodieran gold deposits.

Figure 7.1.1 Leo-Man Rise Geological Map with Gold Occurrences



Source: Avnel, 2014, with USGS database gold occurrence overlay

7.2 Local Geology

The Kalana Project area lies along the western margin of the Bagoé Basin toward the southern extremity of the Yanfolila-Kalana volcano-sedimentary belt which forms the border with the Siguirri Basin in Guinea. Volcanic rocks, primarily basalt, are found in small domains in the eastern and westerns parts of the Basin, pinched along orogenic shear zones. A large Eburnean felsic intrusion, the Bougouni Batholith, occupies the central part of the Bagoé Basin. Small intermediate intrusions, commonly diorite and granodiorite, are mainly found in the eastern part of this area, as well as in the Kalana area. All rock types in the Basin are cut by (Neoproterozoic or potentially Jurrassic) dolerite dykes that are dominated by easterly, northeasterly, and north-northwesterly trends.

Gold mineralisation in the southern portion of the Bagoé Basin appears to be controlled by basin-margin parallel faults that generally trend northwest to north. The Bagoé Basin is host to the multi-million-ounce gold deposits at Morila and Syama.

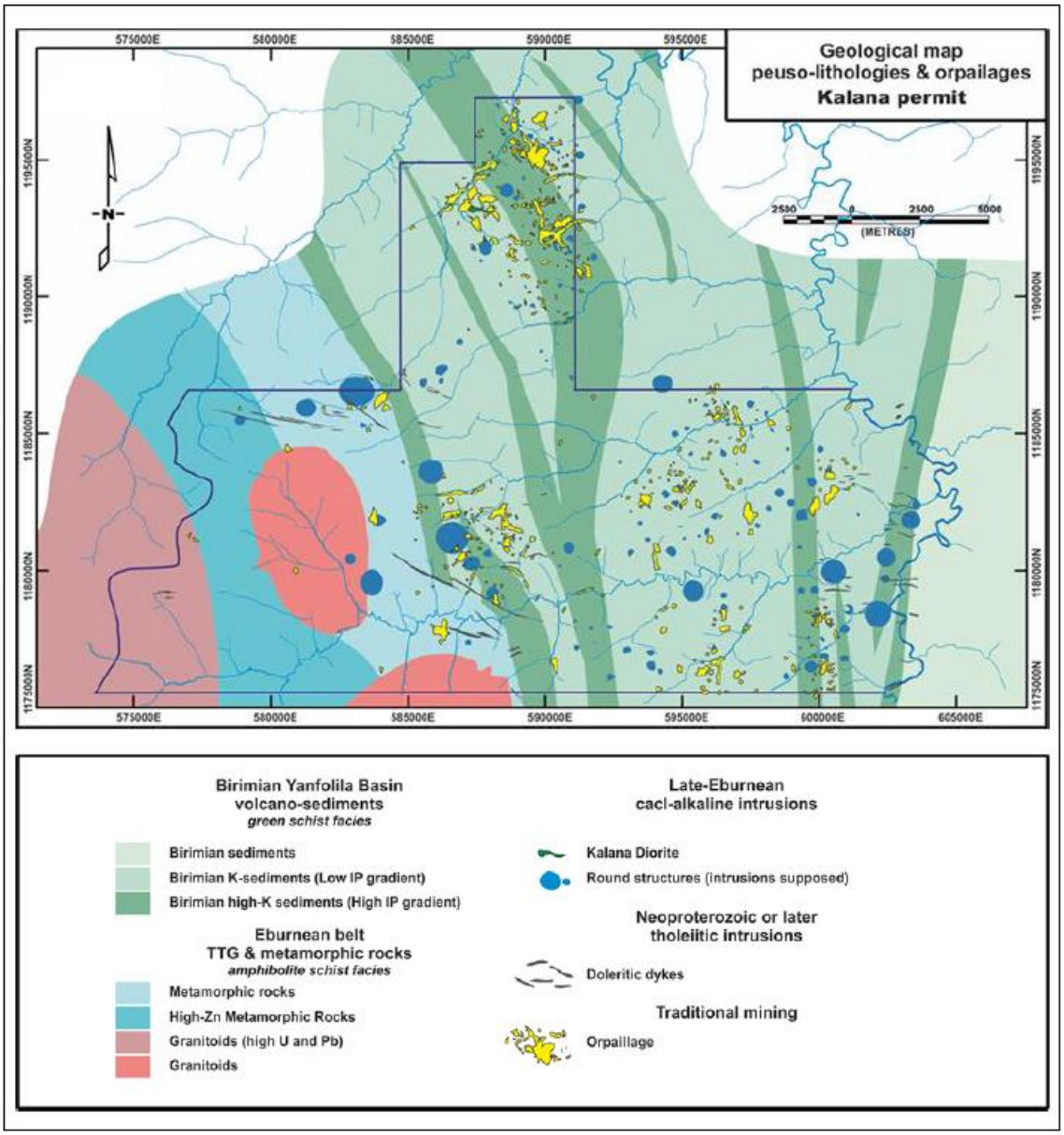
The Kalana Project Area is covered by an extensive lateritic profile, characteristic of the Guinea Savannah climatic zone, obscuring the bedrock. The depth of weathering ranges from 30 m to 130 m.

The Kalana Project gold deposits are hosted by volcano-sedimentary rocks of the lower part of the Upper Birimian Group, Figure 7.2.1. The (meta)-sediments consist of turbidite sandstone-siltstone (+/- tuff) sequences. The sandstone is fairly massive and medium-grained and the siltstone is finely laminated and locally graphitic. They are situated on the steeply dipping western limb of a regional syncline with folds plunging steeply to the north.

The Kalana Project deposits identified to date include:

- Kalana.
- Kalanako.
- The TSF
- Shaft No.2 TSF.
- Dijirla.

Figure 7.2.1 Kalana Project Geological Map



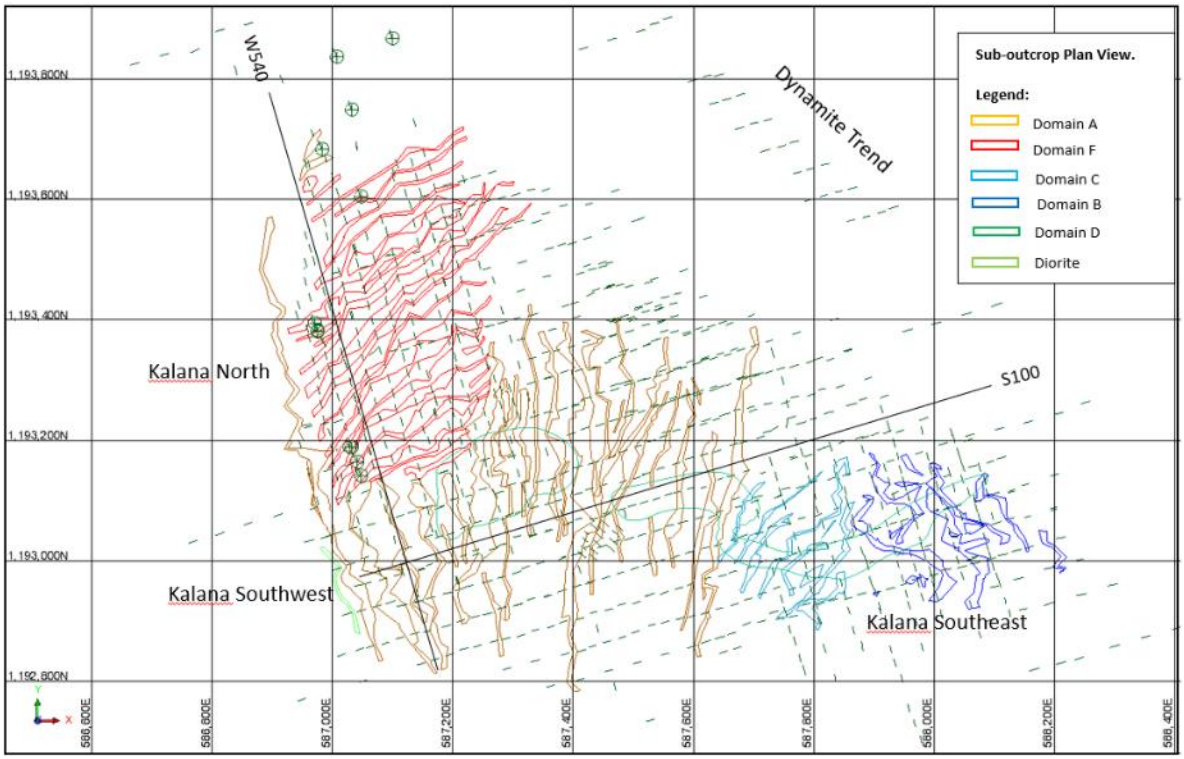
Source: Avnel, 2014; and Avnel, 2016a

7.3 Kalana Deposit Geology

The Kalana gold deposit is composed of mesothermal gold mineralisation hosted mainly in quartz veins following shallow dipping and vertical dykes adjacent to small intrusions emplaced in a sub-volcanic environment. The mafic intrusions played a key role as a source of heat flow driving the hydrothermal cell at the origin of local deuteric alteration and sulphide-carbonate metallotects (i.e. arsenopyrite, pyrrhotite, pyrite, chalocopyrite, ankerite and scheelite). The Kalana deposit can be subdivided into four geographical areas, as illustrated in Figure 7.3.1.

- Dynamite Trend – Located northeast of the main Kalana deposit. Does not host significant gold mineralisation.
- Kalana North – Hosts 1st, 2nd and 3rd Order Veins. Significant amount of historical underground workings on 1st and 2nd Order veins.
- Kalana Southwest – Hosts 1st Order Veins only and the majority of the historical underground workings.
- Kalana Southeast - Hosts 1st Order Veins only.

Figure 7.3.1 Kalana Deposit Geological Map



The Kalana deposit hosts an east-west sigmoidal elongated diorite dyke with two major bodies that have been traced to a depth of 600 m intruding the Birimian rocks. The intrusion is approximately 180 m by 250 m in the main body and 150 m by 750 m to the east (Kalana Southeast). The two parts of the intrusion are surrounded by a contact metamorphic aureole (hornfels) up to 30 m wide.

Low-extension reverse faults are responsible for the tensional, en-echelon, stacked, flat, thin dioritic dykes and sills (10 cm to a few metres thick) that occur around the main intrusion. A magmatic breccia has been traced in areas of diorite emplacement related deformation. It is highly silicified with a high arsenopyrite content but no gold.

Quartz vein packages host the gold mineralisation at Kalana which follow the main planes of thrusting (dipping 25° to 35°), and minor reverse faults and shears being probably responsible for the "pinch and swell" nature of the mineralised quartz veins.

The main sigmoidal diorite and the major part of the shallow, south to east dipping quartz veins emplaced in the same stress field (sigma 1 in N120 - N130) define a major phase of gold mineralisation, referred to as "1st Order Veins".

In the northern part of the deposit, a dipping quartz vein swarm (striking N40 - N60 dipping 45° to the south) occurs as thin packages of mineralisation following a second event of mineralisation partially overprinting the main one, referred to as "2nd Order Veins". These veins are regarded as "relay structures" sandwiched between upper and lower 1st order veins (i.e. mineralised Riedel structures).

A third event of mineralisation is characterised by a local stress field with sigma 1 in N30 - N40 which characterise a shadow pressure domain in the northwestern corner of the diorite occurring after its cooling (Kusters, 2009) resulting in sub-vertical, discontinuous thin veins referred to as "3rd Order Veins".

The Kalana deposit is overlain by saprolite varying from approximately 30 m above the diorite to 80 m at its margins.

7.4 Kalana Deposit Mineralisation

The Kalana gold mineralisation occurs within stacked quartz veins and stockworks hosted by the Birimian age metasediments and the intrusive diorite stock. The northwest-southeast trending mineralised vein system extends over an area of approximately 1,200 m east by 1,000 m north. Mineralisation is restricted to a block bound by northwest and northeast trending faults. Mineralisation has been identified to a depth of 600 m, the limit of current drilling.

Kalana hosts four types of gold mineralisation:

- 1st Order Veins - Flat dipping (an average dip of less than 25°) quartz veins and vein sets in the metasediments and diorite stock.
- 2nd Order Veins – Dipping (typically 45° south) quartz veins and vein zones striking N60 to N40 in Kalana North and Kalana Northeast.
- 3rd Order Veins – Sub-vertical thin quartz veins in Kalana North only. Very minor.
- Stockwork mineralisation at the contact of the diorite.

The quartz veins pinch and swell significantly, anastomosing (i.e. splitting and re-joining) which hampers the identification of continuity of the individual quartz veins. Significant work was undertaken by SOMIKA to identify "vein packages" with good continuity that comprise of a number of separate, individual veins which split, coalesce, disappear, re-appear, thin, thicken, etc. The approximately 60 vein packages identified were further developed by Endeavour to interpretate continuity of the individual veins within distinct vein packages.

The 3rd Order Veins and the stockwork mineralisation have not been wireframed / modelled as separate, distinct mineralisation styles but as "blow-outs" to the 1st and 2nd Order Veins. Endeavour recommends that the 3rd Order Veins are modelled separately in any future work.

Table 7.4-1 below is included for comparative purposes with historical documents and illustrates the evolution in the vein naming system.

Table 7.4-1 Current and Historical Vein Names

SOMIKA Current Name	SOGEMORK Historical Name	SOMIKA Current Name	SOGEMORK Historical Name
1	1, 16, 17	59	18b
2	2	60	18
3	3	61	17
4	4	62	18c
9	8	63	14
10	10	64	17
11	9	65	15
15	Savanne	102	19c, 20b, 20c
18	Superette	399	21
57	18d	400	20

1st Order Flat Dipping Veins

The flat dipping quartz veins (see Figure 7.4.1) account for approximately 75% of the mineralisation identified at Kalana. Generally, these veins are controlled within a system of sub-parallel, en-echelon fractures that dip predominantly to the east or southeast. These veins pinch and swell significantly - the average vein thicknesses ranges from 0.1 m to more than three metres. Despite the good continuity of the mineralised packages across the deposit, many of the veins contained in the packages subdivide and the connectivity of individual veins between drillholes is not always obvious.

The 1st Order Veins are often laminated and contain appreciable amounts of arsenopyrite. They were historically referred to as "Master Veins". Typically, the metre-thick quartz veins are accompanied by weakly mineralised veinlets in the footwall and hangingwall, historically referred to as "disseminated mineralisation".

Vein 1 is the largest example of the flat dipping, 1st Order Veins and is typical of this style of mineralisation. The other significant 1st Order Veins are Veins 2, 3, 4, 9 (mined as number 8), 10, 15 (previously known as Savanne) and 18 (previously known as Superette).

Figure 7.4.1 1st Order Vein 1, 160 Level



2nd Order Dipping Veins

The 2nd Order Veins are found north of the diorite stock, dipping 45° to the south striking N60 in the north and N40 in the south of Kalana North. They are also present in Kalana Northeast.

The 2nd Order Veins partially overprint the 1st Order Veins". These veins are regarded as "relay structures" sandwiched between upper and lower 1st Order Veins (i.e. mineralised Riedel structures). The junction of 1st and 2nd Order Veins are typically enriched in gold (normally modelled / attributed to the 1st Order Vein).

3rd Order Sub- Vertical Veins

The thin 3rd Order Vertical Veins have only been identified in diamond drill holes and underground workings in Kalana North (see Figure 7.4.2). Six sub-vertical cylindrical zones approximately 50 m in diameter and 150 m deep have been delineated to represent zones with 3rd Order Veins. They have not, however, been grade modelled as distinct zones or domains but incorporated into 1st or 2nd Order wireframes. The 3rd Order sub-vertical vein zones appear to be aligned northwest-southeast and northeast-southwest in an "X" shape.

Figure 7.4.2 3rd Order Sub-Vertical Veins



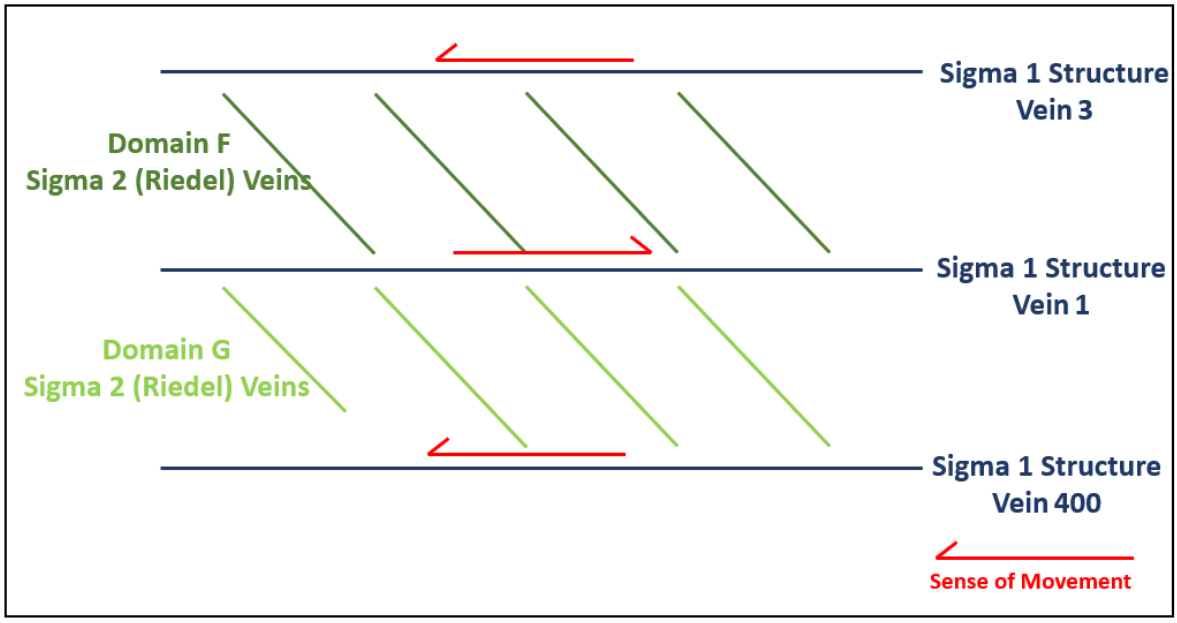
(Source: Avnel, 2014)

7.5 Kalana Vein Modelling Strategy

Endeavour has modelled the Kalana gold mineralised veins using the SOMIKA (Avnel) vein packages and the following assumptions:

- 1st Order Veins 1, 3, 10, 15, 102 and 400 are "marker horizons".
- 1st Order Veins 1, 3 and 400 are the upper, middle and lower respectively structures (Sigma 1) bounding the upper and lower 2nd Order relay or Riedel veins (Sigma 2).

Figure 7.5.1 Sketch of "Riedel" 2nd Order Veins



- 2nd Order Veins are found only north of the diorite.
- 1st Order Veins 100 to 106 are found only south of the diorite below Vein 1.
- 1st Order Veins 396 to 399 are found only north of the diorite below Vein 400.
- Historical underground workings have been used to aid vein continuity.
- Diorite stockwork mineralisation and 3rd Order Sub-vertical Vein mineralisation has been captured as "blow-outs" of 1st and 2nd Order structures, not as separate mineralised entities.

One hundred and thirty-five veins have been modelled within 61 Vein Packages, grouped into eight types or domains based on geometry, orientation and/or location, as summarised in Table 7.5-1.

Twelve 2nd Order vein packages are found above and below Vein 1 but are categorised into upper and lower segments, domained separately into Domains F and G, respectively. Figure 7.5.2. schematically illustrates the relationship between the domains and the diorite plug. Figure 7.5.3 presents the domains in a number of different orientations.

Figure 7.5.2 Sketch of the Geological Modelling and Vein Domain Concepts and Assumptions at the Kalana Deposit

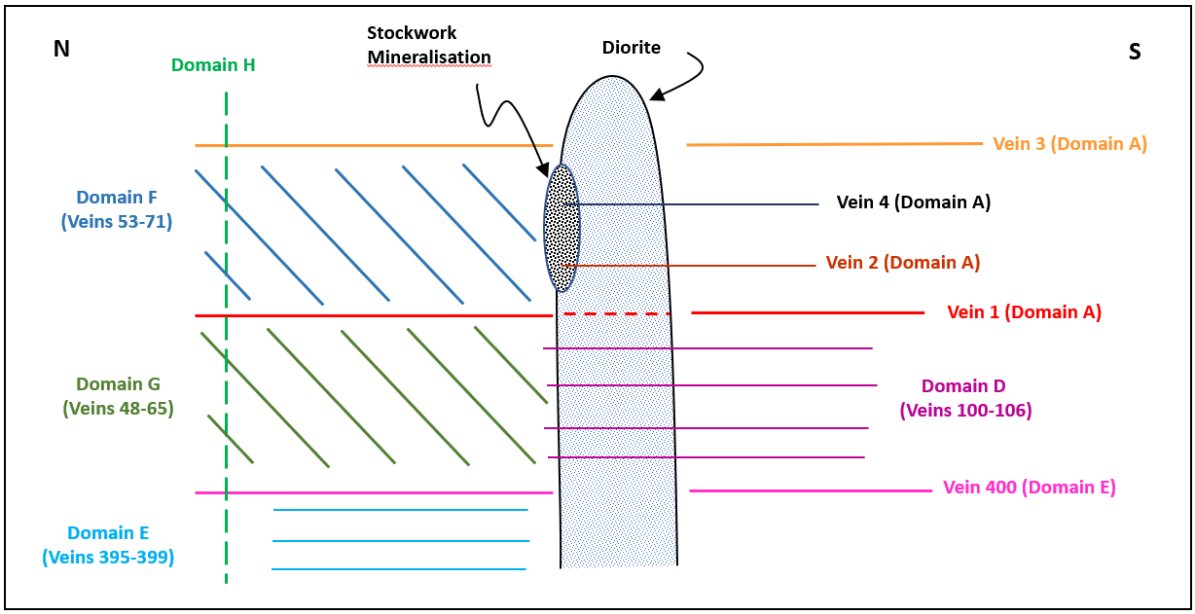


Table 7.5-1 Kalana Deposit Veins by Domain, Order and Package

Domain	A	B	C	D	E	F	G	H	Total
Vein Order	1st	1st	1st	1st	1st	2nd	2nd	3rd	
Number of Vein Packages	15	3	6	7	6	7 Sole	5 Sole	**NA**	**61**
						12 Shared			
Vein Numbers	1	18	22	100	395	53	48		
	2	19	23	101	396	54	49		
	3	20	24	102	397	55	50		
	4		25	103	398	56	51		
	5		26	104	399	57	52		
	6		27	105	400	58	53		
	7			106		59	54		
	9					60	55		
	10					61	56		
	11					62	57		
	12					63	58		
	13					64	59		
	14					65	60		
	15					66	61		
	16					67	62		
						68	63		
						69	64		
						70	65		
						71			
Number of Veins	44	8	12	12	12	23	24	Unknown	**135**

Domain A is the largest domain in regard to volume and is located in the southern half of the deposit with the exception of Vein 1 which is found across the whole deposit. Veins 1, 3, 9 and 15 are the principal veins in this domain. Vein 1 is the lowermost vein in Domain A, and Vein 15 uppermost. Domain A is a "stack" of parallel quartz veins centred on the diorite plug and sigmoidal diorite. It is composed of 15 Vein Packages and 44 veins encompassing 14.30 $Mm^{3,}$ (some of the wireframes have been extended above the topography, therefore the block model vein volumes will be slightly smaller). Vein Package 21 has been renumbered as Vein 15 Lower (15-2), there is no Vein Package 8.

Domain B is located in Kalana Southeast and lies near surface; gold enrichment in and movement of the laterite can be observed in this area. The sigmoidal diorite runs through the middle of this domain. Domain B is composed of three Vein Packages incorporating eight veins and 1.15 Mm^3. Domain. Vein 20 is uppermost and 18 lowermost. Domain B lies unconformably above Domain C and is bound to the north and east by faults.

Domain C lies in Kalana Southeast, lying unconformably above Domain A and below Domain B. It is composed of six packages, 12 veins containing 2.09 Mm^3. Vein 22 is the lowermost package in the domain and abuts Vein 15 (Domain A). Domain C is also transected by the sigmoidal diorite and fault bound to the north and east – Vein 22-1 is an anomaly as it lies to the north of the northern boundary fault.

Domain D lies in Kalana Southwest below Domain A. It is composed of seven packages, 12 veins containing 4.00 Mm^3. It is situated south of the diorite. Some of the veins are substantially thicker proximal to the diorite. Domain D and Domain E contain the same vein systems with different trends separated by the diorite plug; the domain division is a function of vein geometry (dip and plunge).

Domain E lies in Kalana North, below Domain G and north of the diorite. It is composed of seven packages, 12 veins containing 7.90 Mm^3. Some of the veins are substantially thicker proximal to the diorite.

Twelve packages are found in Domain F above Vein 1 and in Domain G below Vein 1. Domains F and G vein packages are composed of closely stacked, parallel, thinner veins than the 1st Order domains. The 2nd Order veins are generally thin (typically four metres in the west and two metres in the east) but can "blow-out" to substantial thickness (over 35 m DTH) when intersected by 3rd Order Vertical Veins.

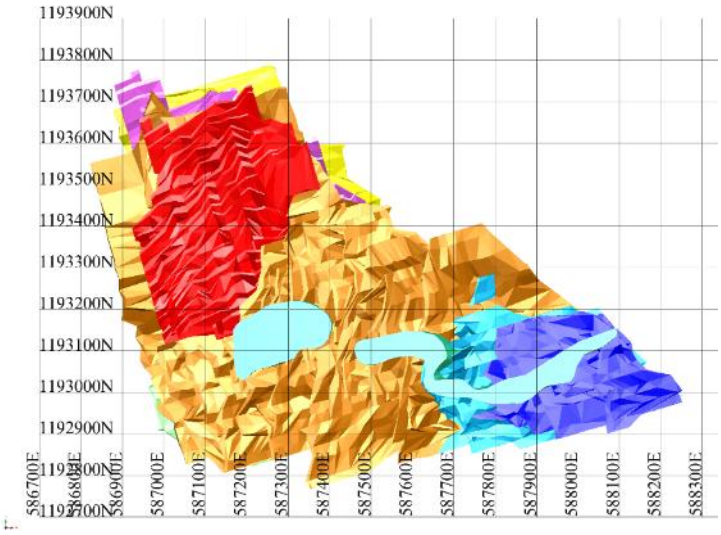
Domain F is composed of the Upper 2nd Order Veins, bound above by Vein 3 and below by Vein 1. It contains 19 packages with 23 veins and 4.04 Mm^3. The veins have a strike of approximately 50° in the west which swings south resulting in a strike of approximately 100° in the east (change in strike on section line W285).

Domain G is composed of the Lower 2nd Order Veins, bound above by Vein 1 and below by Vein 400. It contains 17 packages with 24 veins and 4.67 Mm^3.

The 3rd Order sub-vertical veins have not been modelled to date and Endeavour recommend this is undertaken in any future work.

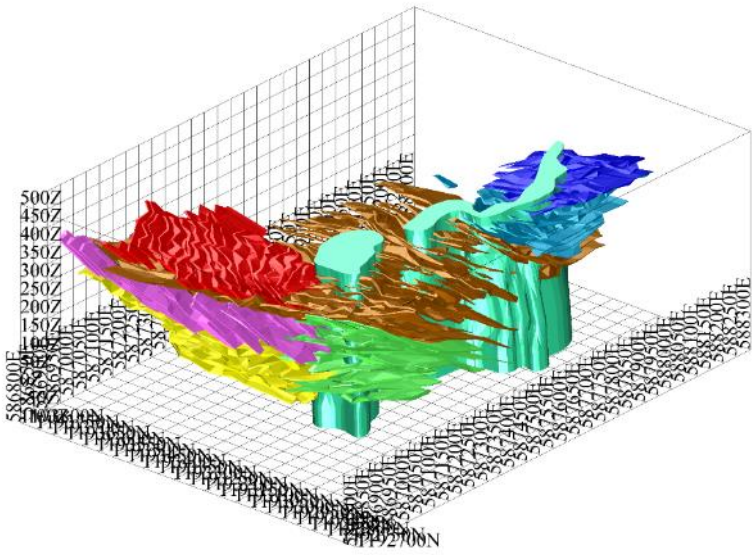
Figure 7.5.3 Kalana Deposit Mineralised Veins by Domain

Plan Veiw



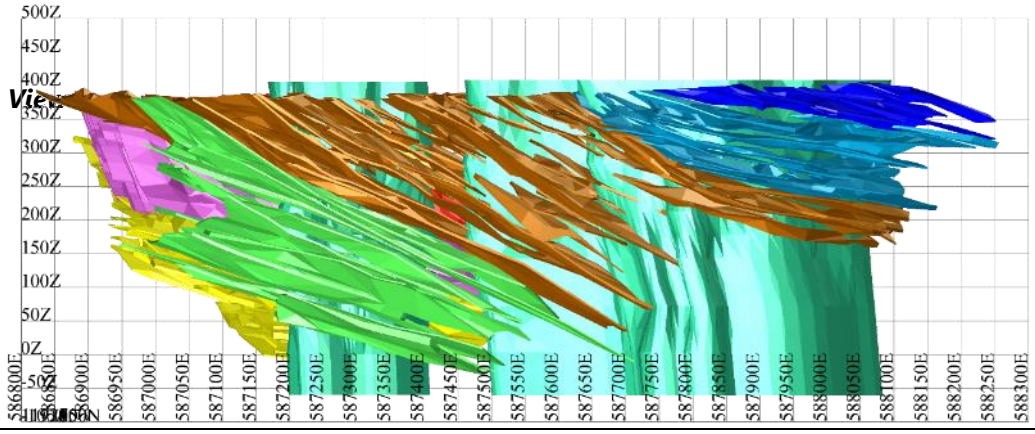
Oblique View Looking NE

Domain	Colour
A	Gold
B	Blue
C	Turquoise
D	Green
E	Yellow
F	Red
G	Purple
Diorite	Light Green



Long Section N - S

Plan View



7.6 Kalanako Deposit Geology

Kalanako is located approximately three kilometres northeast of Kalana. Several mineralised trends have been established that form a northwest-southeast explored corridor of 1,500 m by 250 m. The deposit occurs along a mineralised geophysical structure (aeromagnetic and ground IP), which remains open along strike, Figure 7.7.1.

The depth of saprolite and saprock is between 70 m and 130 m, much deeper than that observed at Kalana. Drilling at Kalanako intersected numerous high strain zones, packets of densely laminated quartz veins with sulphides and locally highly altered and mineralised felsic intrusive rocks.

The volcano-sediments, shear zones and felsic intrusions are cut by flat dioritic dykes (0° to 20° dipping) that are comparable to those at Kalana.

7.7 Kalanako Mineralisation

Kalanako mineralisation is associated with the felsic intrusive rocks and the quartz stockworks that occur along northwest-southeast striking shear zones, parallel or less than 10° in azimuth from the main induced polarisation (IP) boundary between a low and a high IP gradient domain, Figure 7.7.1.

Thirty-four veins have been modelled, six of which account for two-thirds of the mineralised volumes (referred to as "major veins"). The veins have an average thickness of 7.2 m down the hole.

The sub-parallel mineralised zones are typically five metres to 15 m wide, with strike lengths of 250 m to 500 m, and are steeply dipping to the east. Kalanako typically has high-grade intercepts in the oxide part of the deposit.

Kalanako mineralisation appears a bit older than that at Kalana, emplaced in an earlier stage of the late-Eburnean orogenic phase.

Figure 7.7.1 Kalanako Geophysics

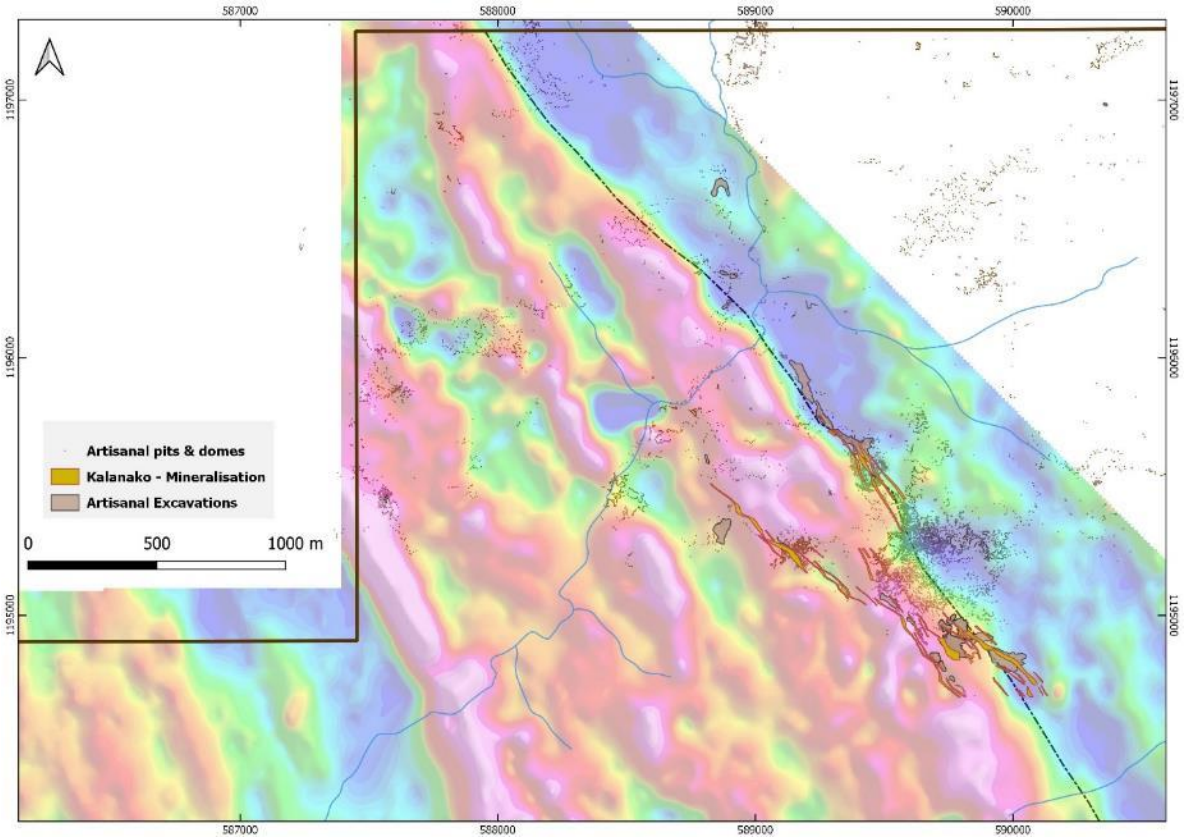


Figure 7.7.2 Kalanako Mineralised Veins and DH Collars (Plan View)

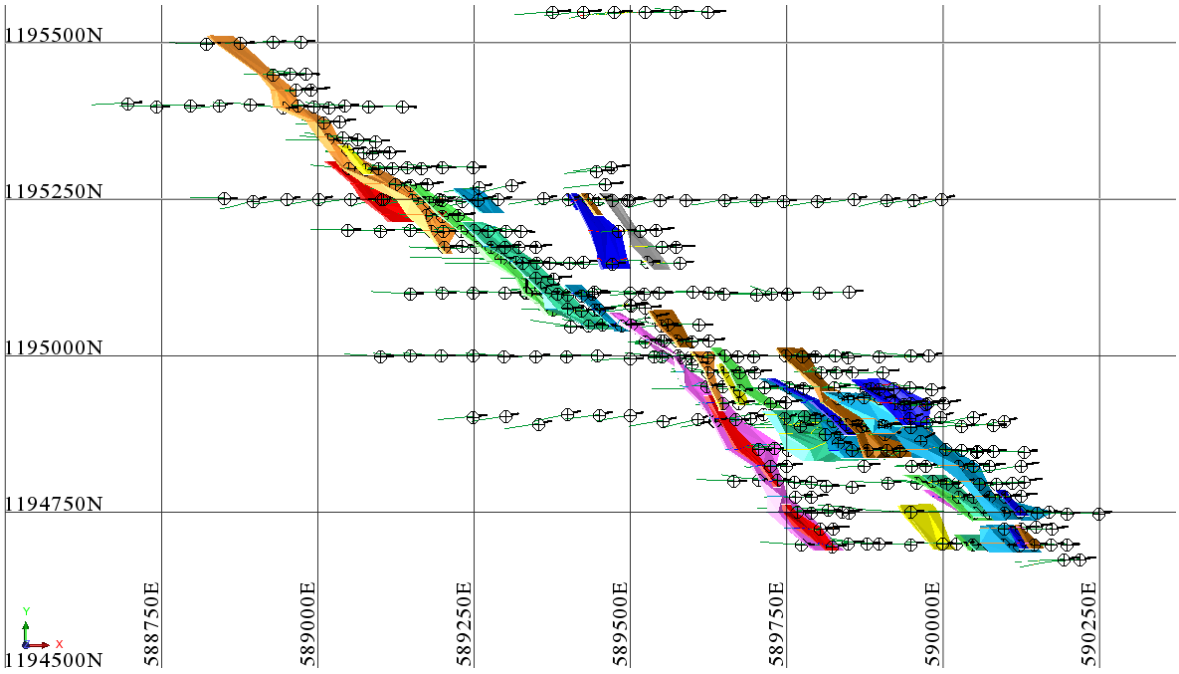
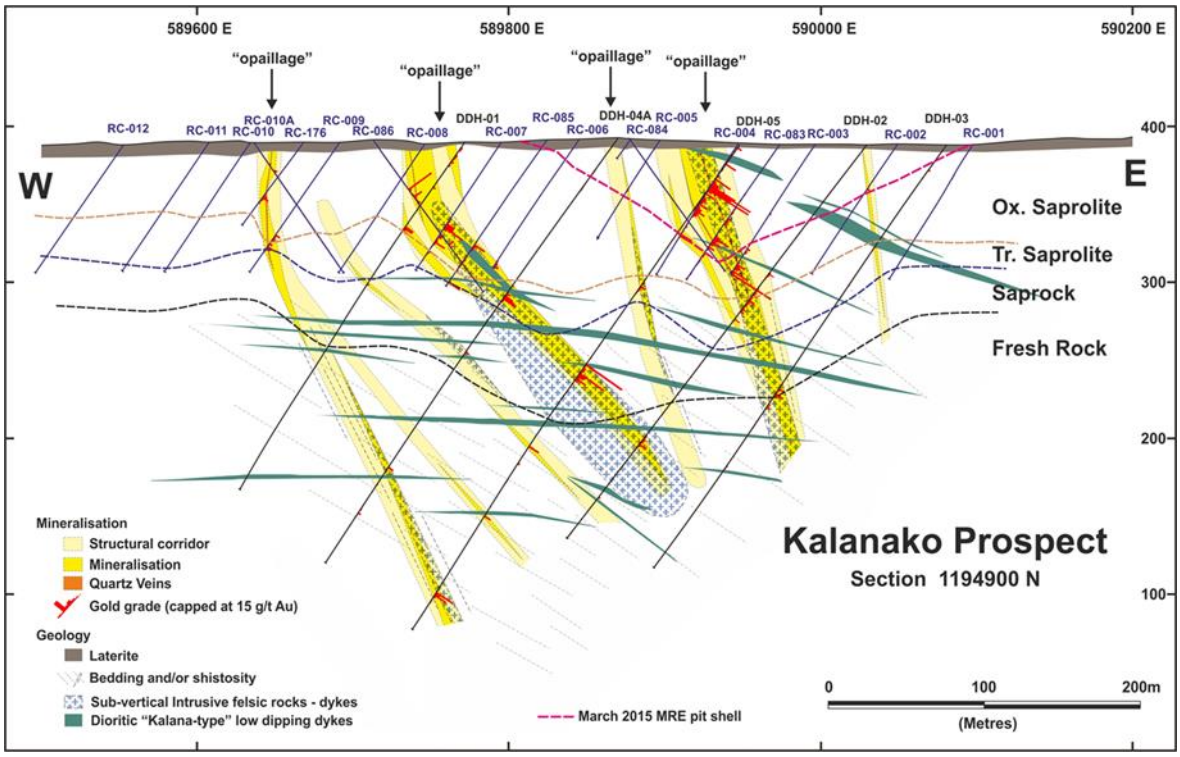


Figure 7.7.3 Kalanako Geological Cross-Section

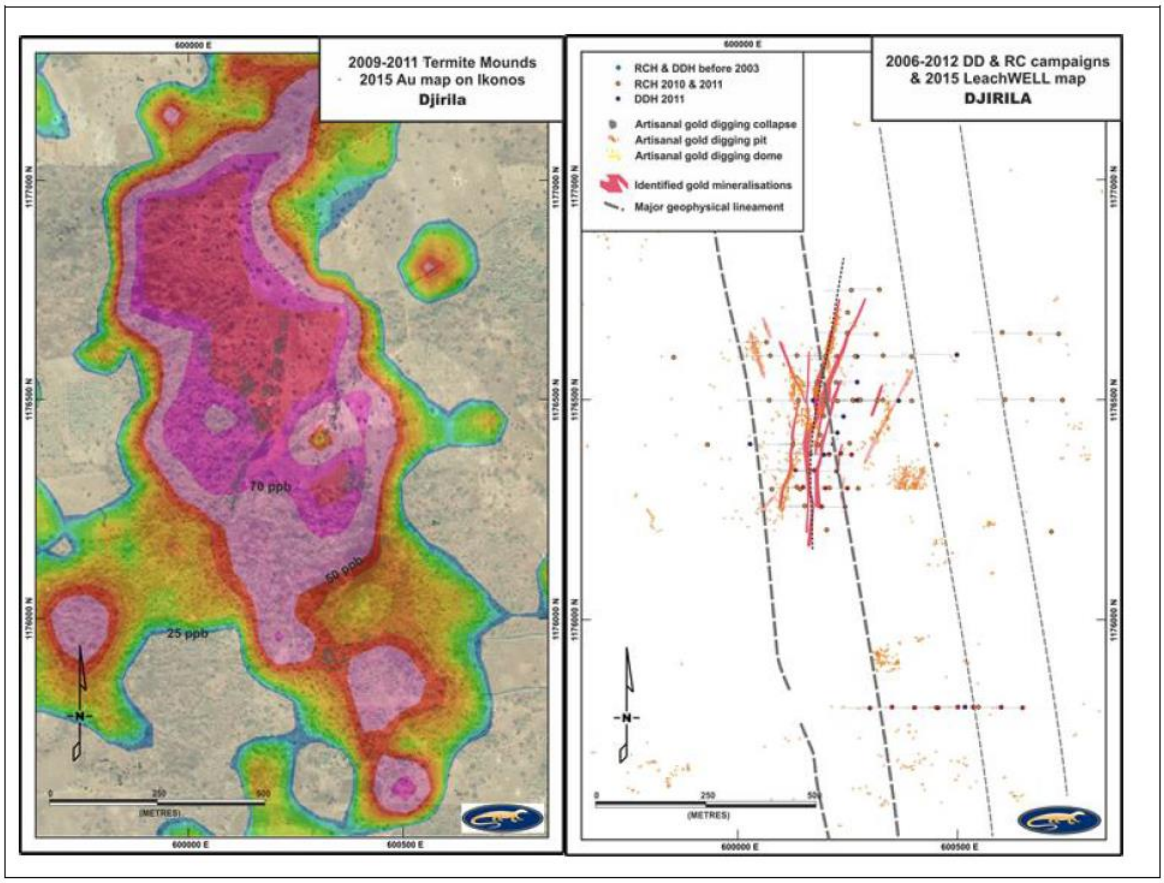


Source: Denny Jones 2017

7.8 Other Deposits

Djirila lies approximately 22 km southeast of Kalana. It was discovered by SONAREM in the 1960's, explored by Avnel Gold in 2004. The saprolite depth is over 70 m, much deeper than Kalana. Mineralisation is associated with high strain zones rich in quartz and quartz veins. Local dioritic dykes can be found.

Figure 7.8.1 Djirila Termite Mound Gold Anomaly and Drill Hole Locations



(Source: Avnel 2015)

7.9 Tailing Storage Facilities (TSFs)

Two Tailing Storage Facilities (TSFs) exist within the Kalana Project, the TSF proper ("TSF") located halfway between Kalana and Kalanako, and the small TSF ("Shaft No. 2 TSF") within the fence close to the plant, see Figure 7.9.1.

The TSF is located 600 m northeast outside the Kalana UG Mine fence, approximately halfway to Kalanako. It was originally constructed in the 1980's by SOGEMORK ("the Russians") as a single impoundment in the shape of an elongated horseshoe. SOGEMORK subsequently sub-divided it to retain approximately 200,000 t of underground tailings and 50,000 t of oxide tailings in separate zones.

The TSF was re-opened in 2004 with a three-metre lift, and extended in 2006, 2012 and 2016 by SOMIKA (Avnel) and closed (i.e. no new material added, not covered or rehabilitated) in December 2017 by SOMIKA (Endeavour). It is of an upstream construction, side-hill impoundment design and has a similar footprint to that created by SOGEMORK with a number of lifts.

The Shaft No. 2 TSF is located within the Kalana UG Mine fenced perimeter, to the west of the old ventilation shaft. It was the original open-pit excavated by SOGEMORK in the 1980's. The exact size, geometry and gold recovered is unknown (the pit was reportedly approximately 100 m long by 50 m wide and 5 m deep, with about 10 kt of mineralised material excavated). It was subsequently infilled with plant tailings from underground workings generated by SOMIKA (Avnel) between 2004 and 2009.

Figure 7.9.1 Kalana TSF Locations



(Source: Google Earth Pro. Image date: January 2016.)

7.10 Mineralisation

The majority of the gold in the quartz veins at Kalana is present as free gold with the gold commonly occurring as grains and small nuggets. Fine grained gold is also associated with arsenopyrite in the quartz veins. There is no disseminated gold in the hangingwall (HW) and footwall (FW) of the veins. In the HW and FW of the main veins of a package, there are millimetric veins (such as the one shown in Figure 7.10.1) which contain gold.

Figure 7.10.1 Visible Gold in 2nd Order Quartz Vein (Drill Hole KA-SEK-DD083a)



(Source: Avnel, 2014)

The grade estimation of the mineralised veins and vein packages at Kalana and Kalanako are hampered by the very coarse nature of the gold; the drill hole databases have numerous instances of barren assays with a number of visible gold grains logged in a known mineralised vein / lode.

The following is taken from SGS, 2011:

"One representative composite sample, identified as KA-M Combined Sample, weighing 4.985 kg, was submitted by SGS Metallurgical Operations for a microscopic gold deportment study on behalf of Iamgold Corporation in December 2011. The objectives of this investigation were to determine the occurrence of gold in the samples and identify and evaluate any mineralogical factors that may affect recovery.

A comprehensive mineralogical and analytical approach, including fire assay, heavy liquid separation (HLS), superpanning (SP), ore microscopy and scanning electron microscope with energy-dispersive spectrometry (SEM-EDS) was used to carry out this microscopic gold deportment study. Representative digital photomicrographs were taken to illustrate the occurrence of gold within the samples.

Gold minerals mainly occurred as native gold with an average content of 88.5% gold, 10.0% silver and the remainder as trace elements.

The overall gold distribution analysis showed that liberated gold accounted for approximately 95.5% of the total gold assay, with a size range of 0.6 – 739.5 µm (average 96.6 µm). The gold attached to silicates, arsenopyrite, their binaries and other minerals accounted for 3.10%, with a size range of 0.6 to 129.4 µm (average 11.8 µm). The gold locked in arsenopyrite, silicates and other minerals accounted for 1.37%, with a size range of 0.6 to 43.9 µm (average 3.3 µm).

The exposed gold grains (liberated and attached) accounted for approximately 98.6% of the total gold assay, mostly coarse liberated gold. Locked gold accounts for 1.37% of the total gold assay. Most of the locked gold grains were fine grained and associated with arsenopyrite and silicates.

It is noted that the occurrence of large liberated gold may cause inconsistent assay results, especially for the small sample size used in this study."

8.0 DEPOSIT TYPES

Kalana and Kalanako are orogenic gold deposits lying within a Birimian-aged volcano-sedimentary sequence of rocks that is known to host several other orogenic gold deposits, including Kodieran, Yanfolila, Morila and Syama.

Orogenic gold deposits in West Africa are typically shear hosted and developed along strike-slip fault systems linked to late stage, non-orthogonal, crustal thinning. Hence, the gold mineralisation is structurally and lithologically controlled and is a function of the association with regional-scale deformation zones.

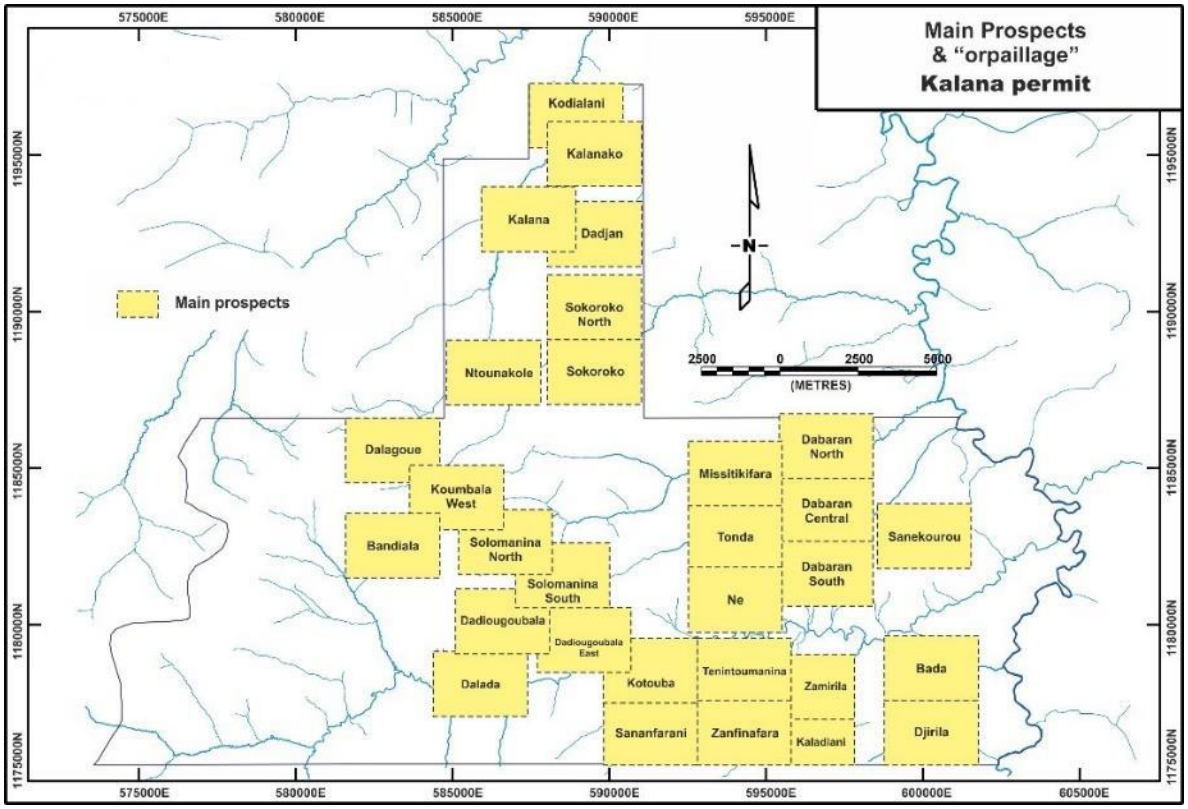
West African orogenic gold mineralisation often has a high fraction of coarse gold, significantly increasing the variability of the mineralisation and grade.

9.0 EXPLORATION

Exploration work described in this Section only covers the period from April 2016 to September 2020. Historical NI 43-101 Technical Reports for the Kalana Project detail exploration activities up to the end of 2004 (Snowden, 2005), 2013 (Snowden, 2014), and April 2016 (Snowden, 2016) respectively.

After the completion of the SOMIKA (Avnel) 2015 delineation drilling campaign on the Kalana deposit, exploration focused on the update of the artisanal work mapping which was completed in early 2016. The targets were subsequently evaluated and ranked based on the gold and/or arsenic anomalies and the artisanal work activity. Figure 9.0.1 presents the location of the Kalana Project exploration prospects.

Figure 9.0.1 Kalana Project Exploration Prospects Location Map



9.1 Advanced Geochemistry

From April 2016, selected 1st Priority exploration targets were subjected to an advanced geochemistry programme with sampling of artisanal (also known as "orpaillage") pit rejects (tailings) based on a 10 m by 10 m grid. In the case of abundant and coarse quartz material, a separate quartz grab sample was also collected.

The samples were assayed using the LeachWELL two kilogram 24-hour bottle roll method at the Bigs Global laboratory located in Ouagadougou, Burkina Faso.

Table 9.1-1 Geochemical Samples 2016-2020

Prospect	No. Orpaillage Reject Samples	Quartz Grab Samples
Solomanina	3,171	1,066
Bandiala	1,409	179
Tonda	823	26
Sanekourou	1,861 (only 945 results available)	219
Dabaran North	1,326	646
Dabaran South	427	139
Dadiougoubala	630	104
Total	**9,647**	**2,379**

Solomanina

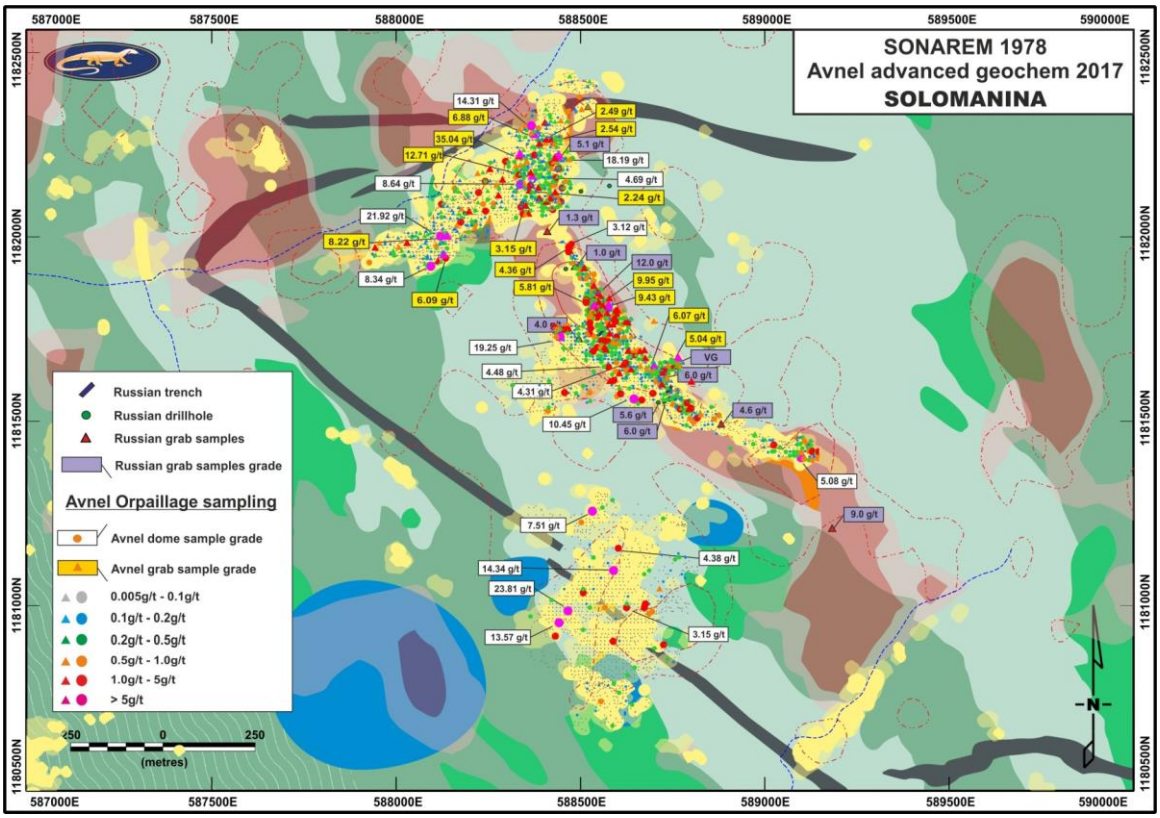
Solomanina is located 12 km south of Kalana town in the centre of the Kalana Permit and includes three separate artisanal workings:

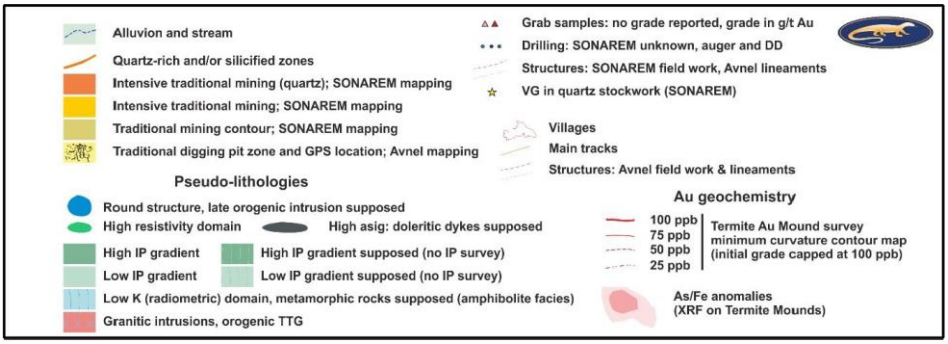
- The main area comprises an old orpaillage overlapping a gold-arsenic anomaly.
- The second area is located along a seasonal river but is along strike of the main anomaly.
- The third area in the south of the Prospect has a weak gold signal and no arsenic.

SONAREM explored the Solomanina main target area between 1978 and 1983 and described gold-bearing quartz veins. They collected grab samples, excavated two trenches and drilled a few holes in the area. Some of the grab samples returned excellent grades (Figure 9.1.1).

SOMIKA (Endeavour) sampled the three artisanal workings in 2016 and 2017. Gold grades up to 35 g/t were obtained in artisanal rejects and grab samples. A mineralised corridor was defined crossing both the Solomanina Main and Solomanina River areas. The southern area returned scarce high grades in the tailings and the quartz vein grab samples did not return significant grades.

Figure 9.1.1 Solomanina Orpaillage Sampling Results



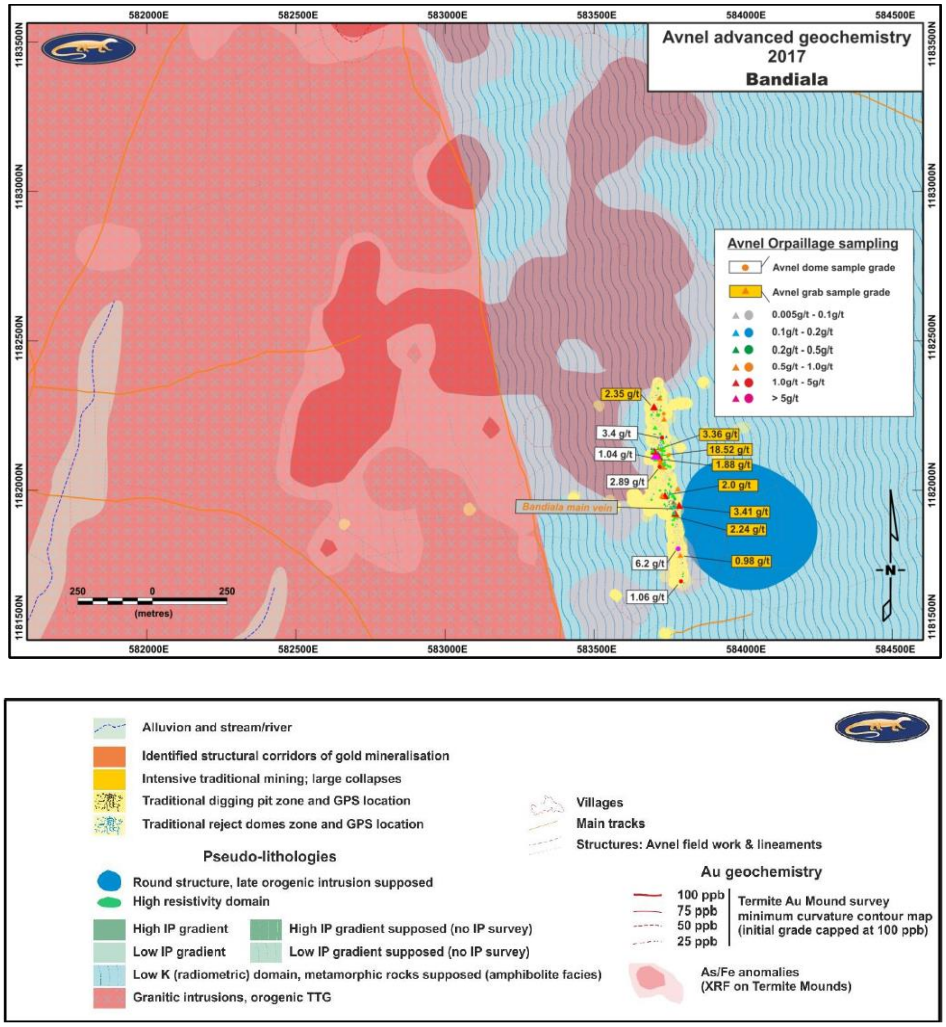


Bandiala

Bandiala is positioned in the western part of the Permit. The prospect comprises an old 650 m long orpaillage centred on a thick three to five metre, steep quartz vein. The orpaillage is located near the contact between the granite body and the volcano sediments found in the majority of the Permit.

SOMIKA (Endeavour) completed an advanced geochemistry sampling on the orpaillage at a 5 m by 5 m grid in 2017. The results delineated three distinctive, well-constrained gold-rich zones, Figure 9.1.2.

Figure 9.1.2 Bandiala Orpaillage Sampling Results



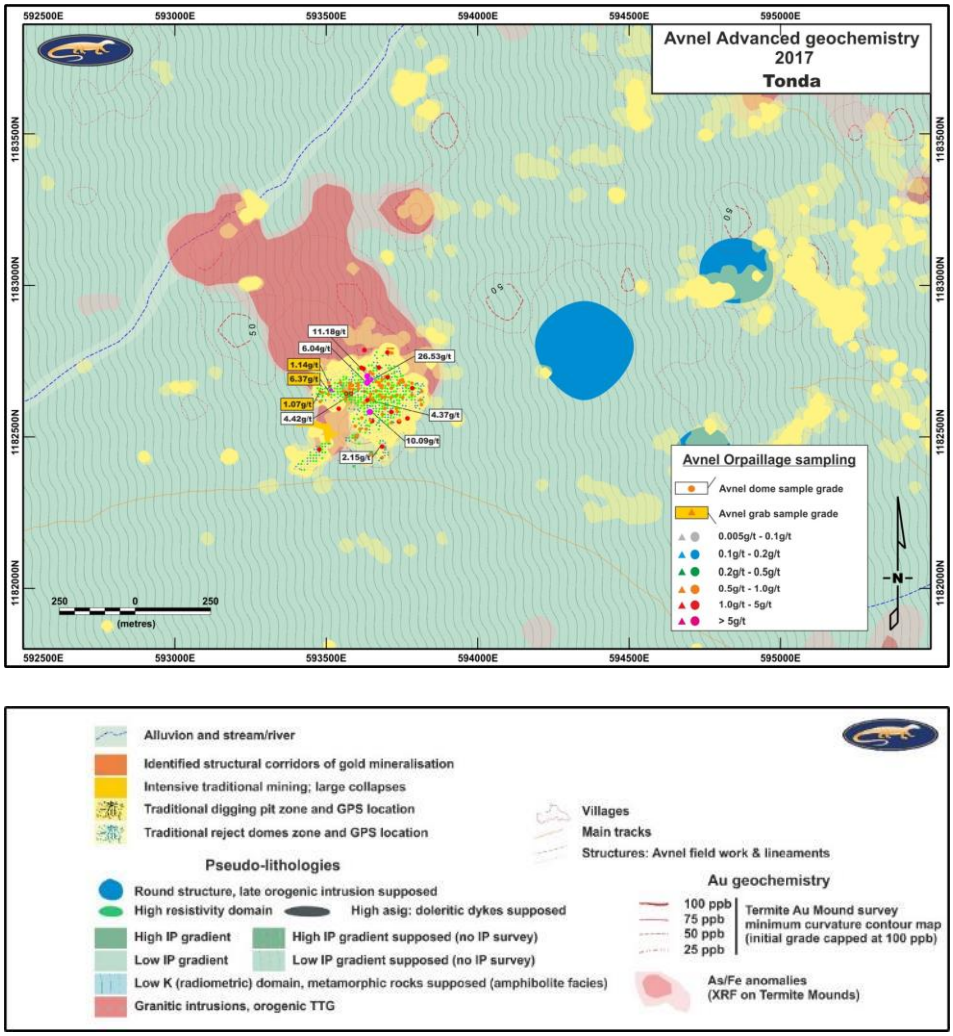
Tonda

Tonda is located 12 km southeast of Kalana town and is characterised by a large gold-arsenic anomaly topped by a well-developed orpaillage zone.

SOMIKA (Endeavour) completed an advanced geochemistry sampling on the orpaillage at a 10 m by 10 m grid in 2016-2017. This area is characterised by the lack of quartz material in the pit rejects, only 26 quartz grab samples were collected over the entire area.

Three other zones located in the eastern part of the prospect were sampled but have not been assayed for gold at the time of writing.

Figure 9.1.3 Tonda Orpaillage Sampling Results

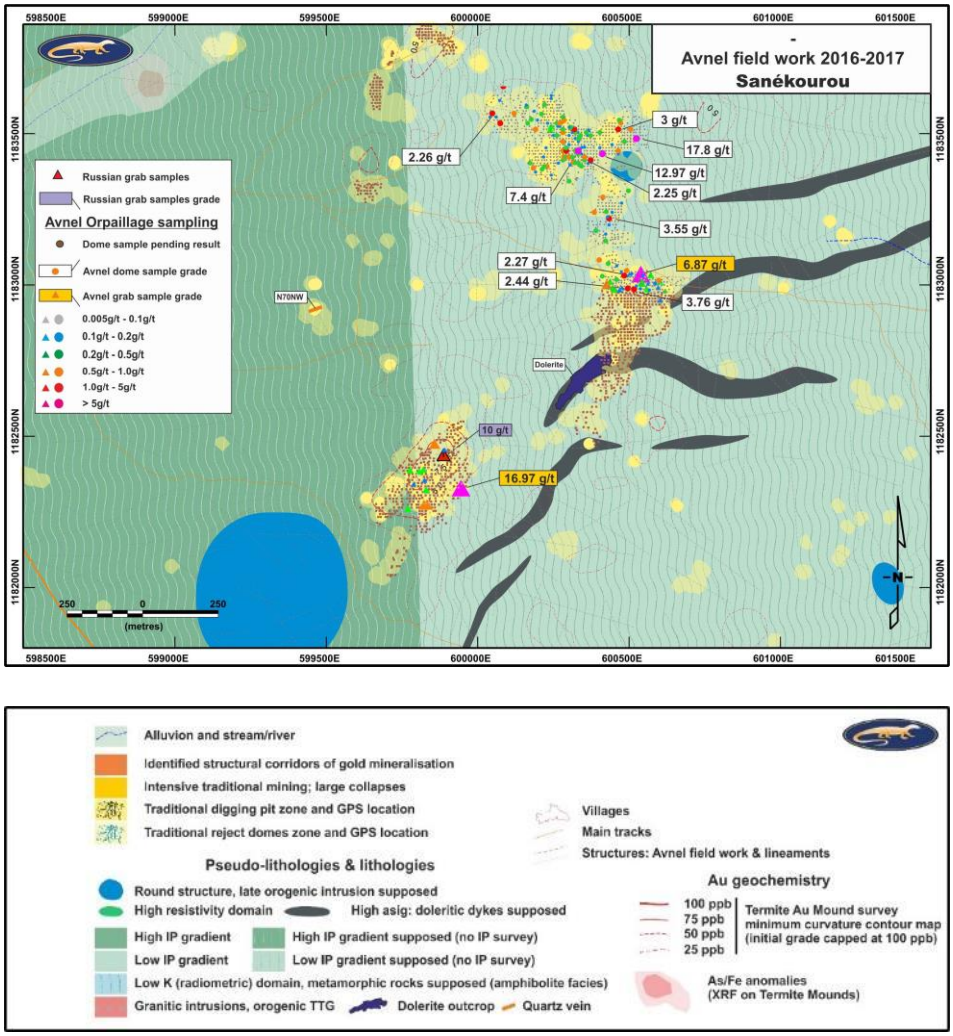


Sanekourou

Sanekourou means the "hill of gold" in the local language. This area is located in the eastern portion of the Permit and includes several orpaillage positioned around "the hill of gold". Sanekourou displays good gold anomalies but no arsenic. SONAREM-SOGEMORK grab sampling in this area in the 1970's returned one sample containing 10 Au g/t.

SOMIKA (Endeavour) completed an advanced geochemistry sampling on various orpaillage of the prospect on a 10 m by 10 m grid in 2017 (Figure 9.1.4). Approximately only half of the assay results are available but good grades in the tailings samples have been noted whilst only a handful of quartz returned grades above 0.5 Au g/t.

Figure 9.1.4 Sanekourou Orpaillage Sampling Results



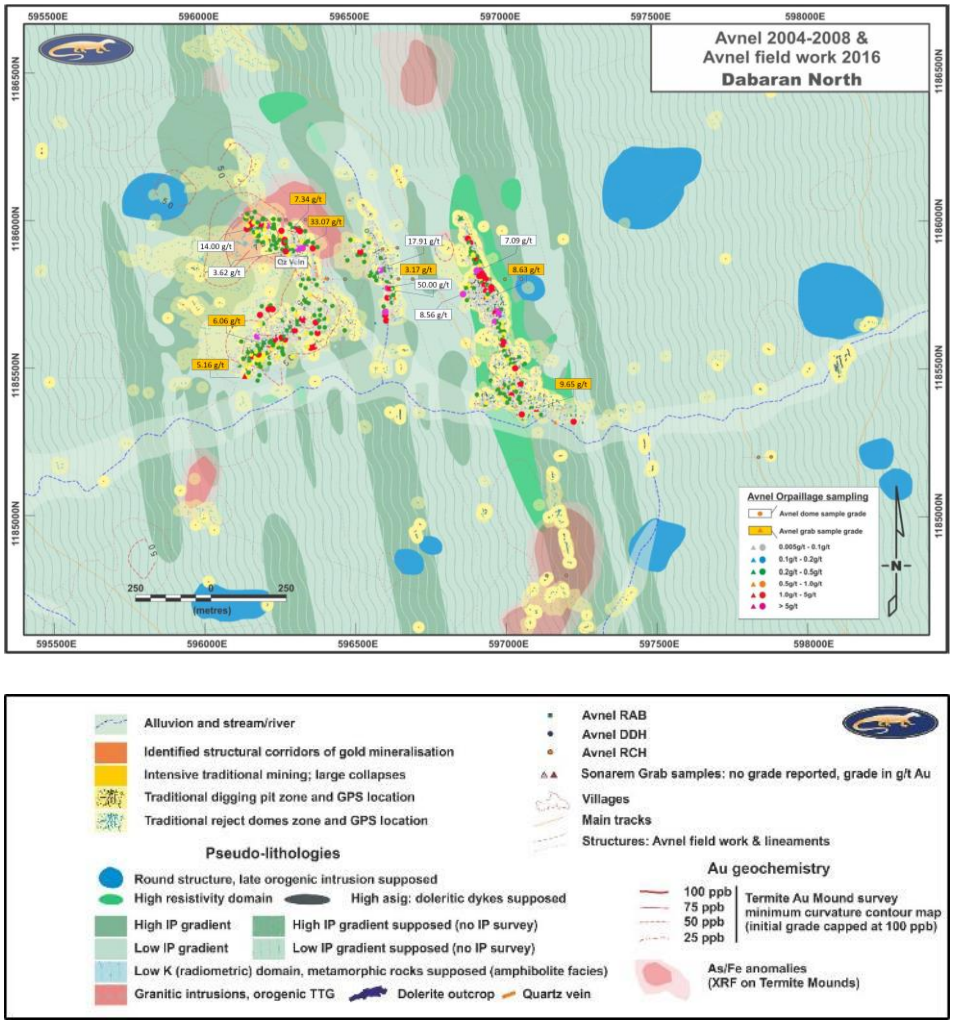
Dabaran

The Dabaran area includes three targets, Dabaran North, South and Centre. Dabaran is located between Tonda and Sanekourou and each target comprises multiple artisanal workings, gold and arsenic anomalies.

SONAREM-SOGEMORK collected grab samples in the Dabaran North area with samples returning grades up to 15.1 Au g/t in the 1970's. In 2005-2006, SOMIKA (Avnel) drilled rotary air blast (RAB) holes in Dabaran North and Dabaran South area with moderate results.

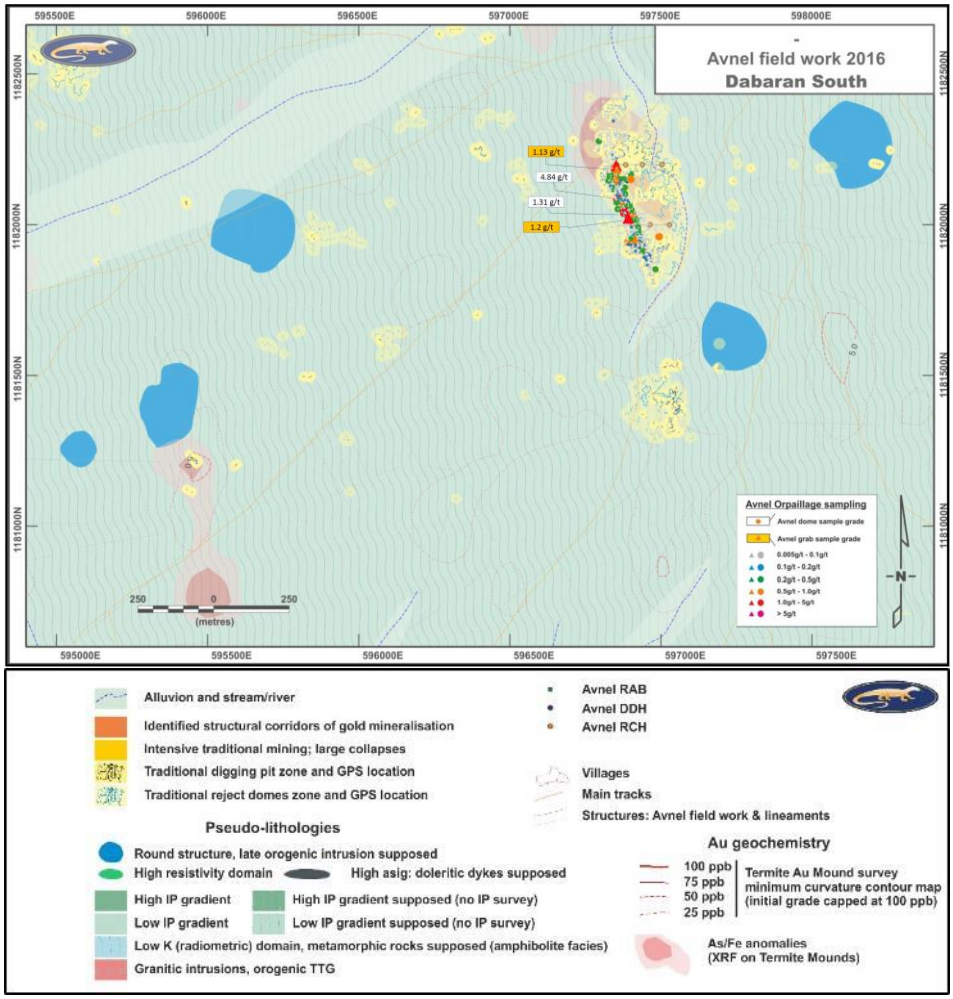
In 2016, SOMIKA (Avnel) sampled the three main orpaillage areas of Dabaran North (Figure 9.1.5). The results obtained in the two NNW-SSE trending orpaillages in the east confirm the mineralisation direction with significant results obtained. The mineralisation control seems more difficult to observe in the orpaillage to the west but significant grades were also obtained.

Figure 9.1.5 Dabaran North Orpaillage Sampling Results



In 2005-2006, SOMIKA (Avnel) also drilled four RAB holes in the Dabaran South area but the results were disappointing. In 2016, SOMIKA (Avnel) sampled Dabaran South and the results were significantly lower than in Dabaran North.

Figure 9.1.6 Dabaran South Orpaillage Sampling Results

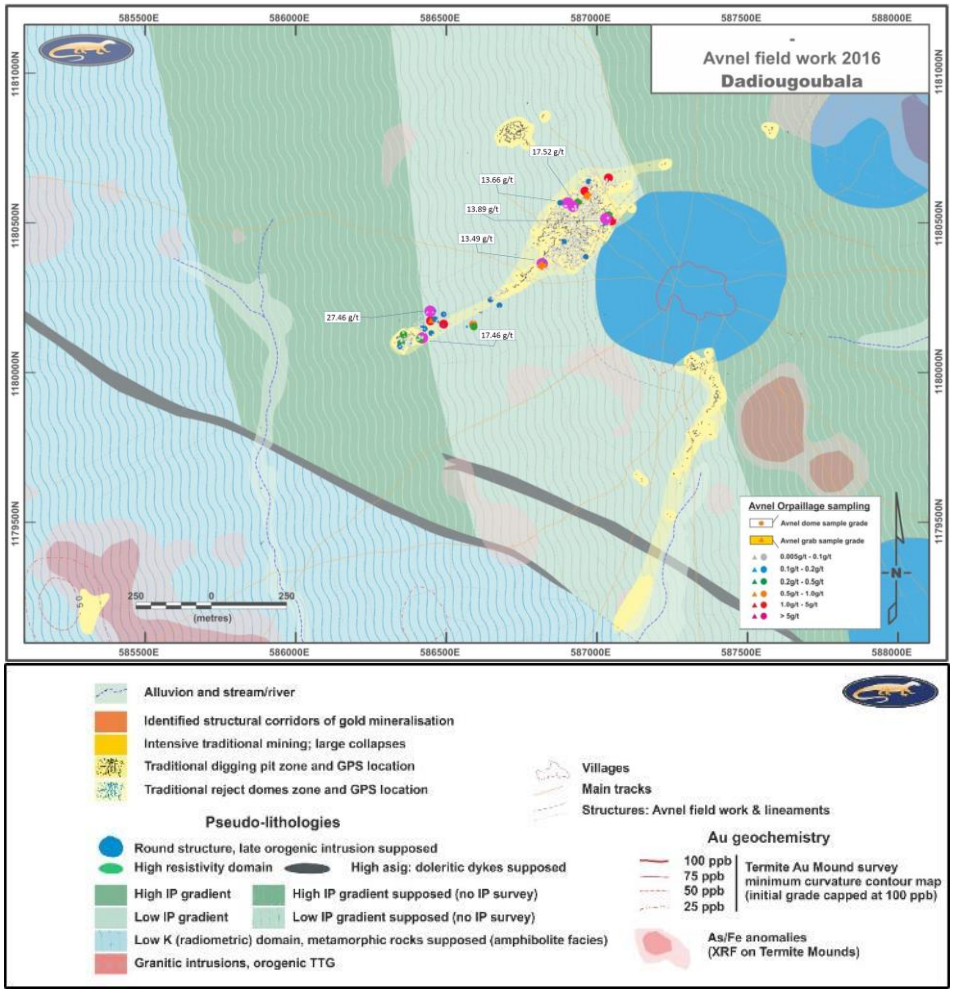


Dadiougoubala

Dadiougoubala is located southeast of the Solomanina target. Based on geophysics, the Russian geologists in the 1980's described a round structure which they thought was an intrusive. Around the potential intrusive, two artisanal workings have developed.

SOMIKA (Avnel) in 2017 collected samples on the biggest orpaillage located on a plateau northwest of Dadiougoubala village (Figure 9.1.7). The sampling mostly consisted of tailings with very few quartz samples taken. Some tailings samples returned scattered high grades but no geometry or mineralisation could be observed.

Figure 9.1.7 Dadiougoubala South Orpaillage Sampling Results

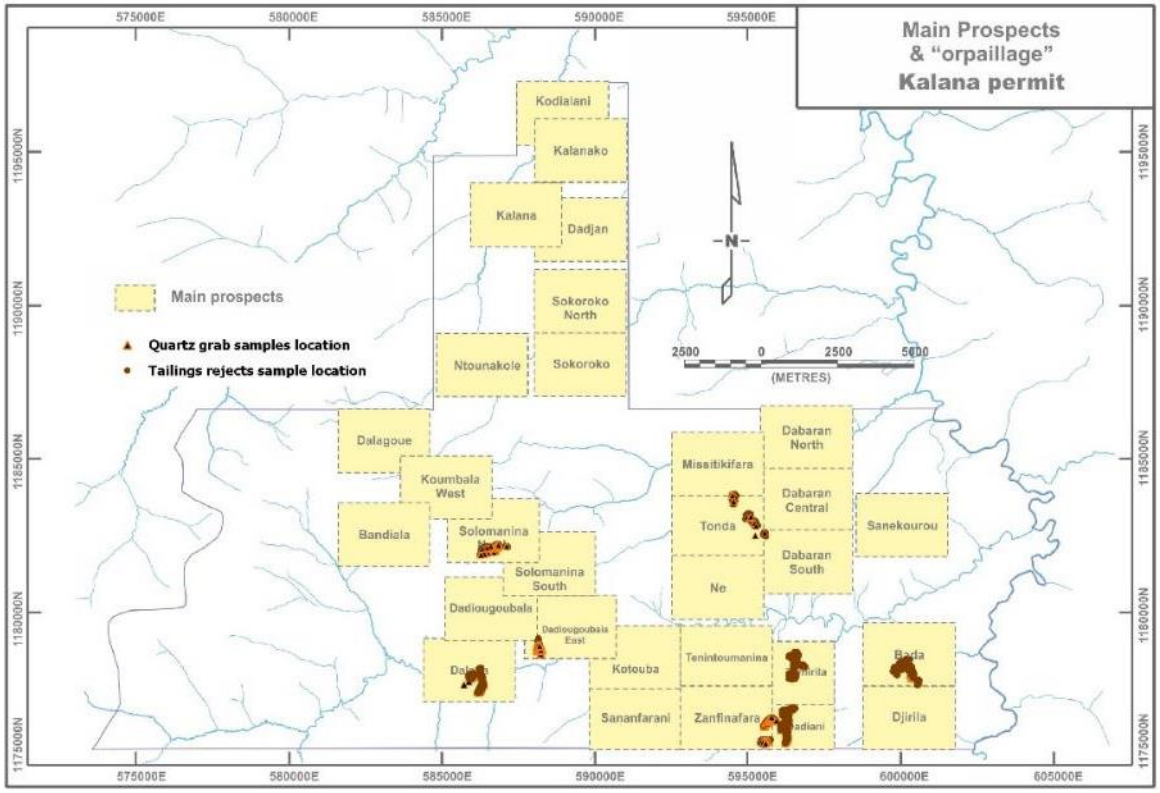


Other Targets

Several other targets (Figure 9.1.8) have been sampled since 2016 but the samples have not been sent to laboratory for assay as of the time of writing. The areas sampled include the 2nd Priority Targets of:

* Kaladiani
* Zamirila
* Zanfinafara
* Dalada,
* Bada
* Dadiougoubala East
* Solomanina North
* Tonda secondary targets

Figure 9.1.8 Orpaillage Sampling On Selected Second Priority Targets, September 2017 - September 2020



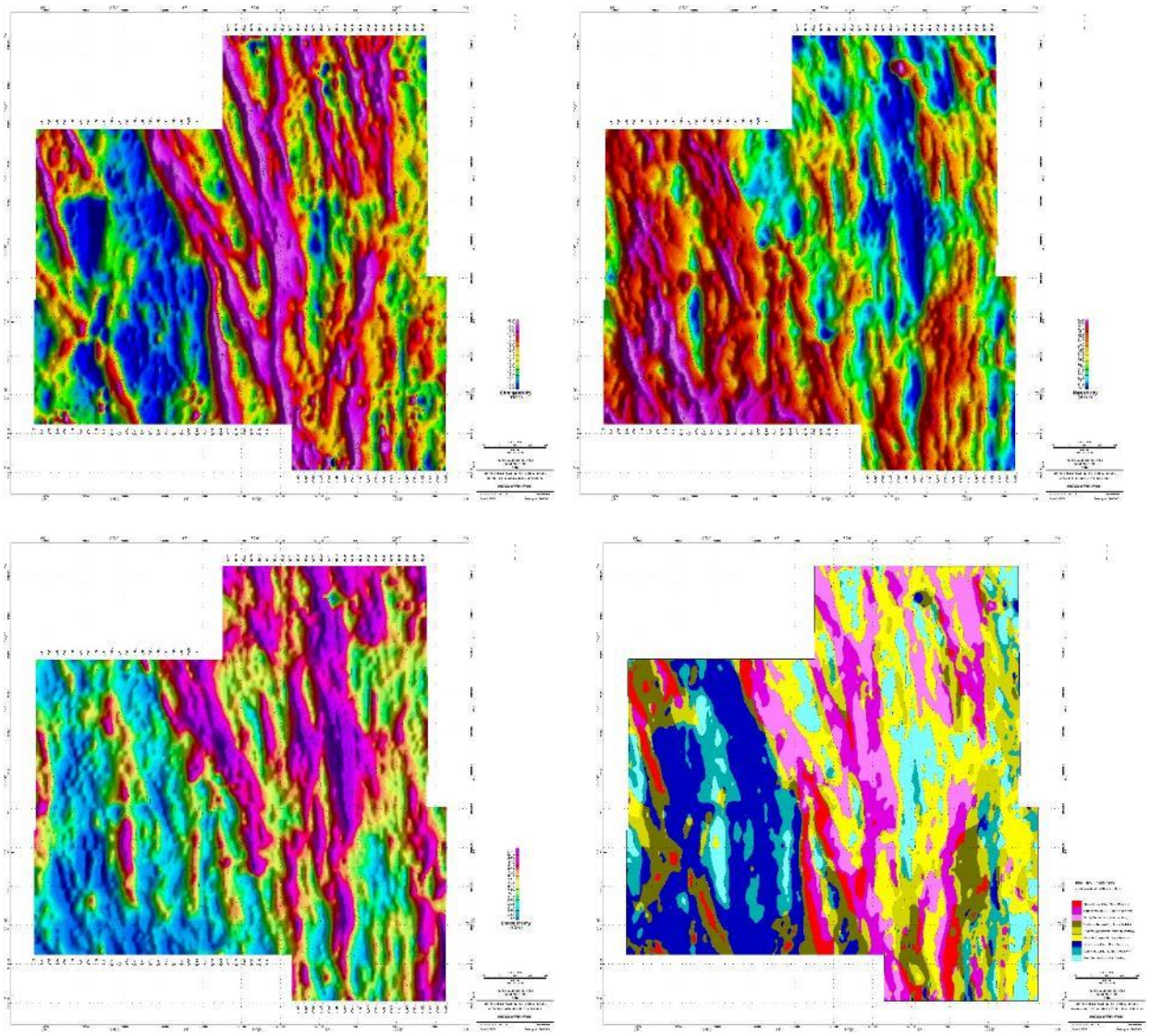
9.2 Ground Geophysics

The Tonda-Dabaran area shows a high concentration of gold anomalies and artisanal workings. Due to the absence of outcrop and the fact that this area does not display any particular structure in the airborne geophysics (due to the thick laterite cover), there is little geological information on this area.

In 2019, Sagax Afrique SA (SAGAX) was contracted by Endeavour Mining to perform an induced polarisation (IP) survey over an area covering 25.8 km^2; measurements were taken every 25 m along lines spaced 100 m, a total of 244 line kilometre surveyed.

Following the survey, SAGAX produced various maps including chargeability, resistivity, conductivity and pseudo-litho, Figure 9.2.1.

Figure 9.2.1 Ground IP Maps Produced by Sagax Afrique SA on the Dabaran-Tonda Area



In addition to the IP survey, SAGAX conducted three pole-dipole array cross-section surveys - two on the Solomanina main target (Figure 9.2.2) and one in Tonda (Figure 9.2.3).

Figure 9.2.2 Solomanina Pole-Dipole Array Cross-Section

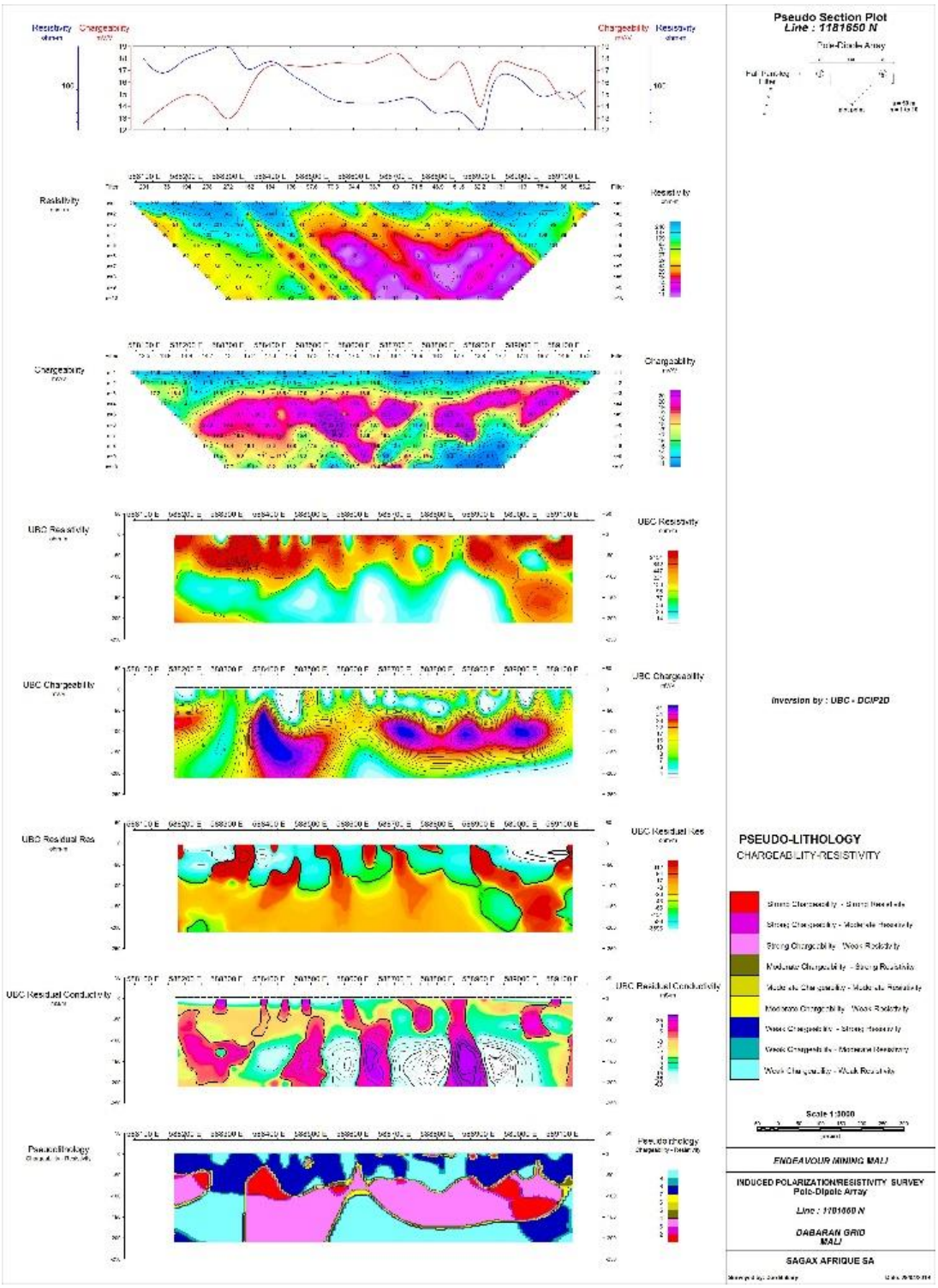
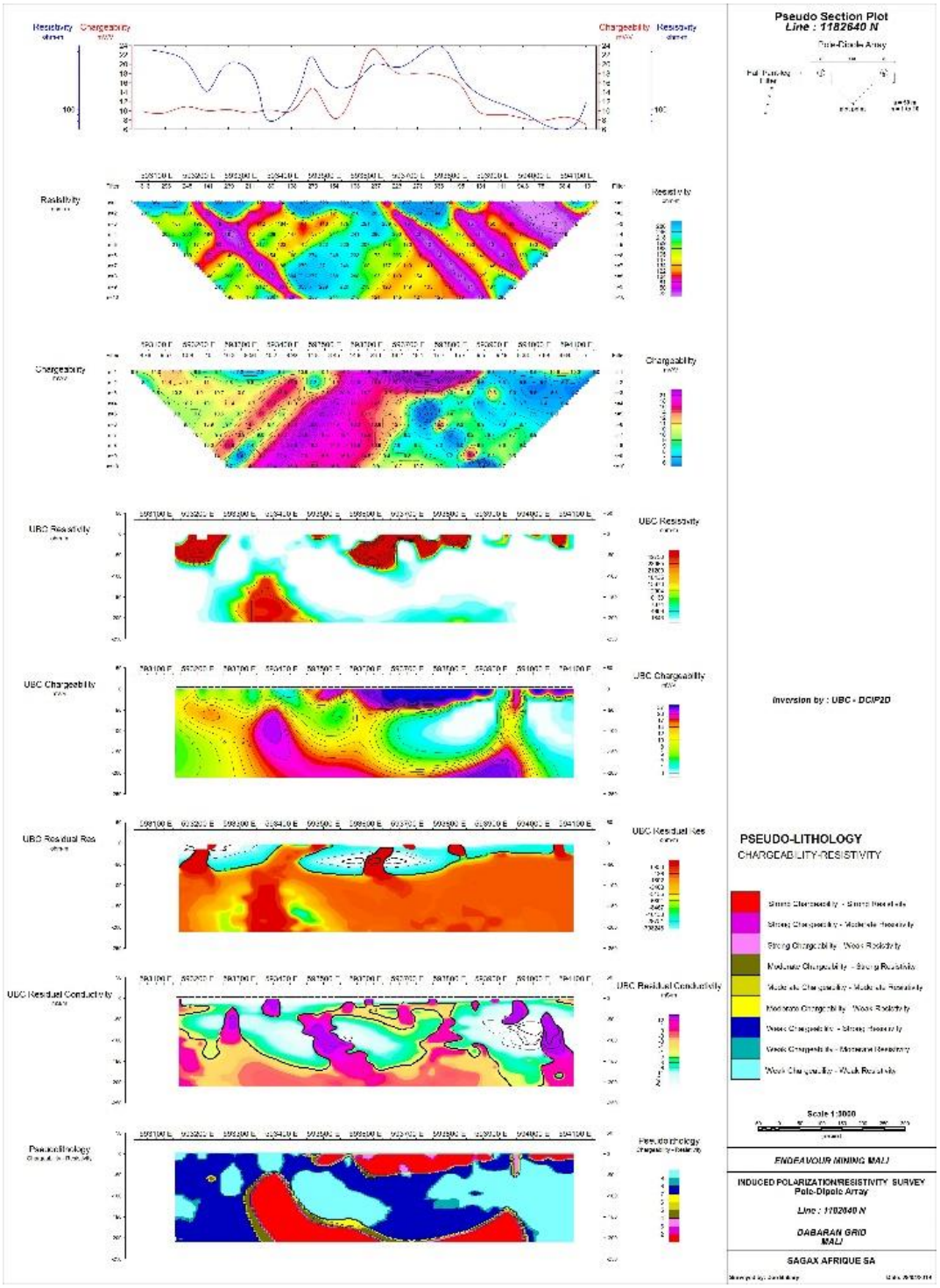


Figure 9.2.3 Tonda Pole-Dipole Array Cross-Section



10.0 DRILLING

10.1 Introduction

The drilling herein referred to was undertaken after the issue of the 2016 Technical Report issued by Snowden on the behalf of Avnel Gold Mining Ltd. The preceding drilling is considered historical and summarised in Section 6 History. The Exploration Team at Kalana has been stable over the last ten years and therefore the procedures and protocols have been developed and refined in-house over time. Hence, the "historical" and "current" drilling data has been collected with the same methods, supervision and overall quality; it is considered to be compatible, comparable and of equal stature by the Qualified Person.

Since June 2016, nearly 97,000 m have been drilled on the different targets across the Kalana Permit including near-mine and exploration targets (Figure 10.1.1 and Table 10.1-1). Two drilling companies were contracted for the completion of the drilling, Geodrill (2016 - 2018) and AMCO (2019).

Figure 10.1.1 Drillhole Location General Map

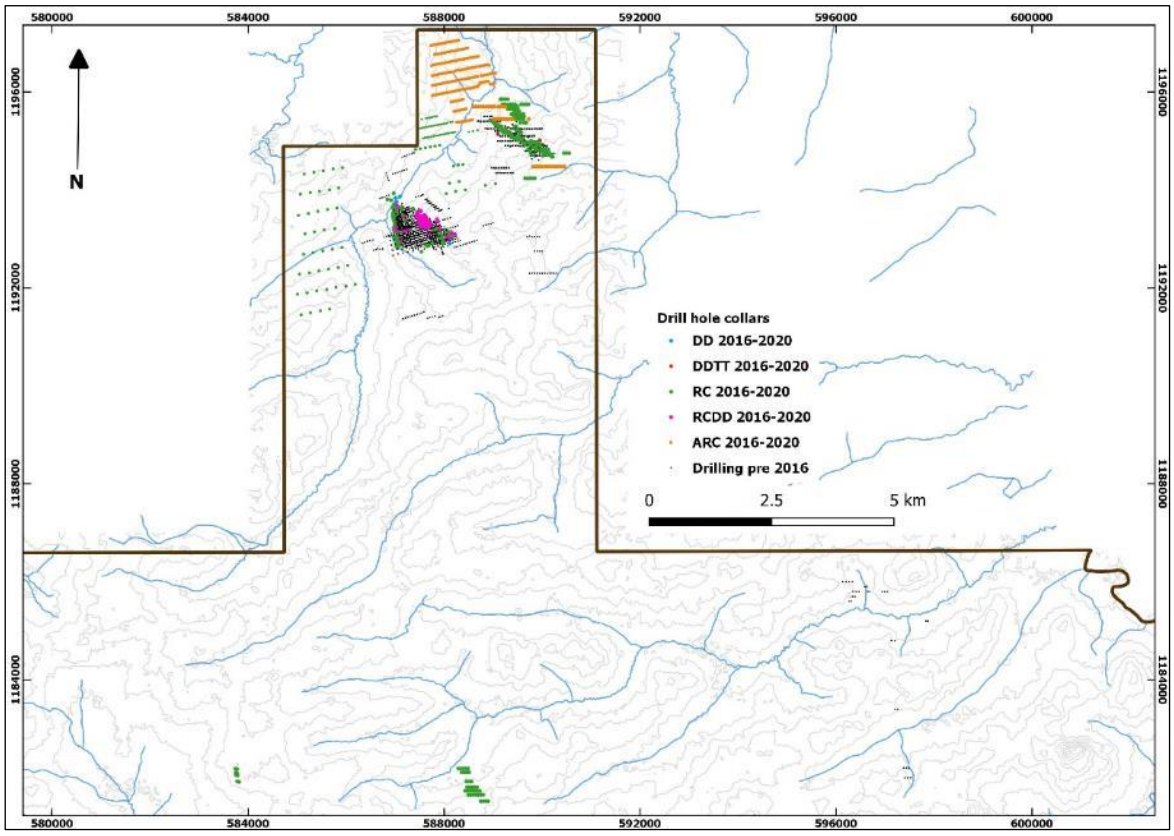


Table 10.1-1 Drillhole Summary, June 2016 to October 2020

Programme	Deposit	No. Holes	Metres
SOMIKA (Avnel) 2016 - 17	Kalanako	82	8,635
SOMIKA (Endeavour) 2017 – 18 Resource Delineation	Kalana	189	44,476
SOMIKA (Endeavour) 2018 West Condemnation	Kalana	57	6,267
SOMIKA (Endeavour) 2018 North Condemnation/Exploration	Kalana	42	3,781
SOMIKA (Endeavour) 2017 - 18 Infill and Exploration	Kalanako and Kalanako Northeast	102	13,305
SOMIKA (Endeavour) 2019 Exploration	Kalanako	42	3,838
SOMIKA (Endeavour) 2019 Target Generation	Kodialani	199	12,044
	Solomani	35	3,541
	Bandiala	8	1,105
Total		**756**	**96,992**

10.2 Procedures

Comprehensive procedures documents have been developed by the SOMIKA geologists on the Kalana Project working for Avnel and subsequently for Endeavour Mining. The documents establish the standard procedures for both diamond drilling (DD) and reverse circulation (RC) drilling (conventional or Air Core (AC)) at the drill site. The manuals propose site layout methodologies and set-out procedures and give examples for the handling of samples, marking of boxes, chip bags and boxes etc. The general guidelines for DD and RC drilling are summarised below.

The procedures described below have been reviewed by external consultants and the Avnel and Endeavour Mining Qualified Person.

Denny Jones stated in the May 2016 Technical Report: "Denny Jones is of the opinion that Avnel personnel have used care in the collection and management of field and assaying exploration data. Furthermore, sample preparation, sample security, and analytical procedures used by Avnel are consistent with generally accepted industry best practice, and the results are adequate for the purpose of Mineral Resource Estimation." The Endeavour Mining Qualified Person, Helen Oliver, concurs and is of the opinion that the new data is also adequate.

10.2.1 Diamond Drilling

Drilling and core logging procedures include downhole survey, core orientation, geotechnical and structural logging, petrological logging and core photography, and specific gravity measurement protocols.

In essence, the DD core is measured, and an orientation line marked on the fresh rock at the rig before being transported to the Exploration Core Yard inside the Kalana Mine perimeter for logging and sampling. Once in the core yard, the orientation line is checked by a geologist and a qualified technician and validated before sampling. After geotechnical and structural logging, digital photographs are taken and samples selected every five metres to measure specific gravity before sampling. The core is cut in half using a diamond saw along the orientation line. Samples are then bagged, weighed, placed into a large rice bag with other primary samples and Quality Control / Quality Assurance (QA/QC) samples (Certified Reference Materials (CRM's) and blanks) and dispatched to the ALS laboratory in Ouagadougou, Burkina Faso.

Core samples are one metre down-the-hole (DTH) long. All data is recorded, validated and entered into the Endeavour Mining proprietary electronic database ("DDHTools") by the Data Manager.

10.2.2 RC Drilling

At the RC drill rig, the RC chips are collected in one metre intervals from the cyclone, collected in labelled rice bags. Information is collected including total sample weight, moisture or water content; any event is recorded in a field book for future reference if required.

Upon reception at the Exploration Core Yard, the RC samples are dried if required before being sampled. The (dried) material is split using a single chute riffle splitter to obtain a two to four kilogram primary sample. The complete reject is then split again using the same method to obtain a Witness Sample to be kept for future reference. Once every 17 samples, a sample is split a third time to generate a Field Duplicate for QA/QC purposes. After splitting, the primary sample is bagged, weighed, placed into a large rice bag with other primary samples and QA/QC samples (CRM's, blanks and field duplicates) and dispatched to the laboratory (Bigs Global or ALS in Ouagadougou, Burkina Faso).

Approximately 500 g of material is sampled from the reject material and is washed and panned. Heavy minerals and visible gold are described and recorded for each sample. Visible gold samples are retained in small, individual, labelled plastic bags.

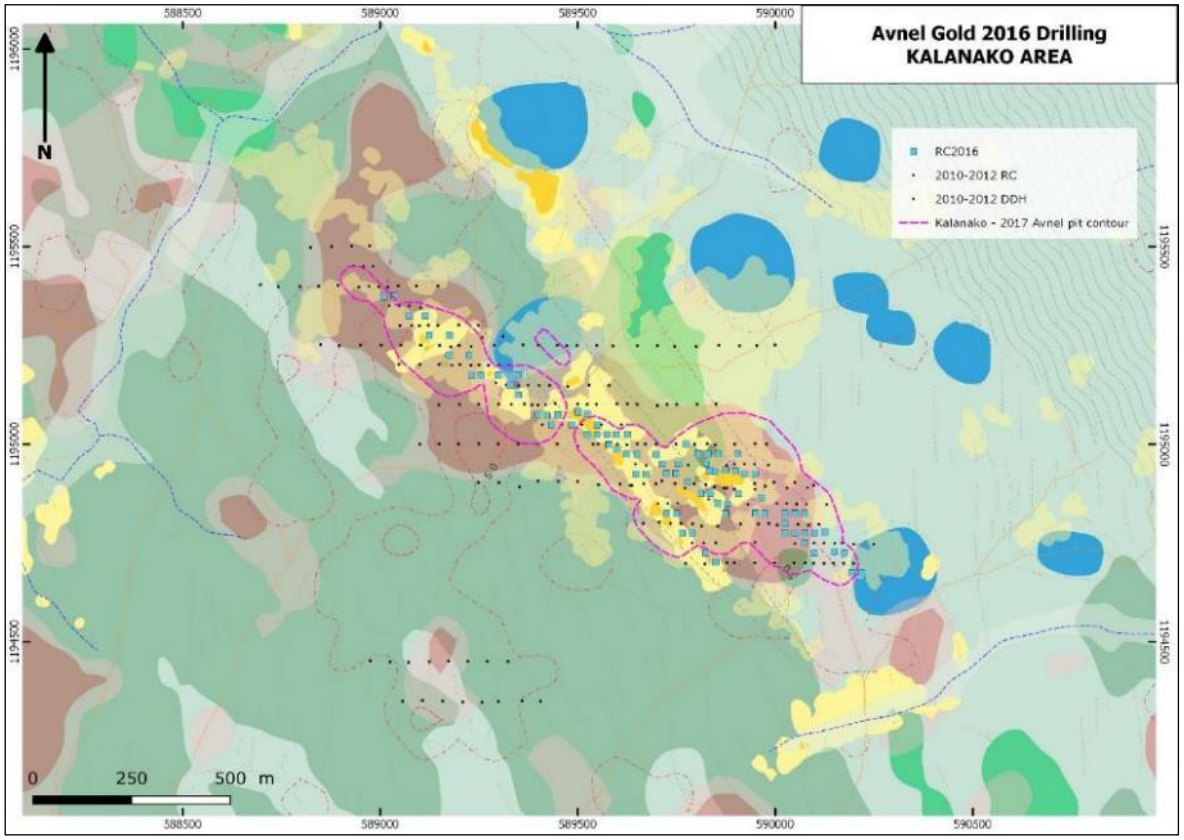
All RC drill material is sampled on a one metre interval except in case of poor recovery, near surface for example.

All data is recorded, validated and entered into the "SiteTools" database by the Data Manager.

10.3 SOMIKA (Avnel) Drilling, 2016 to September 2017

In 2016, SOMIKA (Avnel) completed an 82 RC drill hole programme in the Kalanako area to update the March 2015 Mineral Resource Estimate. The objective was to infill the drill grid as the previous drill spacing of 25 m by 50 m was too broad, resulting in a loss of grade continuity in the resource estimate calculation. A total of 8,635 m of RC drilling was completed to reduce the line spacing to 25 m (Figure 10.3.1).

Figure 10.3.1 2016 SOMIKA (Avnel) Kalanako Drilling



The samples were assayed at Bigs Global using the two kilogram LeachWELL 24-hour bottle roll method with fire assays on the tails on samples for which:

- LeachWELL grade was higher than 0.3 Au g/t.
- Had visible gold; and/or
- Formed internal dilution within a mineralised interval.

The drilling confirmed the mineralisation geometry and continuity and resulted in the update of the Kalanako Mineral Resource Estimation published by Avnel Gold in July 2017.

10.4 SOMIKA (Endeavour) Drilling, September 2017 to March 2019

10.4.1 Kalana Deposit

Following the acquisition of Avnel Gold and its majority shareholding in SOMIKA by Endeavour Mining, SOMIKA (Endeavour) decided to undertake an infill drilling programme on the Kalana and Kalanako deposits.

Between November 2017 and March 2018, a total of 213 holes for 44,476 m were completed on the Kalana deposit as summarised in Table 10.4-1.

Table 10.4-1 SOMIKA (Endeavour) Kalana Drilling Summary 2017 – 2018

Deposit	RC		RCD			DD	
	#	m	#	RC m	DD m	#	m
Kalana	108	18,647	77	15,537	9,257	4	1,347

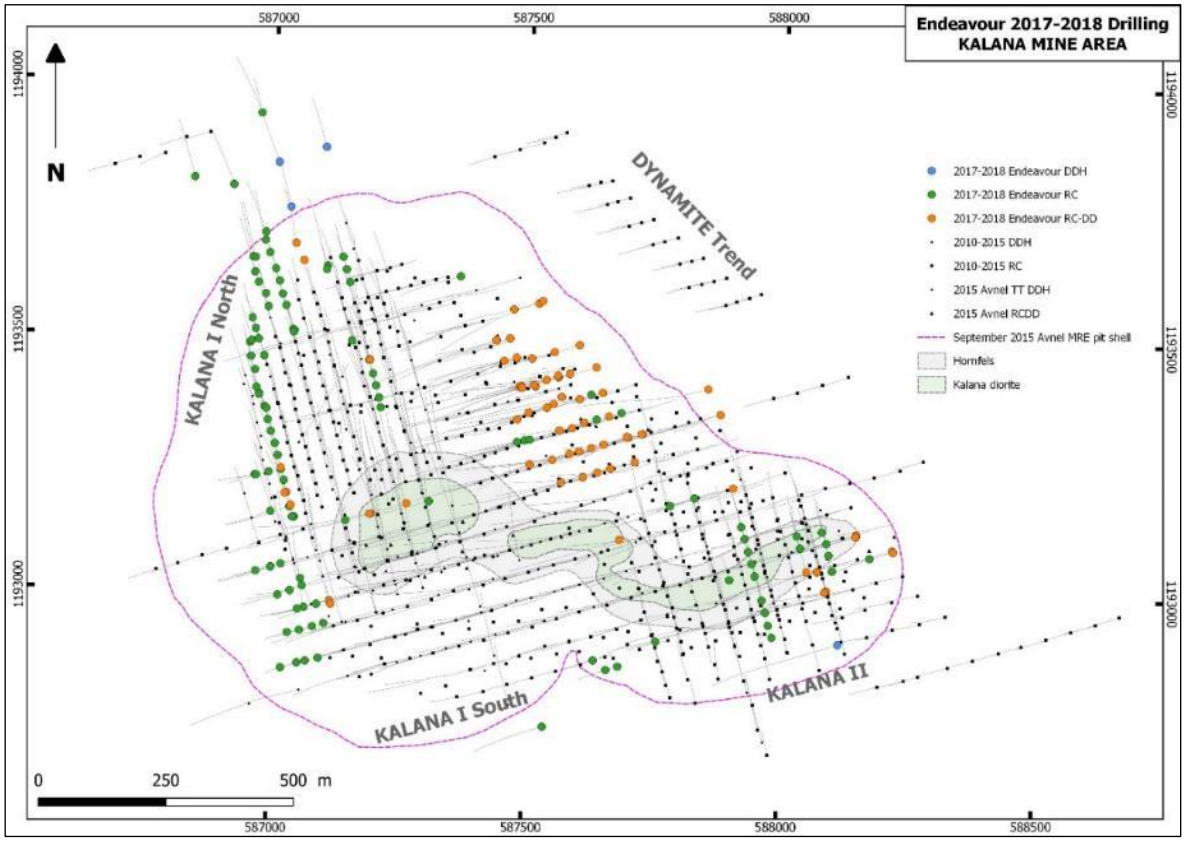
The Kalana exploration drilling programme had four objectives:

- Infill drilling in selected areas of the deposit that required additional information for a better understanding of the geometry, i.e. the 3rd Order Sub-vertical system in the north of the deposit.
- Deep drilling in the northeast to test Veins 1 to 17 extensions at depth.
- Convert Vein 1 Inferred resource near surface in the vicinity of the Shaft No. 2 Tailings Storage Facility.
- Step-out drilling to test potential deposit extension in all directions.

The drilling campaign completed by Geodrill Limited included 108 RC holes, four diamond drillholes and 77 RC-DD holes for which the drillhole was collared in RC and tailed in diamond drilling (RCD) (Figure 10.4.1). PQ-HQ diameters were used for the core drilling to maximise the sample size as per previous SOMIKA (Avnel) drilling methodology. RC holes with a diamond drilled tail (RCD) were drilled using a HQ diameter.

The majority of the drilling was done in RC as it provides bigger, better (in term of quartz recovery in the saprolite) and cheaper samples.

Figure 10.4.1 2017 - 2018 SOMIKA (Endeavour) Kalana Exploration Drilling



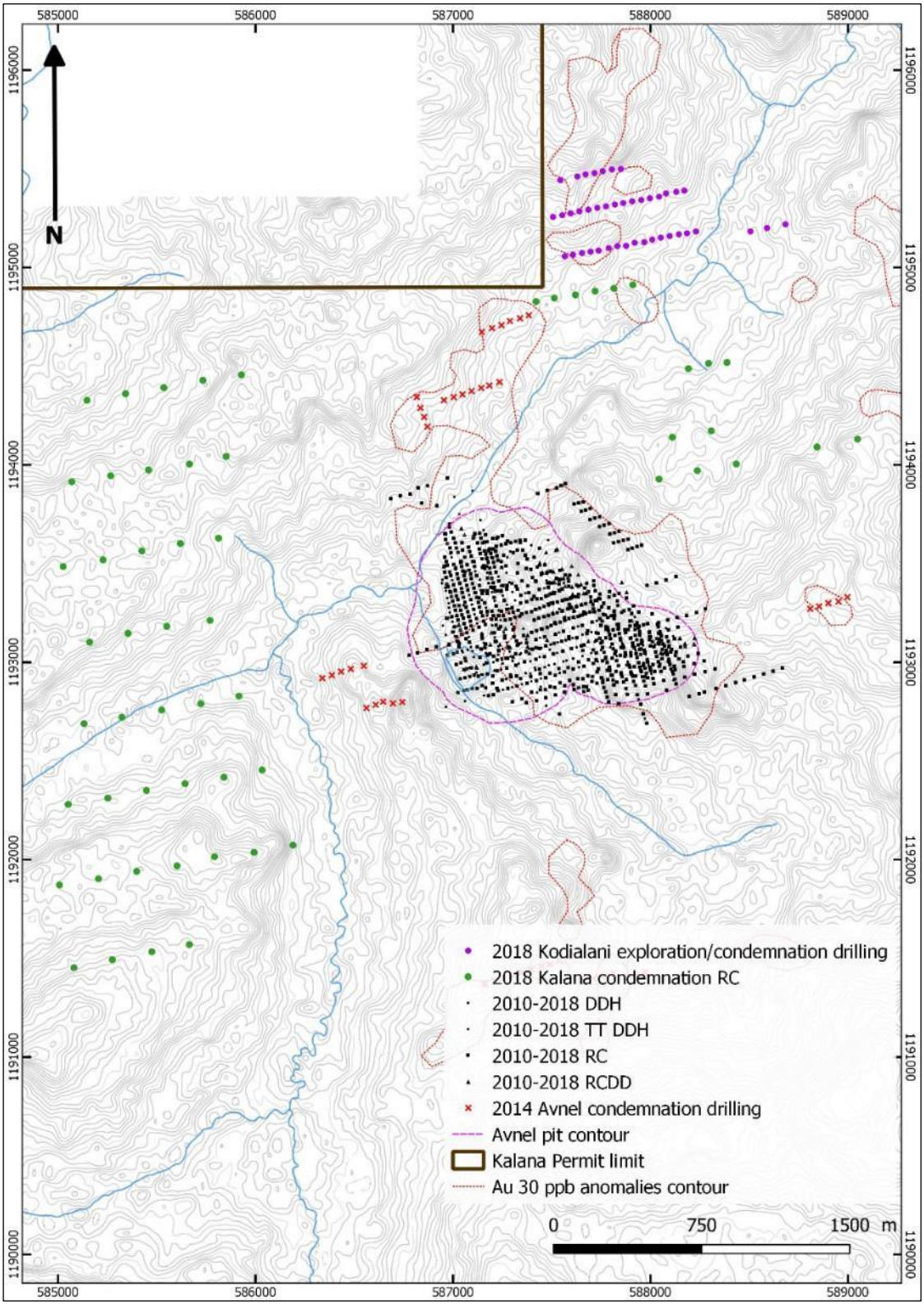
The programme led to the conclusion that the deposit has no continuity laterally or to the south; however, economic gold grades were encountered in the north showing a potential continuity (which is yet to be tested at the time of writing). The deep drilling in the northeast showed the extent of Veins 1 to 17 at depth. Three zones hosting 3rd Order Sub-vertical veins were defined in the north, but they could not be modelled as further very tight drilling is required to generated sufficient resolution. A change in drill hole orientation would also be required to facilitate intersecting the structures at favourable angles of intersection.

In addition to the resource development drilling, SOMIKA (Endeavour) completed a condemnation drilling programme around the Kalana deposit. It included 57 RC holes totalling 6,267 m in areas of proposed infrastructure construction zones. The drilling confirmed that the area was barren and suitable for infrastructure installation (Figure 10.4.2).

In addition, 42 exploration / condemnation RC holes were drilled on a potential camp location to the north of the Kalana Deposit which had some exploration potential due to the presence of artisanal miners activity and termite mound gold anomalies (Figure 10.4.2).

All the samples from the Kalana area were sent to ALS Ouagadougou to be assayed using the two kilogram LeachWELL 24-hour bottle roll method with fire assays on the tails.

Figure 10.4.2 2018 Avnel (Endeavour) Kalana Condemnation Drilling



10.4.2 Kalanako Deposit

SOMIKA (Endeavour) also completed in 2017-2018 an infill and exploration drilling campaign on the Kalanako deposit. Geodrill Ltd. drilled the 99 RC, one DD and two triple tube diamond drill holes (TTDD), Table 10.4-2 and Figure 10.4.3.

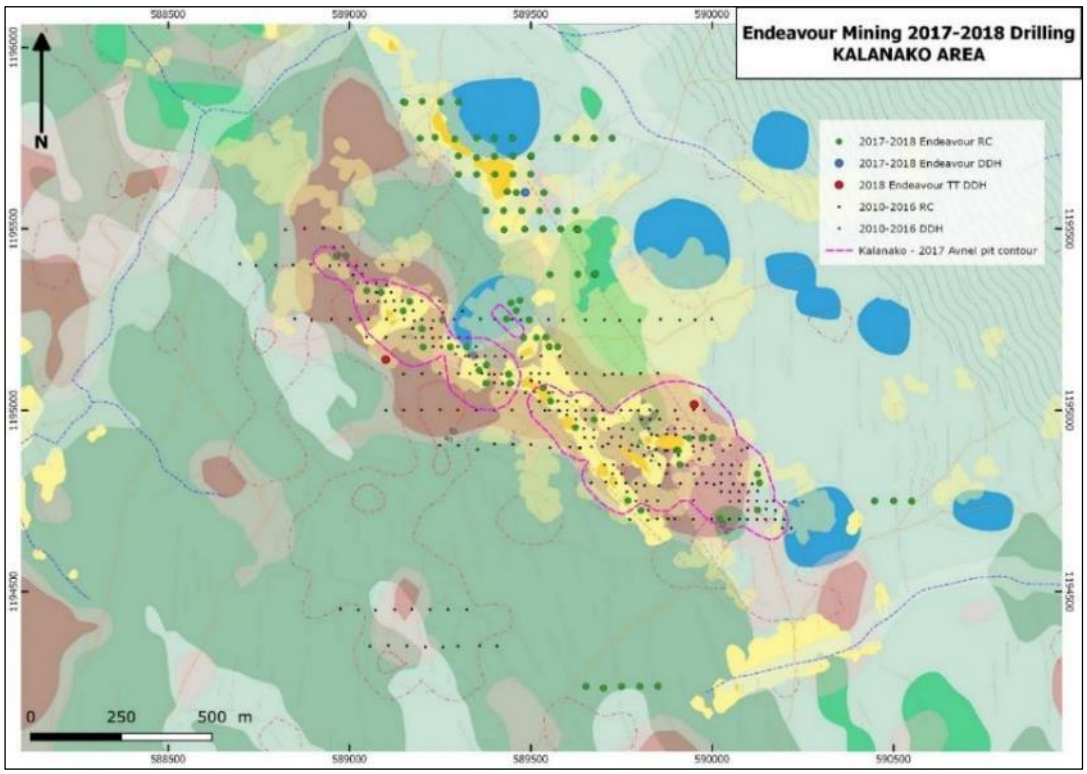
Table 10.4-2 SOMIKA (Endeavour) Kalanako Drilling Summary 2017 – 2018

Deposit	RC		DD		TTDD	
	#	m	#	m	#	m
Kalanako	99	12,765	1	189	2	350

The drilling had three objectives:

- Infill areas of the Kalanako main deposit to increase the confidence in the mineralisation continuity.
- Explore the northeastern extension of the deposit where massive artisanal mining collapses are found.
- Provide material for the geotechnical study of the Kalanako deposit.

Figure 10.4.3 2017-2018 Avnel (Endeavour) Kalanako Drilling

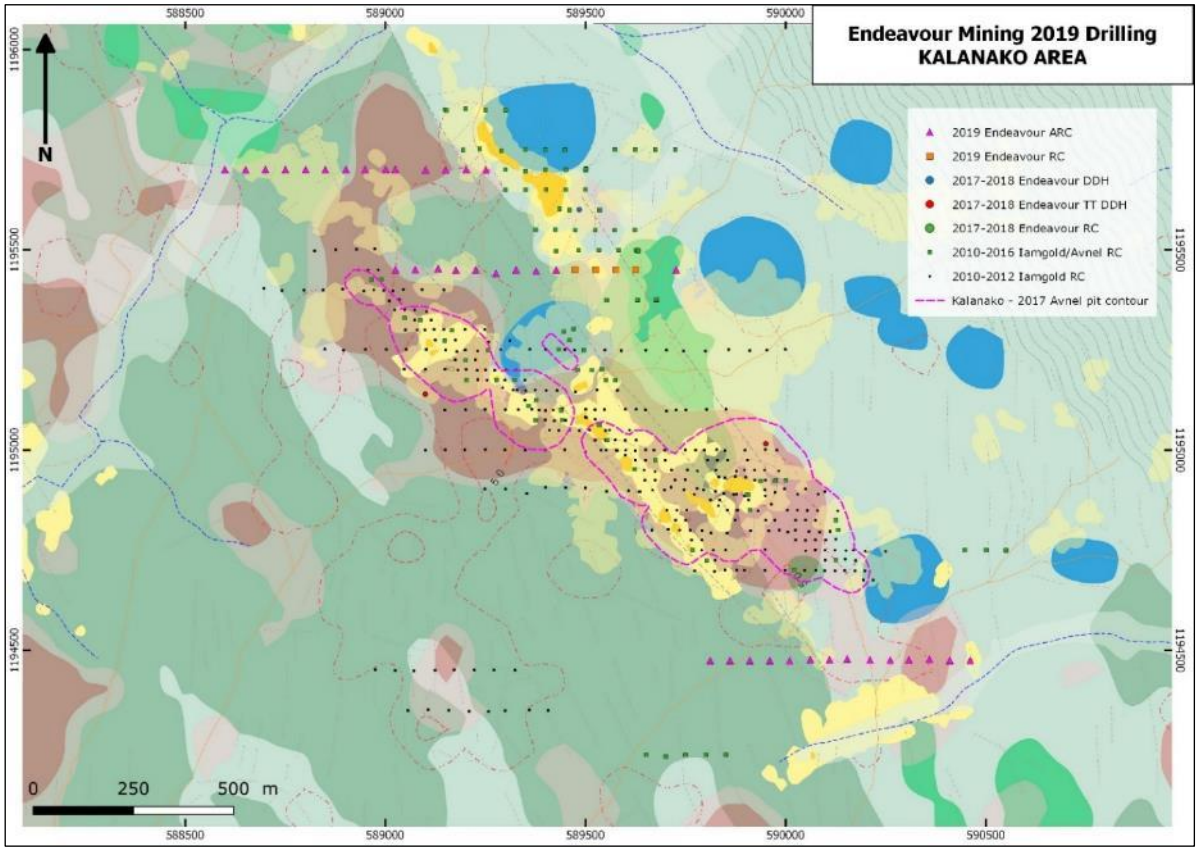


The drilling of the northeastern extension at Kalanako was undertaken in two phases; the first pass reconnaissance drilling on a 100 m by 50 m drillhole spacing demonstrated the potential of the area. The second pass, infill drilling refined the mineralisation outlines bringing the drill spacing to 50 m by 50 m. The second phase confirmed the mineralisation geometry and continuity; however, the results from the Kalanako northeastern extension have not been included in the Kalanako Mineral Resource Estimate Update presented in this Technical Report.

In 2019, following the positive results obtained from the 2017 - 2018 drill programme, SOMIKA (Endeavour) commissioned an extension drilling programme of the Kalanako deposit aiming at extending the known deposits to the south and check for a potential en-echelon extension for the Kalanako Northeast target. AMCO drilled 38 AC holes (3,300 m) and four RC holes (538 m) in May 2019 (Figure 10.4.4). AC drilling was initially planned but it was performed using an RC methodology with a hammer bit instead of blades. AC holes were drilled to a depth of 90 m, and RC used for the deeper holes.

The samples from the first phase of drilling in 2017 were sent to Bigs Global in Ouagadougou, Burkina Faso. The samples from the second phase of drilling in 2018 and from 2019 were sent to ALS Ouagadougou. The assay method at both laboratories was LeachWELL.

Figure 10.4.4 2019 SOMIKA (Endeavour) Kalanako Drilling



10.4.3 Other Targets

All the samples from the 2019 drilling campaign were assayed using the LeachWELL method by ALS Ouagadougou.

In 2019, the SOMIKA (Endeavour) started a regional drilling campaign to test identified targets which had not been drilled to date.

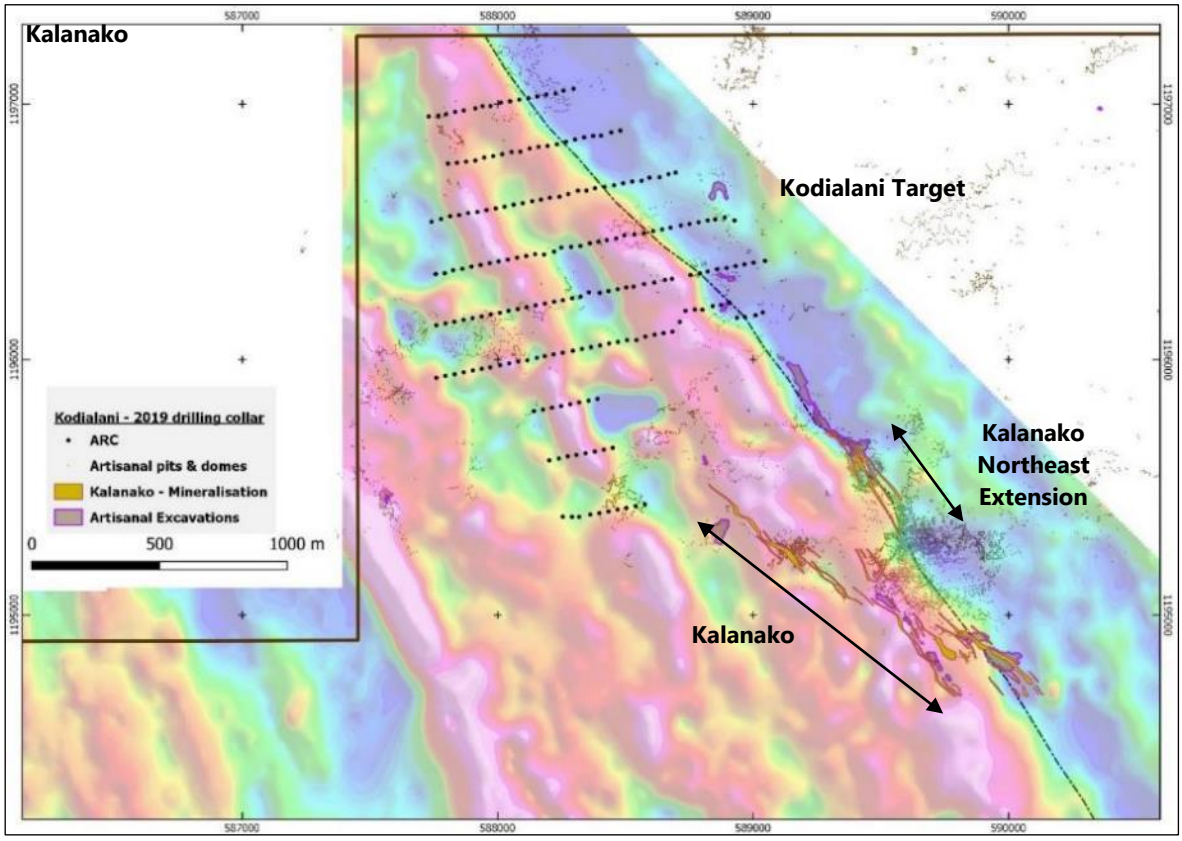
Kodialani Target

Kodialani lies along trend of the Kalanako structure and shows artisanal workings (Figure 10.4.5).

AMCO drilling completed 199 AC holes for a total of 12,044 m, with an average depth of 60 m. The drilling programme aimed at systematically testing the Kalanako structure to the north and also testing artisanal workings and termite mound geochemistry anomalies in the area. AC holes were drilled every 40 m along lines spaced 200 m apart.

The drilling demonstrated the presence of gold mineralisation along the Kalanako structure with drillholes returning gold intercepts on each drilling line.

Figure 10.4.5 2019 SOMIKA (Endeavour) Kodialani Drilling

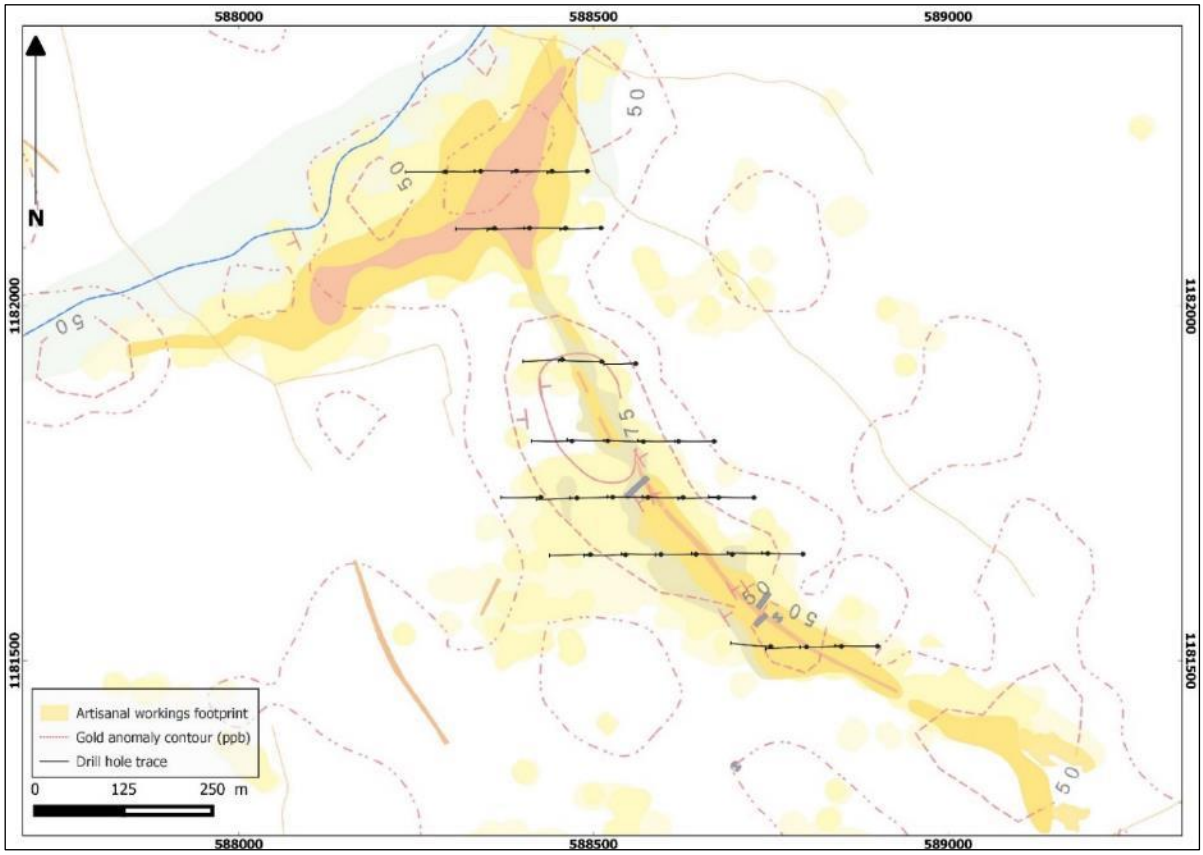


Solomani

Solomanina is located 12 km south of the Kalana town. The area is characterised by an old orpaillage that remains active and a strong gold anomaly outlined by the termite mound geochemistry. SONAREM-SOGEMORK carried exploration works on that area, ranking it in their top priority targets after Kalana and Kalanako in the 1970's. They also collected grab samples, some returning high grades, including 12 Au g/t, 8 Au g/ and 4.6 Au g/t, dug two trenches and drilled auger holes in the area.

SOMIKA (Endeavour) sampled the orpaillage tailings collecting quartz grab samples and pit rejects material in 2016 and 2017. Following the good results obtained, 35 RC holes (3,541 m) were drilled by AMCO along seven lines with a spacing varying from 200 m to 400 m (Figure 10.4.6).

Figure 10.4.6 2019 SOMIKA (Endeavour) Solomanina Drilling

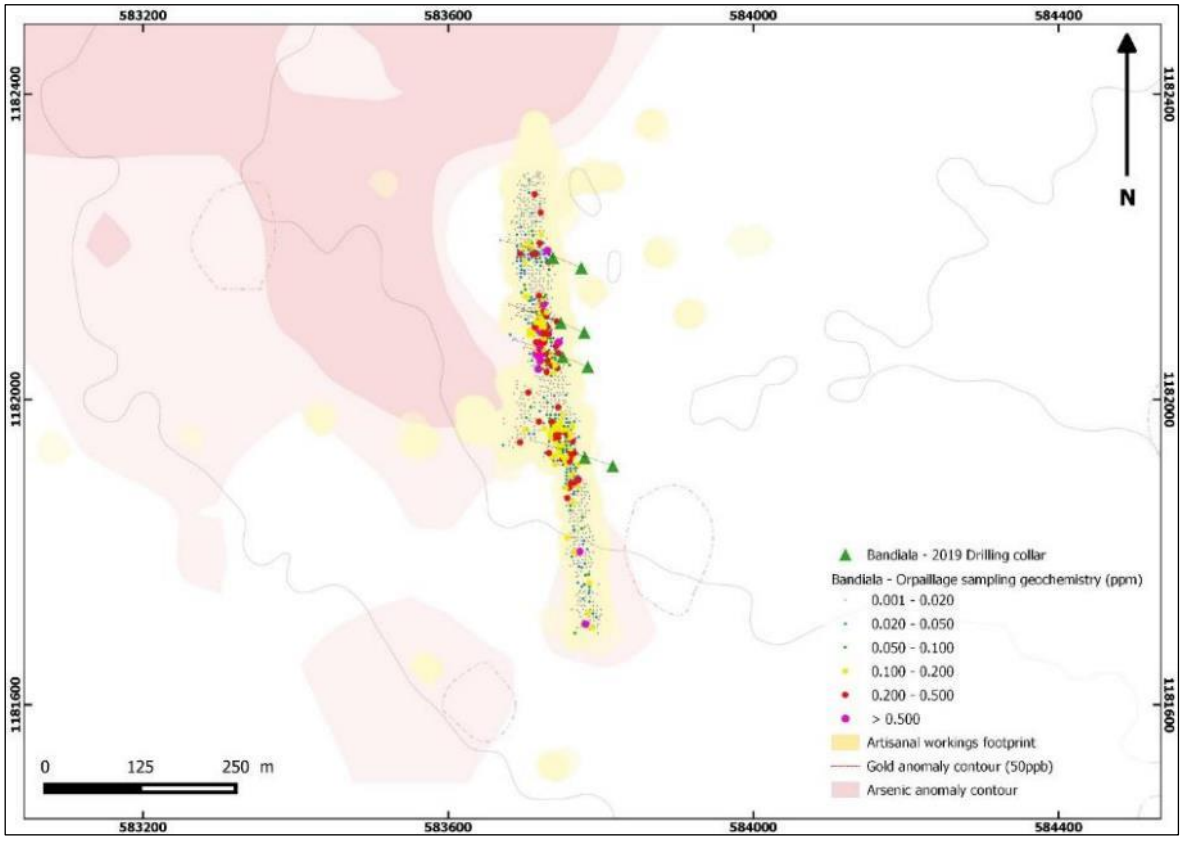


Bandiala

Bandiala is located 12 km southwest of Kalana town at the contact between a granite intrusion and the sediments covering the major part of the Kalana permit. A 700 m long, very old orpaillage sits near the contact between the granite and the sediments. A three metre thick north-south orientated quartz vein has been excavated by the artisanal miners.

In 2016, SOMIKA (Avnel) collected samples from artisanal tailings and grab samples. Results delineated three main mineralised zones that SOMIKA (Endeavour) drilled in 2019. Eight RC holes (1,105 m) were drilled by AMCO along four lines (Figure 10.4.7) across what appeared to be the mineralisation cores.

Figure 10.4.7 2019 SOMIKA (Endeavour) Bandiala Drilling



10.5 Drillhole Surveying

All drillhole collars are sited and checked using dGPS by qualified staff using company equipment.

Downhole survey deviations on DD and RC holes were measured using the Reflex EZ SHOTTM system by the driller contractor. The first measurement is usually taken at 12 m downhole to verify the inclination and orientation of the drillhole. Subsequent measurements are taken at 30 m downhole and then at 30 m intervals. The last measurement is taken at the bottom of the hole.

Orientation of the core was undertaken in fresh rock using the Reflex ACTII system with measurements taken every run.

11.0 SAMPLING PREPARATION, ANALYSES AND SECURITY

11.1 Introduction

The following section includes discussion and comment on the sample preparation and security related aspects relating to the collation of samples which inform the Kalana Project geological modelling and Mineral Resource Estimates.

The Kalana Exploration Yard lies within the perimeter fence of Kalana Mine and is separate to the mine infrastructure and residents' camp. It is a purpose-built facility with locked storage buildings and a covered core yard with a concrete base and lighting. Long-term core sheds and sample storage sheds are located within the perimeter fence.

All aspects of the collection, preparation and dispatch of drill samples are carried out by SOMIKA personnel. The sample collection and preparation, analytical techniques, security and quality assurance / quality control (QA/QC) protocols implemented are consistent with standard industry practice and are suitable for the reporting of exploration results and for use in Mineral Resource estimation. The sampling procedures are adequate for and consistent with the style of gold mineralisation under consideration. The procedures and protocols have evolved over time and have been followed by SOMIKA (Avnel) pre-mid-2017 and by SOMIKA (Endeavour) post-mid-2017. The following section does not differentiate between SOMIKA (Avnel) and SOMIKA (Endeavour); it does, however, distinguish between work completed pre- and post-2016 and the publication of the 2016 Feasibility Study Technical Report.

From 2004 until 2006, SOMIKA used Abilabs to test the exploration samples. The Abilab laboratory was not accredited and joined the ALS Laboratory Group in 2006.

Between 2009 and 2013, exploration samples generated during the Avnel-IAMGOLD joint venture were prepared by a SGS sample preparation facility at Kalana Mine and the samples were assayed by SGS Bamako, Mali and SGS Kaves, Mali. (Mine samples were prepared and assayed by the on-site SGS facility.)

SGS is a global independent provider of assaying and analytical testing services for the mining and mineral exploration industry with consistent quality standards implemented across all regions. Whilst they were not formally accredited, the SGS laboratories at Bamako and Kaves operated to the International Organisation for Standardisation (ISO) 17025:2005 standards. As part of the international group of SGS laboratories, all laboratories took part in regular Round Robin sample analysis to check for bias or systematic error. At SGS laboratories, samples were processed and analysed for gold content by 50 g fire assay, with high-grade samples assayed using a gravimetric finish.

Commencing in 2013, SOMIKA predominately used BIGS Global S.A.R.L. (BIGS Global) in Ouagadougou, Burkina Faso for sample preparation and analysis using the LeachWELL method with a tail fire assay. BIGS Global is accredited to the ISO/International Electrotechnical Commission (ISO/IEC) 9001:2008 and ISO/IEC 17025:2005 for all relevant procedures and is independent of SOMIKA and Endeavour Mining.

The 2017-18 SOMIKA exploration programmes used BIGS Global, ALS Ouagadougou and ALS Kumasi, Ghana for sample preparation and assay using the LeachWELL method. The Kalanako samples were predominately prepared and assayed at BIGS Global whilst the Kalana samples were prepared and assayed by ALS in Ouagadougou with a small portion in Kumasi.

11.2 Historical Samples

The following is a summary of the sample preparation undertaken on the Kalana Project samples pre-2016. Further details can be found in the 2016 Technical Report.

11.2.1 Historical Sample Preparation

Pit, Rock, Soil and Termite Mound Samples, SOMIKA 2003 to 2004

Soil and termite mound samples were transferred from plastic bags to stainless steel trays and dried in the sun at the Kalana Exploration Yard. The dry samples were then:

- Sieved through -80 mesh (180 µm) stainless steel screens and the fine fraction mixed in a bowl.
- A subsample of approximately 150 g was collected with a scoop and transferred into a plastic bag.
- The bagged samples were packed into cardboard boxes in batches of 1,000 for transport by road accompanied by a geological assistant to the Abilab laboratory in Bamako.

The stainless-steel screens were cleaned using ultrasonic cleaners and compressed air. The reject fine fractions were securely stored on-site.

The Phase 2 sample pulps taken October to December 2004 were reduced to 100 g in size.

Pit and rock samples were crushed and milled to 90% passing 200 µm at the Abilab laboratory.

Termite Mound Samples, IAMGOLD 2009 to 2011

Termite mound samples were transferred from plastic bags to stainless steel trays and dried in the sun. The two to 2.5 kg sample was taken to SGS Kalana Exploration, where it was weighed by SGS personnel and recorded. The entire sample was pulverised to a nominal 85% passing 75 µm. A subsample of nominally 200 g was taken and placed in a Kraft paper bag for transport to the analytical laboratory.

Underground Sampling, IAMGOLD 2009 to 2012

Underground samples between four and eight kilograms were taken to SGS Bamako where they were weighed. The entire sample was crushed to two millimetres and a two-kilogram subsample was pulverised to a nominal 85% passing 75 µm. A further subsample of nominally 200 g was taken and placed in a Kraft paper bag for transport to SGS Kayes. (From October 2010, the 200 g subsamples were collected by riffle splitting to avoid possible segregation of heavy gold particles after pulverisation.)

Drill Samples, SOMIKA 2004 to 2008

Drill samples were taken on-site but prepared off-site as summarised below in Table 11.2-1.

Table 11.2-1 Summary of Historical Drill Sample Preparation 2004 to 2008

Programme	Sample Type and Size	On-site Sample Preparation	Off-site Sample Preparation
Phase 3 - Q2 June 2005	RC Chips. 2 kg composite taken over 2 m interval taken by a "Controlab Splitter"	Drying	Abilab, Bamako Dried, crushed and pulverised
Phase 4 – Nov 05 to May 06	Half Core - HQ3 and HQ	-	Abilab, Bamako Dried and crushed. Two kilos pulverised

Drill Samples, IAMGOLD 2009 to 2013

Preliminary sample preparation (DD core sawing and RC sample initial splitting) was carried out by the IAMGOLD exploration team at Kalana. Once sawed or split, samples were dispatched to SGS Kalana Exploration (RC samples) or SGS Bamako (half core) for analytical sample preparation.

Table 11.2-2 Summary of Historical Drill Sample Preparation 2009 to 2013

Programme	Sample Type	Sample Sze	Sample Preparation
2009 - 13	RC	4 – 5 kg	Riffled to 2.5 kg subsample on-site. At lab, total sample pulverised to 85% passing 75 µm and cone split to 200 g.
2009 - 13	DD	1 m interval Half core	Dried, crushed, pulverised to 85% passing 75 µm and split by cone splitter for 200 g subsample.

Re-Assay Programme, SOMIKA (Avnel) 2013-15

IAMGOLD and SOMIKA (Avnel) identified the issue of sample representativity with regard to gold grades from the 50 g fire assays as the sampling protocol was considered not optimal for coarse gold mineralisation and likely to under-estimate the gold grade. Subsequently, in 2013, Avnel explored the opportunity of re-assaying the existing DD and RC samples using refined sampling protocols and larger sub-samples in association with Dr Simon Dominy (an expert on coarse gold in narrow veins who was at the time with Snowden Consultants).

Options for improved protocol included the use of screen fire assays (SFA), LeachWELL cyanide assays and laboratory scale gravity processing of samples. The LeachWELL two kilogram 24-hour bottle roll with fire assay on the tails protocol was selected. Further details can be found in the 2016 Feasibility Study Technical Report. This re-assay programme and switch in the assay method was a game changer for the Project.

Between 2013 and 2015, a total of 31,100 Kalana samples were re-assayed using the LeachWELL protocol. Table 11.2-3 summarises the re-assay test work and the increase in grade seen. In 2015, 2,267 RC and DD samples from Kalanako were re-assayed using LeachWELL which had a similar grade lift as seen at Kalana.

Table 11.2-3 Summary of Kalana Re-Assayed Samples (as of March 2015)

Grade Greater than (Au g/t)	No. IAMGOLD Samples	No. Samples Re-assayed	Proportion Re-Assayed (%)	Grade Upgrade Total (%)
0.0	130,291	31,100	23.9	35
0.5	11,930	10,856	91.0	45
0.9	7,560	6,926	91.6	50

The remaining 1,074 Kalana samples with a grade greater than 0.5 Au g/t that were not re-assayed correspond to either isolated grades with no structural continuity or, more often, samples for which there is no material available for a two kilogram bulk assay.

Drill Samples, SOMIKA 2015

Table 11.2-4 summarises the drill sample preparation methodology used by SOMIKA at the Kalana Exploration Yard after the termination of the IAMGOLD joint venture.

Table 11.2-4 2015 Drill Sample Preparation Summary

Programme	Sample Type	Sample Size	Sample Preparation
SOMIKA 2015	RC	4 – 5 kg	Total sample crushed. 2 kg rotary split and pulverised.
	DD	1 m half core	Total sample crushed. 2 kg rotary split and pulverised.

11.2.2 Historical Analysis

Table 11.2-5 summarises the analytical methods and laboratories historically used for the Kalana Project. Further details can be found in the 2016 Technical Report.

Table 11.2-5 Historical Analytical Methods and Laboratory Summary

Programme	Sample Type	Laboratory	Analytical Method
SOMIKA 2002-04	Soil and termite mound	Abilab, Bamako	30 g FA w AAS finish
	Soil	Assayers Canada Ltd (Vancouver)	35 element ICP-AES
	Pit and rock	Abilab, Bamako	30 g FAA w AAS finish
SOMIKA 2005 & 06	RC Chips and DD Core	Abilab, Bamako	50 g FA w AAS finish
IAMGOLD 2009-11	Termite mound	SGS Kayes	50 g FA w AAS finish
	Underground Aug-Dec 09	ALS	50 g FA
	Underground	IAMGOLD at Kalana	XRF
	Underground Jan 10 – Dec 12	SGS Kayes	50 g FA w AAS finish
IAMGOLD 2009 - 13	RC and DD Core	SGS Kayes	50 g Fire Assay
SOMIKA 2013 - 14	Re-Assay RC and DD	BIGS Global	Cyanide leached (LeachWell) and 50 g tails FA
SOMIKA 2015	RC and DD Core	BIGS Global	Cyanide leached (LeachWell) and 50 g tails FA (if > 0.3 ppm Au)

It is unknown what accreditation Assayers Canada Limited (Assayers) laboratory in Vancouver, Canada held in 2004. It has been part of SGS Laboratory Group since 2010.

The Abilab laboratory was not accredited under recognised accreditation and joined the ALS Laboratory Group in 2006.

Historically, SGS Kayes did not have recognised accreditation for analytical testing.

Between 2009 and 2013, 50 g fire assays were performed by SGS Bamako. Real-time monitoring and laboratory visits were performed by internal and external qualified persons.

After 2013, LeachWELL and fire assay on tails were performed by BIGS Global in Ouagadougou, Burkina Faso. Real-time monitoring and laboratory visits were also performed by internal and external qualified persons.

The re-assay programme in 2013-2015 utilising LeachWELL included a QA/QC programme, as summarised in Section 11.9.

11.3 Sampling

Sampling and initial sample preparation is undertaken by SOMIKA staff based at the Kalana Exploration Yard.

Advanced geochemical (artisanal tailings / rejects and rock / grab) samples are collected in the field. The orpaillage (artisanal) tailings reject pile is cleaned of dust and a transversal trench is dug across the pile with samples taken all along the trench.

The entire RC sample (taken in one-meter intervals) is collected at the drill rig cyclone and taken to the Exploration Yard. RC samples are air dried in large, shallow metal pans by sunlight (and oven) if necessary. The samples are then placed in large plastic bowl and split using a two-part Riffle Jones Splitter in order to obtain a four- to five-kilogram sub-sample. (A second sample approximately six kilograms in weight is also taken for reference and stored in labelled bags of seven on-site.)

DD core is placed in labelled, metal core trays with depth markers and driller markings at the drill rig by SOMIKA staff. The core is marked with an orientation line by the supervising SOMIKA Team Leader. The covered core trays are transported to the Exploration Yard by a SOMIKA exploration driver at the end of each shift.

All diamond core and RC chips sample are systematically logged at the Kalana Exploration Yard. RC chips are logged for weathering and lithology. A portion of the RC reject is washed and panned for visible gold, see Figure 11.3.1. Washed chip trays and chip boards are generated for future reference, see Figure 11.3.2.

Figure 11.3.1 Panning for Visible Gold



Figure 11.3.2 RC Chip Boards and Chip Trays



DD core is logged for weathering, lithology, mineralisation, structure, alteration and geotechnical information.

DD samples are taken in one metre increments (note, intervals are not adjusted by lithological contacts). DD cores in soft material (i.e. saprolite) are cut using a knife, core in hard material (i.e. saprock and fresh rock) are sawn in half using a diamond saw. Half core is systematically sampled. Oriented samples are sawn along the orientation line. The retained half core is kept on site for future reference in designated covered storage sheds with the mine perimeter fence.

The RC and DD samples are placed in labelled clear plastic bags with a paper sample ticket and a metal sample tag inserted, sealed by metal staples.

The sample equipment and bowls are cleaned by water and/or compressed air between samples.

Photos of core boxes and chips boards are systematically digitally captured.

11.4 Sample Submission

RC and DD samples are individually bagged and sealed in larger, labelled rice bags in groups of ten and stored in a locked building. The sealed and labelled bags containing multiple samples are collected from site by the analytical laboratory under the management of SOMIKA staff.

SOMIKA inserts RC field (coarse) duplicates, blanks and Certified Reference Materials (CRMS or "standards") into the RC sample stream, and blanks and CRMS into the DD sample stream prior to dispatch.

11.5 Sample Preparation

Geochemical samples are crushed and milled to 90% passing 200 µm at the analytical laboratory.

The RC and DD samples are dried, crushed and pulverised at the analytical laboratory. The entire 2.5 kg to 5 kg RC sample is crushed to less than 2 mm and split (by rotary splitter) to obtain a 2 kg sub-sample which is pulverised to a nominal 85% passing 75 µm.

11.6 Analysis

The Kalana Project geochemical, RC and DD samples were analysed by the two kilogram LeachWELL 24-hour bottle roll with fire assay on the tails method at BIGS Global and ALS Ouagadougou in Burkina Faso and ALS Kumasi in Ghana.

The 2017 Kalanako samples were analysed at BIGS Global. The majority of the 2017-18 Kalana samples were analysed by ALS Ouagadougou with a small portion were analysed by ALS Kumasi. The 2018 condemnation drilling samples were analysed by SGS Bamako.

The laboratories inserted their own QA/QC samples (blanks and CRMs) into the sample stream after every nine samples.

11.7 Density

Over 14,300 volumetric mass density measurements to determine density on Kalana and Kalanako diamond drill cores were taken by IAMGOLD and SOMIKA between 2010 and 2018 using a standard weight-in-air and weight-in-water technique (the Archimedes Principal), Table 11.7-1. The samples are selected by geological technicians in the core shack and measured in the sample preparation laboratory on-site. Samples comprise a 200 g to 500 g piece of half core taken approximately every five metres downhole. Volumetric mass density measurements are taken on all rock types and states of weathering. Saprolite samples are wrapped in thin plastic ("cling film").

Table 11.7-1 Density Measurements Taken by Deposit and Programme

Deposit	Programme	No. Measurements
Kalana	2010-12 IAMGOLD/SOMIKA	9,475
	2015 SOMIKA	1,414
	2017-18 SOMIKA	2,176
Kalanako	2010-12 IAMGOLD/SOMIKA	1,265
Total		**14,302**

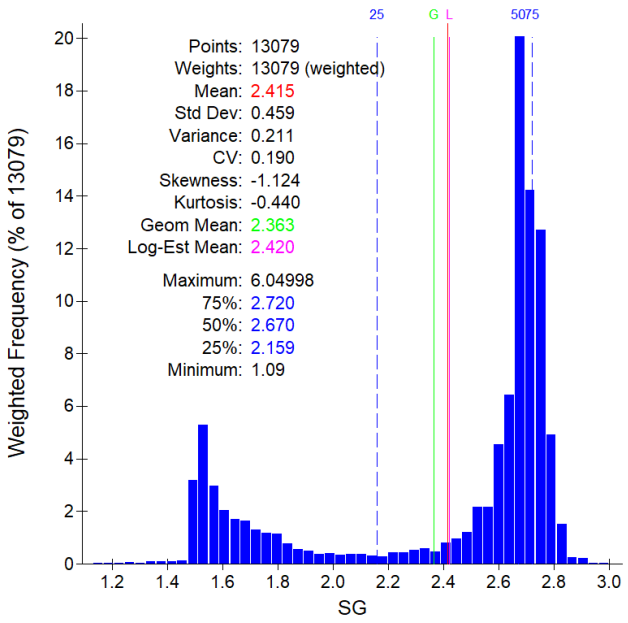
At Kalana, lithology-based density contrasts are very small. For each weathering subdivision, length-weighted averages are calculated after rare outlier values are removed. The saprock exhibits a greater degree of variability than the other weathering categories, Figure 11.7.1; this is caused by the density values within this category retaining some bimodality in their distribution which is consistent with the mixed characteristics of this domain. At Kalana, the saprolite has an average density of 1.64 kg/m^3, the saprock 2.15 kg/m^3 and the fresh rock 2.68 kg/m^3, see Table 11.7-2.

Table 11.7-2 Kalana Density Measurements by Weathering Type (Excluding Outlier Values)

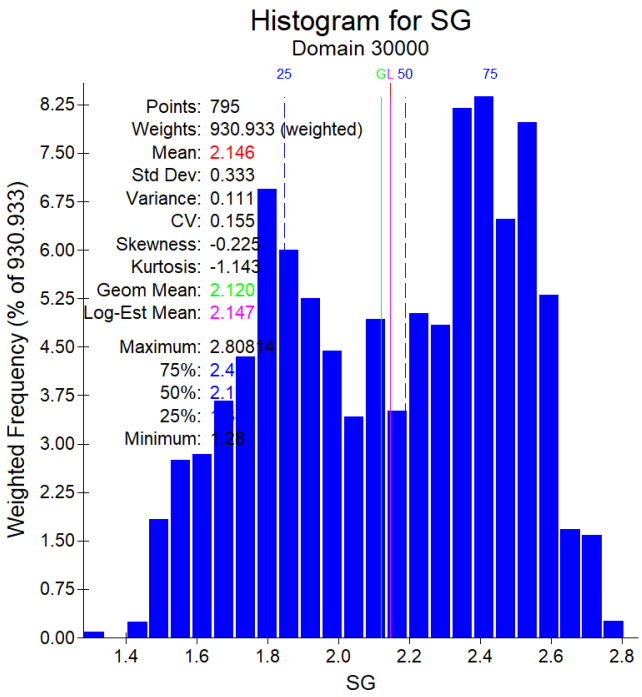
	Regolith	Saprolite	Saprock	Fresh Rock
No. Samples	198	1,833	795	10,107
Minimum Density (kg/m^3)	1.40	1.50	1.28	2.00
Maximum Density (kg/m^3)	2.17	2.60	2.81	2.99
Mean Density (kg/m^3)	**1.67**	**1.64**	**2.15**	**2.68**
Standard Deviation	0.14	0.17	0.33	0.09
CV	0.08	0.10	0.16	0.03

Figure 11.7.1 Kalana Density Data

All Data



Saprock Data



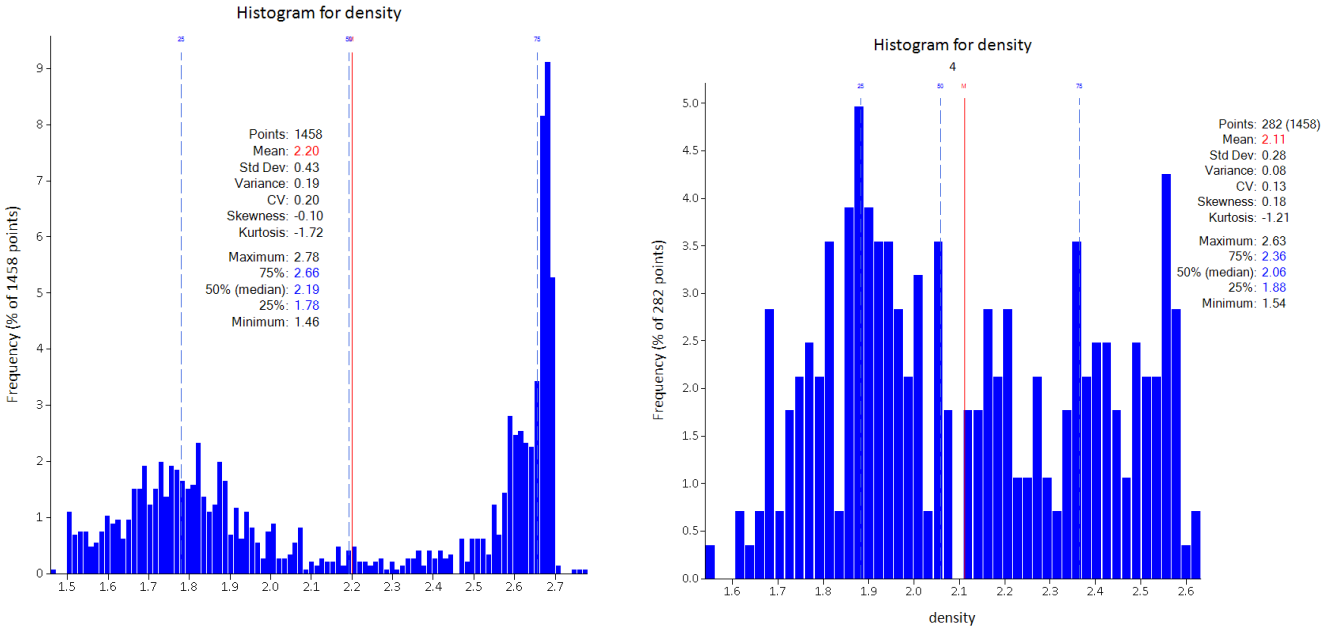
The Kalanako density data exhibits a distinct bimodality, the most common lower density is around 1.8 kg/m^3 and the most common higher density is around 2.68 kg/m^3 (Figure 11.7.2). The least frequent density values are in the range 2.1 kg/m^3 to 2.5 kg/m^3. Some of the Kalanako density readings straddled the interpreted weathering surfaces and were subsequently double accounted; hence the resource estimation has some 120 more density values, see Table 11.7-3. At Kalanako, the saprolite has an average density of 1.76 kg/m^3, the saprock 2.11 kg/m^3 and the fresh rock 2.64 kg/m^3.

Table 11.7-3 Kalanako Density Measurements by Weathering Type (Excluding Outlier Values)

	Laterite	Mottled Zone	Saprolite	Saprock	Fresh Rock
No. Samples	19	20	520	282	617
Minimum Density (kg/m^3)	1.51	1.51	1.46	1.54	1.96
Maximum Density (kg/m^3)	1.96	1.94	2.29	2.63	2.78
Mean Density (kg/m^3)	**1.71**	**1.72**	**1.76**	**2.11**	**2.64**
Standard Deviation	0.15	0.14	0.14	0.28	0.06
CV	0.09	0.08	0.08	0.13	0.02

Figure 11.7.2 Kalanako Density Data

All Data ***Saprock Data***



11.8 Chain of Custody and Security

All diamond core and RC samples are transported to SOMIKA's Exploration Yard within the Kalana Mine perimeter fence. All diamond core and RC samples have QA/QC samples inserted into the sample stream. Sample batches are placed in sealed and numbered polyweave and plastic bags for transport. The samples are collected in Kalana by the laboratory truck. Upon receipt, the laboratory personnel sign for the secure receipt of all the samples.

All aspects of the sample collection and dispatch is conducted by SOMIKA personnel, or under the supervision of SOMIKA staff.

A Chain-of-Custody documentation chain is compiled and managed throughout the process.

11.9 Quality Assurance / Quality Control

Quality assurance (QA) consists of evidence to demonstrate that assay data has precision and accuracy within generally accepted limits for the sampling and analytical method(s) used in order to have confidence in resource estimations. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of sampling, preparing and assaying exploration samples.

In general, QA/QC programs are designed to prevent or detect contamination and allow analytical precision and accuracy to be quantified. In addition, a QA/QC programme can identify the overall sampling and assaying variability of the sampling method itself. The programme can also determine the reporting accuracy for clerical and data transfer errors.

Accuracy is assessed by reviewing assays of commercially available Certified Reference Material (CRM) or in-house standards, and by check assaying at outside accredited laboratories (referee, umpire, or check samples). Precision is assessed by processing duplicate samples from each stage of the analytical process from the primary stage of sample splitting, through sample preparation stages of crushing/splitting, pulverising / splitting, and assaying. Control samples can also help identify possible mix-ups or mislabels during sample preparation.

In addition, the BIGS Global and ALS laboratories have their own internal quality performance processes which follow best practice guidelines required for qualification under International Organisation for Standardisation ("ISO") standards. The standard QA/QC protocols for the laboratories include the insertion of CRMs, blanks, duplicates and repeat assaying to monitor the quality of the preparation and analytical processes of the laboratory.

11.9.1 Historical QA/QC

The evolution and details of the QA/QC programme at the Kalana Project can be found in the 2016 Technical Report. The following is a synopsis thereof:

- SONAREM-SOGEMORK (1967 to 1991) – no QA/QC information available.
- Ashanti – JCI (1995 to 1996) – No details but scanned certificates available.
- SOMIKA (2003 to 2009) – Implemented QA/QC programmes for soil samples and RAB, RC and DD programmes.
- IAMGOLD (2009 to 2013) – Implemented external analytical control measures on all samples. SRK (2013) reported that they reviewed written field procedures and analytical quality control measures used by IAMGOLD. Their opinion was "IAMGOLD personnel used care in the application and management of field and assaying explorations data. Furthermore, sample preparation, sample security and analytical procedures used by IAMGOLD are consistent with generally accepted industry best practices and were, therefore, adequate for the purpose of mineral resource estimation."
- SOMIKA (2013 to 2015) – All LeachWELL samples (RC and DD) were subject to industry standard procedures for QA/QC. The re-assay programme also included QA/QC samples.

Denny Jones as the MRE QP for the 2016 Technical Report concluded "… reviewed the written field procedures and analytical quality control measures used by Avnel. Denny Jones is of the opinion that Avnel personnel have used care in the collection and management of field and assaying exploration data. Furthermore, samples preparation, sample security and analytical procedures used by Avnel are consistent with generally acceptable industry best practices and the results are adequate for the purpose of mineral resource estimation. It is Denny Jones' opinion that the two-kilogram LeachWELL results are a more robust result than the 50 g fire assay."

11.9.2 Summary 2017/18 QA/QC Results

A number of QA/QC procedures were rigorously implemented to monitor the accuracy, precision and potential for contamination of the analytical and assay data received from all laboratories during the 2017/18 drilling programme at Kalana and Kalanako. The procedures are the same as implemented by SOMIKA prior to 2017. Each sample was assigned an individual sample number.

In addition to the QA/QC procedures put in place by the assay laboratory, a rigorous QA/QC routine was in place for all exploration completed by SOMIKA. The procedures in place include:

- Blanks - previously assayed material returning less than detection assay.
- Certified Reference Material (CRM) - Independently certified commercial material from Rocklabs of New Zealand.
- Field duplicates - A second RC sample collected in the field from the same source at the same interval.

The QA/QC procedures followed during the Kalana and Kalanako drilling programmes in 2017 and 2018 subscribe to industry standards with one blank sample and one CRM sample per 20 samples for RC and DD drilling and one field duplicate per 20 samples in RC drilling.

Due to the coarse nature of the gold in the Kalana and Kalanako deposits, a LeachWELL (LW) and Tails Fire Assay analysis regime is used. The results are reported by analysis type (i.e. LW and TFA) in parts per million (ppm) (which is equivalent to grams per tonne, g/t). The QA/QC for LW analysis is reported and evaluated separately from the QA/QC for the TFA analysis.

Quality control samples are assessed in a corporate standard QA/QC analysis report. This report is auto-generated within the Endeavour Corporate Database Management System when batches of new assay results are imported.

Sample batches with failed QA/QC samples are reviewed and are re-assayed or partially re-assayed where appropriate. Table 11.9-1 summarises the QA/QC programme for the 2017/18 drilling programme.

Table 11.9-1 QA/QC Result Summary, 2017/18 Programme

Blanks						
	LeachWELL			Tails Fire Assay		
	Number	Failures	Failure Percent	Number	Failures	Failure Percent
Kalana	2,567	1	0.04%	317	0	0.00%
Kalanako	373	0	0.00%	49	0	0.00%

CRMs						
	LeachWELL			Tails Fire Assay		
	Number	Failures	Failure Percent	Number	Failures	Failure Percent
Kalana	2,566	185	7.21%	443	44	9.93%
Kalanako	373	21	5.63%	48	6	12.50%

Field Duplicates						
	LeachWELL			Tails Fire Assay		
	Number	Failures	Failure Percent	Number	Failures	Failure Percent
Kalana	1,989	18	0.90%	257	3	1.17%
Kalanako	371	0	0.00%	37	4	10.81%

11.9.3 Blanks

The regular submission of blank material is used to assess contamination during sample preparation and to identify sample numbering errors. The SOMIKA QA/QC protocol stipulates for blanks to be inserted in the sample stream at a rate of at least one in 20 samples.

For the 2017/18 deposit definition programmes, the blank material was coarse crushed sandstone sourced from a quarry in Bamako, Mali. This material was certified blank after analysis at several laboratories in Mali.

A blank assay is considered a failure by Endeavour if it returns a value greater than ten times the detection limit of the assay method.

For the Kalana deposit, SOMIKA inserted over 2,500 blanks for LeachWELL (LW) analysis and over 317 blanks for the Tails Fire Assay (TFA) analysis.

The blanks analysis was excellent with only one failure encountered in LW analysis and none in the TFA analysis. Table 11.9-2 summarises the blanks used in the 2017/18 drilling programme.

Table 11.9-2 Summary of Kalana Blank Statistics for 2017/18 Drilling Programmes

	Kalana LW	Kalana TFA
Number of Blanks	2,567	317
Blanks > Detection Limit (0.005 ppm)	712	133
Max Blank Grade (ppm Au)	0.055	0.04
Blank Failures (>0.05 ppm Au)	1	0
Blank Failures Percentage	0.04%	0.00%

For the Kalanako area, SOMIKA inserted 373 blanks for LW analysis and 49 blanks for the TFA analysis. The blanks analysis was excellent with no failures encountered from either LW or TFA analysis as presented in Table 11.9-3.

Table 11.9-3 Summary of Kalanako Blank Statistics for 2017/18 Drilling Programme

	Kalanako LW	Kalanako TFA
Number of Blanks	373	49
Blanks > Detection Limit (0.005 ppm)	28	8
Max Blank Grade (ppm Au)	0.004	0.02
Blank Failures (>0.05 ppm Au)	0	0
Blank Failures Percentage	0.00%	0.00%

11.9.4 Duplicates

Duplicate samples can be field (core, trench or underground), coarse (crushed reject), or pulp (pulverised reject) duplicates. The type of duplicate can assess the natural local-scale grade variation or nugget effect (i.e., inherent grade variation) at the different stage of sampling, sample preparation and analysis.

Field duplicates assess the variability introduced by sampling the same interval and detecting sample number mix-ups. The duplicate splits are bagged separately with different sample numbers in order to be blind to the sample preparation laboratory. The duplicates contain all levels of error and are used to calculate field, preparation, and analytical precision. They are also a check on possible sample over selection, that is, the sampler has either purposely or inadvertently sampled the geological material (i.e., drill core) so as to preferentially place visible mineralisation in the sample bag going for analysis.

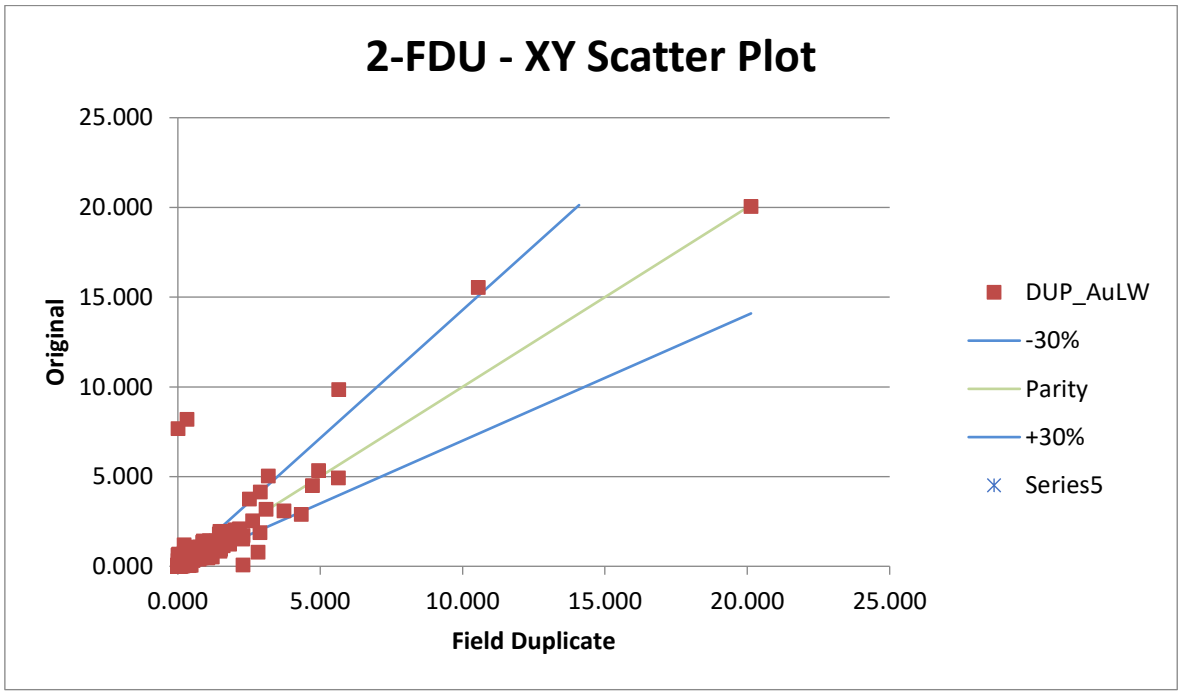
Endeavour applies several procedures for assessing the precision of field duplicates. A set of calculations have been integrated into the corporate standard QA/QC report based on an approach used by some assay laboratories. A tolerance value can be calculated for each individual duplicate pair based on the mean grade of the pair, the lower limit of detection for the analytical method used, and the method precision, as determined by the laboratory. The absolute relative difference is then calculated for each individual pair and if the result exceeds the calculated tolerance the duplicate pair is considered to have failed, as demonstrated in Table 11.9-4.

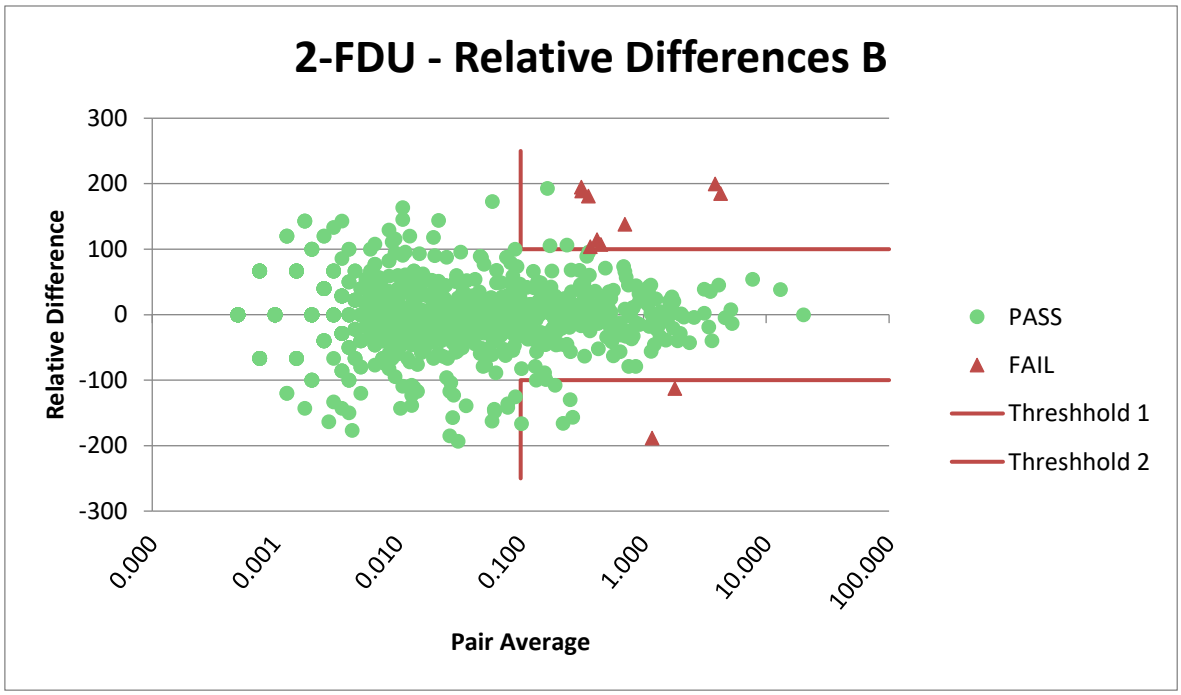
Table 11.9-4 Example of Field Duplicate Analysis Report

ORIG_SampleID	ORIG_AuLW	DUP_SampleID	DUP_AuLW	PairAvg	RelDiff	pctTolerance	pctDifference	Pass_Fail
KA-RC121039	0.001	KA-RC121040	0.001	0.001	0	5080	0	PASS
KA-RC121059	0.005	KA-RC121060	0.014	0.01	-0.9	343.2	94.7	PASS
KA-RC121079	0.001	KA-RC121080	0.001	0.001	0	5080	0	PASS
KA-RC121099	0.022	KA-RC121100	0.034	0.028	-0.4	169.3	42.9	PASS
KA-RC121119	0.532	KA-RC121120	0.488	0.51	0.1	84.9	8.6	PASS
KA-RC121139	1.287	KA-RC121140	0.895	1.091	0.4	82.3	35.9	PASS
KA-RC121159	0.407	KA-RC121160	0.247	0.327	0.5	87.6	48.9	PASS
KA-RC121179	0.499	KA-RC121180	0.191	0.345	0.9	87.2	89.3	FAIL
KA-RC121199	1.064	KA-RC121200	0.679	0.872	0.4	82.9	44.2	PASS
KA-RC121219	0.082	KA-RC121220	0.114	0.098	-0.3	105.5	32.7	PASS
KA-RC121239	1.136	KA-RC121240	1.596	1.366	-0.3	81.8	33.7	PASS
KA-RC121259	0.004	KA-RC121260	0.004	0.004	0	705	0	PASS
KA-RC121279	0.265	KA-RC121280	0.082	0.174	1.1	94.4	105.5	FAIL
KA-RC121299	0.024	KA-RC121300	0.014	0.019	0.5	211.6	52.6	PASS
KA-RC121319	0.009	KA-RC121320	0.009	0.009	0	357.8	0	PASS
KA-RC121359	0.417	KA-RC121360	0.588	0.503	-0.3	85	34	PASS
KA-RC121379	0.001	KA-RC121380	0.003	0.002	-1	1330	100	PASS
KA-RC121399	0.561	KA-RC121400	0.177	0.369	1	86.8	104.1	FAIL
KA-RC121419	0.166	KA-RC121420	0.112	0.139	0.4	98	38.8	PASS
KA-RC121539	0.477	KA-RC121540	0.331	0.404	0.4	86.2	36.1	PASS
KA-RC121559	0.093	KA-RC121560	0.136	0.115	-0.4	101.8	37.6	PASS

In addition, graphical evaluation of field duplicate performance is depicted in XY scatter plots and relative difference against mean grade plots, see Figure 11.9.1 for examples.

Figure 11.9.1 RC Field Duplicate Results Example





The coarse nature of the gold and the tendency of the gold to concentrate in distinct clusters at Kalana and Kalanako has resulted in very high rates of failure in duplicates in DD samples in the past. It was difficult to assess if these were true or natural failures. Therefore, the team stopped generating coarse duplicates in diamond core commencing in 2010 when IAMGOLD was on-site. For the 2017/18 drilling programme, they continued to follow this policy and coarse duplicates were not generated in diamond drill core samples.

For the 2017/18 Kalana drilling programme, coarse duplicates were only generated and inserted in the sample stream for RC drill samples.

The performance of the RC coarse duplicates during the 2017/18 drilling programme at Kalana were well within acceptable limits with 43% of the data having precision worse than 30%. No significant bias is present in the duplicate assays, as summarised in Table 11.9-5.

Table 11.9-5 Summary of 2017/18 Kalana RC Field Duplicate Statistics

	Kalana LW	Kalana TFA
No. Duplicates	1,989	257
No. Duplicate Pairs > D.L.	1,907	255

	Original	Duplicate	Original	Duplicate
Mean Grade	0.21	0.12	0.19	0.11
Minimum Grade	0.00	0.01	0.00	0.01
Maximum Grade	30.89	3.45	25.31	1.61
Median Grade	0.01	0.05	0.01	0.05
Variance	1.49	0.10	1.22	0.04
Standard Deviation	1.22	0.32	1.11	0.20
Coefficient of Variation	5.83	2.60	5.68	1.78
Correlation Coefficient	0.95		0.72	
% Difference Between Means	42.5%		49.7%	

11.9.5 Certified Reference Materials

Results of the regular submission of Certified Reference Material (CRM's) are used to identify problems of accuracy with specific sample batches and long-term biases associated with the regular assay laboratory.

CRMs from Rocklabs were used exclusively for internal accuracy control during the 2017/18 drilling programme at Kalana and Kalanako. ALS-Chemex Ltd.'s internal laboratory QA/QC used Rocklab CRMs with a very narrow range at the commencement of the programme. Upon request, the laboratory introduced CRM's from other suppliers with a larger gold value range.

Specific pass / fail criteria are determined from the standard deviation provided for the CRMs. The conventional approach to setting acceptance limits is to use the mean assay (also known as the Certified Reference Value (CRV)) plus / minus two (±2) standard deviations as a warning limit and ±3 standard deviations as a failure limit; however, the CRMs used in the Kalana and Kalanako 2017/18 drilling programmes are certified for fire assay (FA) analysis, not LeachWELL (LW). The LW digestion process does not necessarily result in 100% liberation of gold in the sample in contrast to FA analysis. Therefore, a universally recognised lower threshold of ±4 standard deviations is applied for failure in CRM's analysed by LW. The ±3 standard deviations as a failure limit is maintained for the FA analysis performed on the LW tail material. Hence, results falling outside of the failure limit of ±3 standard deviations must be investigated to determine the source of the erratic result, either analytical or clerical. Endeavour considers an overall CRM failure rate of less than 2% to be acceptable.

Table 11.9-6 Summary of CRMs Used in the 2017/18 Drill Programmes at Kalana and Kalanako

Source	CRM Name	Certified Value	Certified Std Dev	+/- 3 SD
Rocklabs	Std-HISILK4	3.463	0.09	0.27
Rocklabs	Std-OxC129	0.205	0.007	0.021
Rocklabs	Std-OXC145	0.212	0.007	0.021
Rocklabs	Std-OxD128	0.424	0.011	0.033
Rocklabs	Std-OXD144	0.417	0.009	0.027
Rocklabs	Std-OxE126	0.623	0.015	0.045
Rocklabs	Std-OxF125	0.806	0.02	0.06
Rocklabs	Std-OxG103	1.019	0.028	0.084
Rocklabs	Std-Oxi81	1.807	0.033	0.099
Rocklabs	Std-OxJ120	2.365	0.063	0.189
Rocklabs	Std-OxK119	3.604	0.105	0.315
Rocklabs	Std-OxL118	5.828	0.149	0.447
Rocklabs	Std-OXP116	14.92	0.36	1.08
Rocklabs	Std-OxQ90	24.88	0.56	1.68
Rocklabs	Std-SE68	0.599	0.013	0.039
Rocklabs	Std-SE86	0.595	0.015	0.045
Rocklabs	Std-SF57	0.848	0.03	0.09
Rocklabs	Std-SF85	0.848	0.018	0.054
Rocklabs	Std-SG84	1.026	0.025	0.075
Rocklabs	Std-SH69	1.346	0.026	0.078
Rocklabs	Std-SH82	1.333	0.027	0.081
Rocklabs	Std-SI54	1.78	0.011	0.033
Rocklabs	Std-Si81	1.79	0.03	0.09
Rocklabs	Std-SJ80	2.656	0.057	0.171
Rocklabs	Std-SK93	4.079	0.089	0.267
Rocklabs	Std-SK94	3.899	0.084	0.252
Rocklabs	Std-SL76	5.96	0.192	0.576
Rocklabs	Std-SN75	8.671	0.199	0.597
Rocklabs	Std-SP37	18.14	0.38	1.14
Rocklabs	Std-SP59	18.12	0.36	1.08
Rocklabs	Std-SP73	18.17	0.42	1.26
Rocklabs	Std-SQ71	30.81	0.73	2.19
Rocklabs	Std-SQ83	30.64	0.717	2.151
Rocklabs	Std-SQ88	39.72	0.947	2.841

SOMIKA inserted 2,566 CRM's for LW analysis and 442 CRM's for TFA analysis in all sample batches submitted for the 2017/18 drilling on the Kalana deposit. Three hundred and seventy-three (373) CRM's were used for LW analysis and 48 CRM's for TFA analysis in all sample batches were submitted for the 2017/18 drilling on the Kalanako deposit. A CRM insertion summary can be found in Table 11.9-7.

Table 11.9-7 Insertion Summary by Deposit for CRMs Used for 2017/18 Drill Programmes at Kalana and Kalanako

CRM Name	Kalana CRM Count	Kalanako CRM Count
Std-HISILK4	52	-
Std-OxC129	1	-
Std-OXC145	45	43
Std-OxD128	36	-
Std-OXD144	32	14
Std-OxE126	154	15
Std-OxF125	53	27
Std-OxG103	49	42
Std-Oxi81	1	-
Std-OxJ120	200	16
Std-OxK119	73	15
Std-OxL118	201	24
Std-OXP116	23	48
Std-OxQ90		16
Std-SE68	57	1
Std-SE86	121	20
Std-SF57	1	-
Std-SF85	292	14
Std-SG84	247	17
Std-SH69	17	6
Std-SH82	99	-
Std-SI54	3	-
Std-Si81	55	12
Std-SJ80	46	-
Std-SK93	54	13
Std-SK94	223	-
Std-SL76	136	2
Std-SN75	63	7
Std-SP37	3	-
Std-SP59	2	-
Std-SP73	120	21
Std-SQ71	53	-
Std-SQ83	23	-
Std-SQ88	30	-
Grand Total	**2,565**	**373**

In summary, of the 2,566 CRMs used for LW analysis reviewed in the Kalana dataset, 185 samples, or 7.2%, returned values outside ±3SD for gold and are considered as failures. Of the 442 CRMs used for the Kalana drilling for TFA analysis, 43 samples failed or 9.7%. Of the 373 CRMs used for the Kalanako drilling for LW analysis, 21 samples failed or 5.6%. Of the 48 CRMs used for the Kalanako drilling for TFA analysis six samples, or 12.5% failed. CRM performance statistics are summarised in Table 11.9-8 and Table 11.9-9.

Endeavour considers an overall CRM failure rate of two percent to be acceptable. The Kalana and Kalanako inserted CRMs for LW and TFA analysis have universally provided greater than 2% failure; however, when a CRM failure is encountered the first step is to perform an internal assessment to determine if there was database miscoding or field level mis-insertion. If the failure is verified, the samples surrounding the CRM are evaluated. If the failed CRM occurs in a zone devoid of gold mineralisation, the laboratory is informed of the failure; however, a re-assay request is not made to the laboratory. If the failed CRM occurs in a zone of gold mineralisation, the laboratory is informed of the failure and a re-assay request is made. The interval requested for re-assay includes the failed CRM and the samples above and below which the CRM are intimately linked. All failed CRMs in zones of gold mineralisation were re-assayed over the course of the 2017/18 drilling programme at both Kalana and Kalanako. The failure statistics presented and outlined below in Table 11.9-8 and Table 11.9-9 represent failed CRMs in zones devoid of gold mineralisation.

When a re-assay is requested, the team at Kalana send the four-kilogram reference sample associated with the original sample. (This because a full two-kilogram is used for the LW/TFA analysis of the original samples and there is rarely a full two-kilogram is retained in the coarse reject.)

Table 11.9-8 Performance Summary for CRMs Used in the 2017/18 Kalana Drill Programme

CRM	No CRM LW	No CRM TFA	Min of AuLW_ppm	Min of AuTFA_ppm	Max of AuLW_ppm	Max of AuTFA_ppm	Average of AuLW_ppm	Average of AuTFA_ppm	Certified Value	Certified Std Dev	CRV - 3 SD	CRV + 3 SD	Failures LW	% Failures LW	Failures TFA	% Failures LW
Std-HISILK4	52	8	3.302	3.4	3.765	3.74	3.58	3.57	3.46	0.09	3.19	3.73			1	1.9%
Std-OxC129	1		0.459		0.459		0.46		0.21	0.01	0.18	0.23	1	100.0%		
Std-OXC145	45	5	0.193	0.2	0.232	1.83	0.21	0.53	0.21	0.01	0.19	0.23			1	2.2%
Std-OxD128	36	6	0.363	0.38	0.426	0.43	0.40	0.40	0.42	0.01	0.39	0.46	6	16.7%	2	5.6%
Std-OXD144	32	8	0.376	0.38	0.444	0.43	0.40	0.41	0.42	0.01	0.39	0.44	9	28.1%	1	3.1%
Std-OxE126	154	36	0.556	0.57	0.786	0.91	0.60	0.62	0.62	0.02	0.58	0.67	4	2.6%	4	2.6%
Std-OxF125	53	8	0.712	0.61	0.839	0.81	0.77	0.77	0.81	0.02	0.75	0.87	2	3.8%	1	1.9%
Std-OxG103	49	5	0.92	0.97	1.371	1.02	0.99	0.99	1.02	0.03	0.94	1.10	3	6.1%		
Std-Oxi81	1	1	1.743	1.81	1.743	1.81	1.74	1.81	1.81	0.03	1.71	1.91				
Std-OxJ120	200	25	0.996	2.2	2.626	2.62	2.35	2.35	2.37	0.06	2.18	2.55	5	2.5%	1	0.5%
Std-OxK119	73	12	3.404	2.72	4.03	3.77	3.58	3.48	3.60	0.11	3.29	3.92	1	1.4%	2	2.7%
Std-OxL118	201	25	0.994	5.42	7.629	6.12	5.89	5.74	5.83	0.15	5.38	6.28	4	2.0%		
Std-OXP116	23	1	14.88	16.55	17.445	16.55	15.83	16.55	14.92	0.36	13.84	16.00	5	21.7%	1	4.3%
Std-SE68	57	14	0.442	0.57	0.682	0.61	0.56	0.59	0.60	0.01	0.56	0.64	14	24.6%		
Std-SE86	121	25	0.489	0.55	0.77	6.13	0.57	0.81	0.60	0.02	0.55	0.64	17	14.0%	1	0.8%
Std-SF57	1		3.764		3.764		3.76		0.85	0.03	0.76	0.94	1	100.0%		
Std-SF85	292	44	0.573	0.77	1.071	0.89	0.80	0.82	0.85	0.02	0.79	0.90	48	16.4%	7	2.4%
Std-SG84	247	47	0.577	0.89	2.408	1.08	0.99	1.00	1.03	0.03	0.95	1.10	11	4.5%	6	2.4%
Std-SH69	17	5	1.2	1.28	1.345	1.42	1.26	1.35	1.35	0.03	1.27	1.42	7	41.2%		
Std-SH82	99	19	0.957	1.25	1.673	1.42	1.28	1.34	1.33	0.03	1.25	1.41	6	6.1%	3	3.0%
Std-SI54	3		1.738		1.832		1.80		1.78	0.01	1.75	1.81	2	66.7%		
Std-Si81	55	16	1.602	1.64	1.858	1.92	1.72	1.78	1.79	0.03	1.70	1.88	11	20.0%	3	5.5%
Std-SJ80	46	9	2.388	2.48	2.829	2.67	2.54	2.60	2.66	0.06	2.49	2.83	3	6.5%	1	2.2%
Std-SK93	54	10	3.754	3.92	4.935	4.33	4.13	4.10	4.08	0.09	3.81	4.35	2	3.7%		
Std-SK94	223	38	2.358	3.72	4.256	4.22	3.91	3.94	3.90	0.08	3.65	4.15	3	1.3%	2	0.9%
Std-SL76	136	22	0.636	5.66	7.385	6.18	5.82	5.93	5.96	0.19	5.38	6.54	6	4.4%		
Std-SN75	63	18	8.366	8.41	9.632	9.05	8.96	8.73	8.67	0.20	8.07	9.27	3	4.8%		
Std-SP37	3	2	18.36	17.05	18.79	17.05	18.58	17.05	18.14	0.38	17.00	19.28				
Std-SP59	2		17.825		19.13		18.48		18.12	0.36	17.04	19.20				
Std-SP73	121	8	16.35	17.1	22.41	20.4	18.69	18.54	18.17	0.42	16.91	19.43	11	9.1%	2	1.7%
Std-SQ71	53	12	29.32	28.3	32.99	32.2	31.09	30.12	30.81	0.73	28.62	33.00			1	1.9%
Std-SQ83	23	7	28.12	27.4	31.19	31.4	30.06	29.44	30.64	0.72	28.49	32.79			2	8.7%
Std-SQ88	30	6	38.72	30.3	42.69	41.5	40.71	38.62	39.72	0.95	36.88	42.56			1	3.3%
Grand Total	**2,566**	**442**											**185**	**7.2%**	**43**	**9.7%**

Table 11.9-9 Performance Summary for CRM Used for 2017/18 Drill Programme at Kalanako

CRM	No CRM LW	No CRM TFA	Min of AuLW_ppm2	Min of AuTFA_ppm	Max of AuLW_ppm	Max of AuTFA_ppm	Average of AuLW_ppm	Average of AuTFA_ppm	Certified Value	Certified Std Dev	CRV - 3 SD	CRV + 3 SD	Failures LW	% Failures LW	Failures TFA	% Failures TFA
Std-OXC145	43	8	0.193	0.2	0.255	0.21	0.21	0.21	0.21	0.01	0.19	0.23	1	2.3%		
Std-OXD144	14	1	0.351	0.76	0.568	0.76	0.41	0.76	0.42	0.01	0.39	0.44	5	35.7%	1	100.0%
Std-OxE126	15		0.565		0.639		0.59		0.62	0.02	0.58	0.67				
Std-OxF125	27	3	0.728	0.77	0.996	0.818	0.80	0.79	0.81	0.02	0.75	0.87	1	3.7%		
Std-OxG103	42	8	0.954	0.99	1.267	1.05	1.01	1.01	1.02	0.03	0.94	1.10	1	2.4%		
Std-OxJ120	16		1.952		2.543		2.33		2.37	0.06	2.18	2.55	1	6.3%		
Std-OxK119	15	2	3.511	3.58	3.773	4.06	3.66	3.82	3.60	0.11	3.29	3.92			1	50.0%
Std-OxL118	24	3	5.612	5.57	6.139	5.92	5.71	5.72	5.83	0.15	5.38	6.28				
Std-OXP116	48	3	14.435	14.497	15.676	15.6	15.27	14.94	14.92	0.36	13.84	16.00				
Std-OxQ90	16		23.82		25.98		25.07		24.88	0.56	23.20	26.56				
Std-SE68	1		0.503		0.503		0.50		0.60	0.01	0.56	0.64	1	100.0%		
Std-SE86	20	1	0.507	0.595	0.606	0.595	0.58	0.60	0.60	0.02	0.55	0.64	4	20.0%		
Std-SF85	14	2	0.811	0.82	0.826	0.826	0.82	0.82	0.85	0.02	0.79	0.90				
Std-SG84	17	3	0.933	1.005	1.027	1.012	1.00	1.01	1.03	0.03	0.95	1.10				
Std-SH69	6	3	1.251	1.39	1.416	1.51	1.31	1.46	1.35	0.03	1.27	1.42			2	66.7%
Std-Si81	12	4	1.666	1.762	1.795	1.81	1.73	1.78	1.79	0.03	1.70	1.88	1	8.3%		
Std-SK93	13		3.555		4.214		3.95		4.08	0.09	3.81	4.35	1	7.7%		
Std-SL76	2	1	6.055	4.57	6.146	4.57	6.10	4.57	5.96	0.19	5.38	6.54			1	100.0%
Std-SN75	7	3	8.645	8.3	9.559	9.37	9.26	8.97	8.67	0.20	8.07	9.27	2	28.6%	1	33.3%
Std-SP73	21	3	17.692	17.93	20.09	19.05	18.44	18.36	18.17	0.42	16.91	19.43	3	14.3%	0	0.0%
Grand Total	**373**	**48**											**21**	**5.6%**	**6**	**12.5%**

11.10 Independent Reviews

Verification of the Kalana Project has been completed by a number of companies over the years including:

- IAMGOLD in 2009 and 2010.
- Snowden in 2013. They reviewed the sampling and assaying procedures in 2013. Recommendations were implemented during the large re-assay campaign (2013-2014) and in 2015 during the last drilling campaign.
- Denny Jones in 2015. They also reviewed sample collection practices and assaying in 2015 and completed an audit of the assay database. Of 160,094 assay records used for the September 2015 resource estimate, 6,618 fire assay values and 23,456 LeachWELL assays were checked against assay certificates. No errors were identified and no serious issues were identified.
- CSA Global (Andre Vorster) visited the Project in 2018 (as part of an Endeavour Exploration-wide review) and audited the drilling and sampling protocols. Some recommendations were made to improve the sampling protocol, for example making sure the splitters were levelled and securing the splitters were shut.

The above authors visited the site, reviewed procedures, logging and exploration practices and geological interpretations. Some also included sampling, re-analysis of pulps, collections of new samples, metallurgical samples and database audits. The primary conclusions included:

- An understanding that the 50 g fire assay was inadequate to represent the actual grade of the mineralisation because of the coarse nature of some of the gold.
- There was good correlation between higher grades and the occurrence of visible gold.
- Overall, the standard exploration procedures are good and suitable for evaluating the deposits.
- The LeachWELL assay technique is appropriate and provides reasonable results.
- The geological interpretations are consistent with the information available (especially the underground workings data).
- There have been no material differences identified in the spreadsheets used for modelling when compared with the original data.

The overall opinion was that the exploration data was collected in a manner that is suitable for the purpose of Mineral Resource Estimation.

12.0 DATA VERIFICATION

12.1 Introduction

The following section includes discussion and comment on the data verification aspects relating to the underlying geological data utilised to support the geological models and ultimately the Mineral Resources as reported herein.

The Kalana Project is the outcome of several different generations of data, site visits, data verification exercises and Qualified Person (QP) assessments.

12.2 Historical Data Validation and Verification

In 2017, Endeavour Mining took control of the Kalana Project exploration programme which had been historically managed in Microsoft Excel. The drill hole database was subsequently migrated to an in-house, industry standard database management system (DBMS) called "DDHTools". All historical data was audited for errors such as inconsistent collar coordinates, incorrect or missing down hole survey records, missing sample-assay records, missing or overlapping errors in DTH interval data, etc. If and when these errors were identified, they were corrected in the master database.

Overall, the historical data is considered to be robust and suitable for use in Resource Estimation. There are, however, a number of exceptions including:

- The logs / records of the 2004 auger drill holes in the TSF cannot be located.
- The 2012 TSF auger holes in the TSF and the Shaft No. 2 TSF do not have surveyed collar elevations.

There are a number of historical drill holes in the Kalana Deposit Exploration Database that are not in the Resource Database, as summarised in Table 12.2-1, due to changing work practices, owners and operators and unreliable or unknown methodologies and protocols. The Resource Drill Hole Database also does not include condemnation holes (including 57 RC holes totalling 6,267 m on proposed infrastructure construction zones and 42 holes in a possible camp location) drilled since 2016.

Table 12.2-1 Exploration Drill Holes Not in the Kalana Deposit Resource Drill Hole Database

Company	Campaign	No. Holes	Type	Total Metres
SONAREM & SOGEMORK	1967 – 1982	815	Core	81,524
SOGEMORK	1989 - 1991	56	Core	16,000
("The Russians")	**Subtotal**	**871***		**97,524**
Ashanti-JCI	1995 - 1996	20	RC	1,419
		2**	DD	645
	Subtotal	**22**		**2,064**
SOMIKA	2004 Tailings Dam		Auger	
*Predominately underground holes **One hole also in resource DH DB (ASARCDD6)				

Information from an additional 27 historical Ashanti and 16 historical Russian drill holes are not in the Kalanako Exploration drill hole database as they have been superseded by modern holes.

The Kalana Project has a number of historical MRE's which were signed-off by Independent QPs who were satisfied with the integrity of the data and considered it to be satisfactory to be used as the basis of the respective Estimates.

12.3 Database Checks

Since 2017, data is managed with the strict built-in data integrity requirements of DDHTools. Database checks include identifying:

- Inconsistent collar co-ordinates.
- Incorrect or missing DTH survey records.
- Missing sample-assay records.
- Missing data or overlap errors in DTH interval data.
- Incorrect 3D plots of the drill hole traces.

All inconsistencies are reported to the database management team which are then actioned.

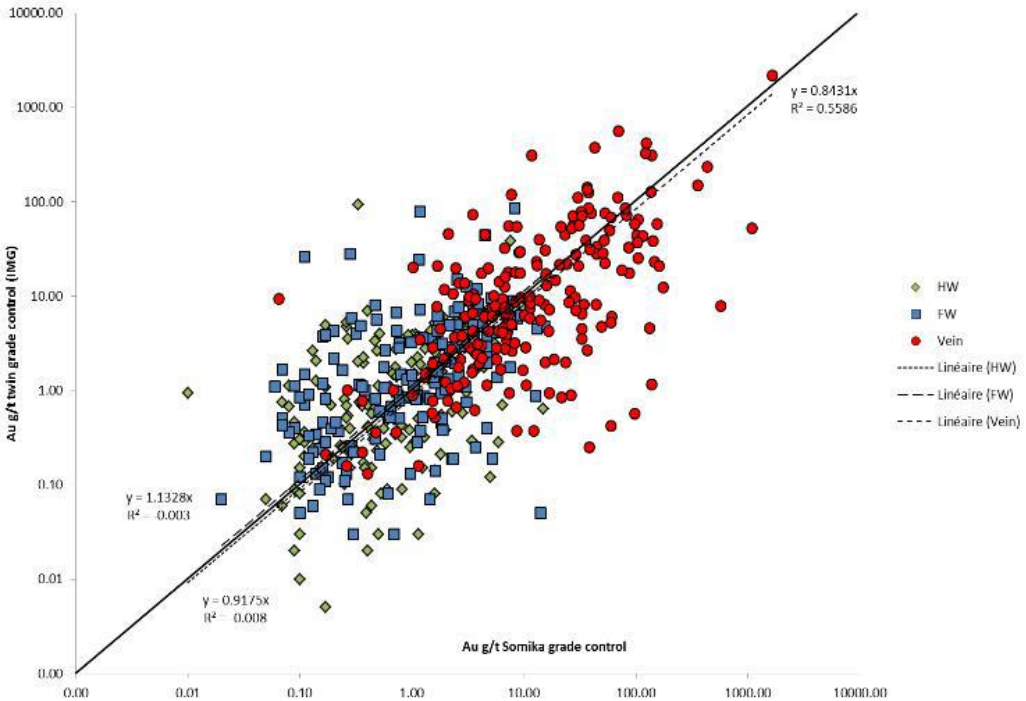
In April 2019, independent consultants CSA-Global visited the Project and reviewed the field procedures which generate the data in the database. All procedures related to drill hole location, downhole surveys, sample collection (analytical and density), quality control management and sample dispatch were subject to scrutiny. In 2019, drilling campaigns were underway at Kalanako Northeast and the targets in the south of the Project Area (e.g. Solomania, Kodialani, etc.). From the review, a report was generated, and all recommendations included were implemented. In addition, a new comprehensive Endeavour Exploration Standard Operating Procedure was generated for all projects across the Company.

12.4 Twinned Hole Comparison

Since Project inception, there have been a number of attempts to twin data (RC versus RC holes, RC versus RCDD holes, RC versus DD holes, underground raises versus pilot drillholes, twin underground grade control samples, etc.) but they have all had limited success due to the very high nugget effect of the coarse gold.

In 2010, 325 twin grade control samples were taken underground to assess the repeatability of the gold content. As Figure 12.4.1 illustrates, there was significant variability of gold grade between pairs, but the population distribution was comparable.

Figure 12.4.1 Underground Twin Grade Control Sample Results



The gold grade of the Kalana veins is highly variable over short distances, as demonstrated by Figure 12.4.2, which hampers grade repeatability between twin samples.

Figure 12.4.2 Vein 17 Raise, Level -150 m with Gold Grade and Thickness

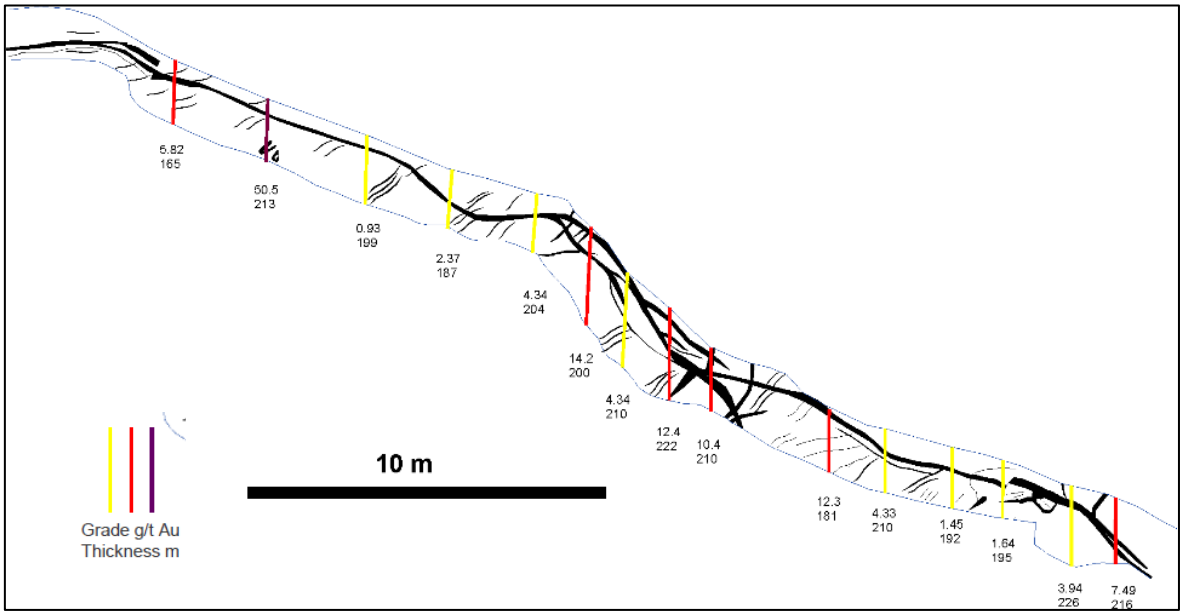
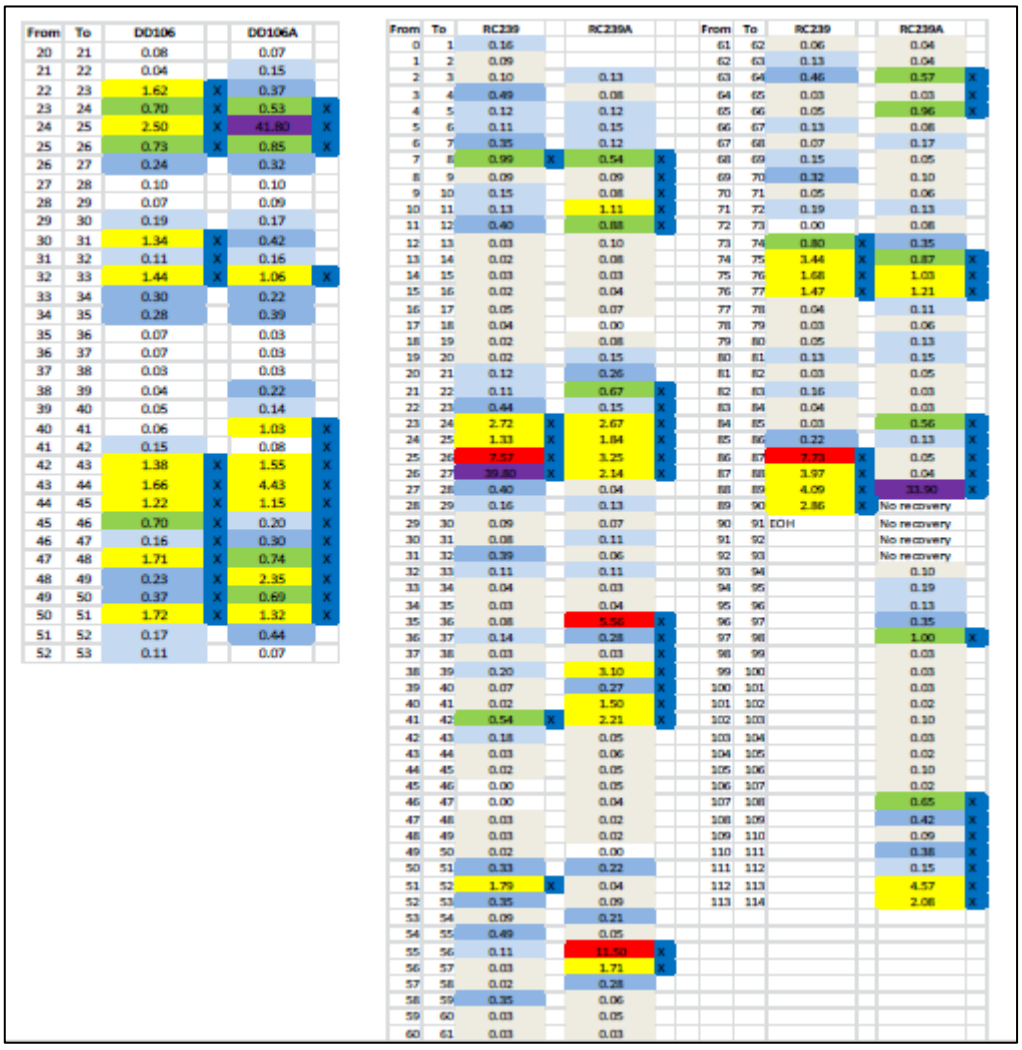


Figure 12.4.3 illustrates the gold grade and position of logged quartz veins for two DD holes (DD106 and DD106A) and two pairs of RC holes (RC39 v RC 39A and RC239 v 239A) drilled in 2011. The holes were typically one metre apart. As can be seen, there is a high variability of grade and there is some variation in the position and thickness of the vein (due to its anastomosing and pinching and swelling), which also impacts the reliability of twin samples.

Figure 12.4.3 Schematic of RC v DD Twin Holes Results



From	To	DD106		DD106A	
20	21	0.08		0.07	
21	22	0.04		0.15	
22	23	1.62	X	0.37	
23	24	0.70	X	0.53	X
24	25	2.50	X	41.80	X
25	26	0.73	X	0.85	X
26	27	0.24		0.32	
27	28	0.10		0.10	
28	29	0.07		0.09	
29	30	0.19		0.17	
30	31	1.34	X	0.42	
31	32	0.11	X	0.16	
32	33	1.44	X	1.06	X
33	34	0.30		0.22	
34	35	0.28		0.39	
35	36	0.07		0.03	
36	37	0.07		0.03	
37	38	0.03		0.03	
38	39	0.04		0.22	
39	40	0.05		0.14	
40	41	0.06		1.03	X
41	42	0.15		0.08	X
42	43	1.38	X	1.55	X
43	44	1.66	X	4.43	X
44	45	1.22	X	1.15	X
45	46	0.70	X	0.20	X
46	47	0.16	X	0.30	X
47	48	1.71	X	0.74	X
48	49	0.23	X	2.35	X
49	50	0.37	X	0.69	X
50	51	1.72	X	1.32	X
51	52	0.17		0.44	
52	53	0.11		0.07	

From	To	RC239		RC239A		From	To	RC239		RC239A	
0	1	0.16				61	62	0.06		0.04	
1	2	0.09				62	63	0.13		0.04	
2	3	0.10		0.13		63	64	0.46		0.57	X
3	4	0.49		0.08		64	65	0.03		0.03	X
4	5	0.12		0.12		65	66	0.05		0.96	X
5	6	0.11		0.15		66	67	0.13		0.08	
6	7	0.35		0.12		67	68	0.07		0.17	
7	8	0.99	X	0.54	X	68	69	0.15		0.05	
8	9	0.09		0.09	X	69	70	0.32		0.10	
9	10	0.15		0.08	X	70	71	0.05		0.06	
10	11	0.13		1.11	X	71	72	0.19		0.13	
11	12	0.40		0.88	X	72	73	0.00		0.08	
12	13	0.03		0.10		73	74	0.80	X	0.35	
13	14	0.02		0.08		74	75	3.44	X	0.87	X
14	15	0.03		0.03		75	76	1.68	X	1.03	X
15	16	0.02		0.04		76	77	1.47	X	1.21	X
16	17	0.05		0.07		77	78	0.04		0.11	
17	18	0.04		0.00		78	79	0.03		0.06	
18	19	0.02		0.08		79	80	0.05		0.13	
19	20	0.02		0.15		80	81	0.13		0.15	
20	21	0.12		0.26		81	82	0.03		0.05	
21	22	0.11		0.67	X	82	83	0.16		0.03	
22	23	0.44		0.15	X	83	84	0.04		0.03	
23	24	2.72	X	2.67	X	84	85	0.03		0.56	X
24	25	1.33	X	1.84	X	85	86	0.22		0.13	X
25	26	7.57	X	3.25	X	86	87	7.73	X	0.05	X
26	27	39.80	X	2.14	X	87	88	3.97	X	0.04	X
27	28	0.40		0.04		88	89	4.09	X	33.90	X
28	29	0.16		0.13		89	90	2.86	X	No recovery	
29	30	0.09		0.07		90	91	EOH		No recovery	
30	31	0.08		0.11		91	92			No recovery	
31	32	0.39		0.06		92	93			No recovery	
32	33	0.11		0.11		93	94			0.10	
33	34	0.04		0.03		94	95			0.19	
34	35	0.03		0.04		95	96			0.13	
35	36	0.08		5.56	X	96	97			0.35	
36	37	0.14		0.28	X	97	98			1.00	X
37	38	0.03		0.03	X	98	99			0.03	
38	39	0.20		3.10	X	99	100			0.03	
39	40	0.07		0.27	X	100	101			0.03	
40	41	0.02		1.50	X	101	102			0.02	
41	42	0.54	X	2.21	X	102	103			0.10	
42	43	0.18		0.05		103	104			0.03	
43	44	0.03		0.06		104	105			0.02	
44	45	0.02		0.05		105	106			0.10	
45	46	0.00		0.05		106	107			0.02	
46	47	0.00		0.04		107	108			0.65	X
47	48	0.03		0.02		108	109			0.42	X
48	49	0.03		0.02		109	110			0.09	X
49	50	0.02		0.00		110	111			0.38	X
50	51	0.33		0.22		111	112			0.15	X
51	52	1.79	X	0.04		112	113			4.57	X
52	53	0.35		0.09		113	114			2.08	X
53	54	0.09		0.21							
54	55	0.49		0.05							
55	56	0.11		11.50	X						
56	57	0.03		1.71	X						
57	58	0.02		0.28							
58	59	0.35		0.06							
59	60	0.03		0.05							
60	61	0.03		0.03							

The Qualified Person is of the opinion that twinning is not a robust method to assess the reliability and precision of collecting data due to the very high nugget effect seen at the Kalana Project.

12.5 Site Visits

Regular site visits to the Kalana Project have been made by the following Endeavour Mining Qualified Persons since 2017 to review the application and adherence to procedures and protocols:

- Kevin Harris, CPG, Vice-President Resources.
- Helen Oliver, MSc FGS CGeol, Group Resource Geologist.
- Geoff Day P.Geo, Exploration Group Database and Quality Control Manager.

Several independent consultants visited the Kalana Projects during the SOMIKA (Avnel) exploration campaigns between 2004 and 2015. Ivor Jones B.Sc (Hons), MSc, FAusIMM from Snowden Consultants and Denny Jones Pty Ltd visited the site a number of times between 2013 and 2015 and was satisfied with the quality of the data generated and considered it to be satisfactory to be used as the basis of a MRE.

12.6 Conclusions

Helen Oliver MSc FGS CGeol, as the Qualified Person on behalf of Endeavour Mining, is satisfied in the reliability, accuracy and precision of the data collected. The data verification processes completed to date have been completed in accordance with the relevant industry benchmarks and practices. Furthermore, the analysis completed to date has not identified any significant issues which would result in any inherent bias in the geological modelling and resource modelling which inform the Mineral Resources as reported in this Technical Report in the opinion of the Qualified Person.

The Qualified Person is of the opinion that the Kalana Project database is robust and suitable to be used as the basis of a Mineral Resource Estimate and this technical report.

13.0 METALLURGY AND FLOWSHEET DEVELOPMENT

13.1 Introduction

Endeavour Mining Corporation (Endeavour) is preparing a preliminary economic assessment (PEA) of the existing Kalana Gold Mine in south-western Mali, located in the Sikasso Region, approximately 250 km south of the capital city Bamako. The previous underground mining operation, closed by Endeavour in 2017, delivered ~50 ktpa of high grade ore to a process plant, to recover a gravity concentrate only. Endeavour intends to convert the operation to a conventional open pit mine with a CIL process plant facility.

Testwork conducted during a DFS undertaken by the previous project owners in 2015 did not adequately address some key processing options and metallurgical parameters required for process plant design and overall assessment of gold recoveries and operating costs and so Lycopodium developed a comprehensive metallurgical testwork programme for a DFS.

Lycopodium, in conjunction with Cube Consulting (Cube), selected appropriate core samples to represent all major ore lithologies prior to commencing the infill testwork programme.

The majority of the testwork was carried out between January and July 2018, with some additional tests performed between November 2018 and January 2019. The testwork programme was performed under the direction of Lycopodium using drill core recovered from a drilling campaign. The various laboratories detailed below performed the testwork:

- Australian Metallurgical and Mineral Testing Consultants (AMMTEC) Pty Ltd, (now ALS Ltd), Perth, Australia.
- JKTech Pty Ltd (JKTech), Queensland, Australia.
- Outotec, Perth, Australia.
- Knight Piésold, Perth, Australia.

The following provides a summary of the testwork undertaken as part of this infill testwork programme:

- Comminution tests performed on 14 samples representing a range of lithologies across the three main weathered rock types (saprolite, saprock, and primary) intended to produce a range of comminution data (SMC values, Bond Ball Mill Work indices, and Abrasion indices) for input into equipment sizing.
- Initial leach variability tests performed on 32 samples representing a range of lithologies across the three main weathered rock types, under standard cyanide leaching conditions and at a fixed particle size. The intention of these tests was to determine the relative gold recovery from the various lithologies and identify any anomalies prior to preparation of the Master Composites.

- LeachWELL tests performed on 56 primary ore samples to establish overall gold recoveries across the resource, with the results used to select appropriate samples, to establish two Primary Ore Master Composites based on 'high' and 'low' gold extraction.
- Optimisation tests performed on five Master Composites, to establish optimum leaching conditions (grind size, pulp density, and cyanide concentration) and to establish CIL, cyanide detoxification, and arsenic precipitation parameters.
- Oxygen uptake rate tests, and rheology tests for each of the five Master Composites.
- Final Variability tests performed on 20 samples representing a range of lithologies across the three main weathered rock types, under optimised grind, pulp density, and cyanide leaching conditions.

13.2 Previous Testwork Programmes

13.2.1 Testwork Programmes

Mineral Development Services (MDS), July 2004

Gravity recovery, leaching, and flotation testwork was conducted on run-of-mine samples, hanging wall, footwall, and vein samples, plant tailings and tailings dam samples.

DRA Comminution Testwork, 2014

Comminution testwork conducted at the Mintek Laboratories, Johannesburg, using selected competent material from the fresh ore zone. Ball work indices and breakage rates for the crushing and milling circuit design were determined.

DRA Leach and Detox Testwork, 2014

A programme of gravity recovery and leaching testwork was undertaken by SGS, Johannesburg, using six composite samples representing the saprolite (oxide), saprock (transitional), fresh (primary) ore, and tailings. The programme included head assaying, determination of gravity gold recovery potential, leach circuit testing and optimisation, and cyanide destruction requirements.

DRA Feasibility Study Testwork, 2015

A programme of testwork was conducted at SGS, Johannesburg, under the management of DRA, as part of the 2015 Kalana Feasibility Study. The programme included optimisation testing on a composite sample as well as variability testing on saprolite, saprock, and fresh ore samples to produce data for development of a gravity-CIL processing plant flowsheet and included abrasion index testing, head assaying, mineralogy, gravity recoverable gold determination, leach optimisation and variability testing, high shear reactor tests, cyanide destruction and arsenic precipitation requirements.

13.2.2 Testwork Outcomes

The following provides a summary of the outcomes from the previous comminution and metallurgical testwork programmes.

Mineral Development Services (MDS), July 2004

- Samples tested had a significant portion of gravity recoverable gold. Gravity recovery ranged from 35% to 53%.
- Gold extraction by cyanidation at a grind P_{80} of 75 µm and a leach time of 24 hours was high and ranged from 89% to 95%.
- Flotation tests conducted on fresh zone ore indicated that approximately 85% of the gold in primary samples could be recovered in approximately 5% of the feed mass.

DRA Comminution Testwork, 2014

- A limited number of comminution tests were conducted on fresh ore zone samples and indicated a Bond ball mill index range of 15.2 kWh/t to 17.6 kWh/t, indicative of a relatively hard ore.

DRA Leach and Detox Testwork, 2014

- Head assay analysis indicated that arsenic levels ranged from 1,124 ppm to 8,385 ppm.
- A high gravity gold component was identified in the ore types with saprolite, saprock and fresh ore zone samples containing 6%, 70% and 50% to 91% gravity recoverable gold, respectively.
- A grind P_{80} of 75 µm was selected as the optimal grind size along with a 24-hour leach time using CIL conditions. Cyanidation of the saprolite, saprock and fresh ore zone samples resulted in average gold extractions of 96%, 93% and 89%, respectively. Gold extraction from tailings samples averaged 78%.
- Air / SO_2 cyanide destruction tests successfully reduced weak acid dissociable cyanide (CN_{WAD}) to below 1 ppm from a starting concentration of 115 ppm, when 60 minutes of pre-conditioning with sodium metabisulphite (SMBS) was followed by continuous detox testing for a further 60 minutes. SMBS addition rates were 0.85 kg/t and copper sulphate dosage rate was 230 g/t.
- Hybrid cyanide destruction and arsenic precipitation tests conducted using SMBS and ferric chloride resulted in the CN_{WAD} being reduced to below detectable levels and soluble arsenic being reduced to 250 µg/L.

DRA Feasibility Study Testwork, 2015

- Gold head grade analysis using standard fire assay and the screened metallic protocol identified large variations in gold grades in many instances.
- Detailed head analysis confirmed the trend of high arsenic levels seen in the historical testwork (53 ppm to 4,875 ppm). Mineralogy identified arsenopyrite (FeAsS) as one of the sulphide minerals present. Base metal assays were generally low.
- An abrasion index for a fresh ore sample was determined (0.189 g).
- Gravity gold recoveries averaged 62%, 71% and 83% for the saprolite, saprock and fresh ore variability samples, respectively.
- A design grind P_{80} size of 75 µm was selected, even though similar gold recoveries were obtained at P_{80} of 90 µm.
- Gold extractions remained largely unchanged as pH was increased from 9.8 to 10.2, while cyanide usage decreased from 0.90 kg/t to 0.63 kg/t. All further testwork was conducted at pH 10.2.
- Testwork indicated that a pre-oxidation stage reduced overall cyanide consumption.
- Air / SO_2 detoxification tests successfully reduced CN_{WAD} concentrations to below the detection limit of 1 ppm (ICMC target of 50 ppm).
- A hybrid scheme whereby free and CN_{WAD} was destroyed and arsenic precipitated in a single stage was tested. Free and WAD cyanide were reduced to below the detection limits of 1 ppm. Arsenic was reduced from 1,100 ppb to 259 ppb.
- Limited static settling testwork was conducted on a saprolite, a Master Composite and a tailings samples. A high flocculant dosage rate of 100 g/t was determined for all samples and on this basis pre-leach thickening was excluded from the flowsheet proposed at that time.
- Variability testwork – Six saprolite, seven saprock and seven fresh samples were tested using the selected conditions of: 80% passing 75 µm, double gravity concentration using a lab Knelson concentrator with recovery of concentrate using a TBE separation, four hours of pre-oxidation at pH 10.2 with oxygen up to 20 ppm, 24-hour CIL test (40% w/w solids, 20 g/L carbon, 20 ppm DO, pH of 10.2).
  - Overall gold extraction for the saprolite samples tested ranged from 96.2% to 98.4% (average of 97.2%). Overall residue grade increased with head grade. Samples tested ranged from 2.32 g Au/t to 15.5 g Au/t (average 6.43 g Au/t).

- Overall gold extraction for the saprock samples tested ranged from 87.8% to 99.4% (average of 92.1%). Overall residue grade increased with head grade. Samples tested ranged from 1.44 g Au/t to 8.53 g Au/t (average 4.14 g Au/t).

- Overall gold extraction for the fresh samples tested ranged from 91.0% to 99.5% (average of 97.5%). Overall residue gold grade was relatively constant irrespective of gold head grade. Samples tested had grades ranging from 1.63 g Au/t to 42.8 g Au/t (average 13.9 g Au/t).

13.3 ALS Testwork Programme Outline

Based on the results of the previous testwork programme summarised above, a conventional gravity recovery and cyanidation treatment route was selected for the initial evaluation of samples from the Kalana prospect. The testwork programme was undertaken by ALS, Perth, under the guidance and supervision of Lycopodium, and had the following objectives:

- Select the most suitable processing route.
- Determine the optimum plant operating parameters for the ore being processed.
- Evaluate the variability in metallurgical performance for the primary ore source.
- Obtain design data required for plant design.

13.4 Sample Selection Methodology

A total of 1,326 kg of drill core from the Kalana resource, together with 56 kg of drill core from the Kalanako resource, was delivered to ALS, Perth, on 22 December 2017. No anomalies were reported with the inventory.

13.5 Comminution Testwork

The following comminution tests were carried out:

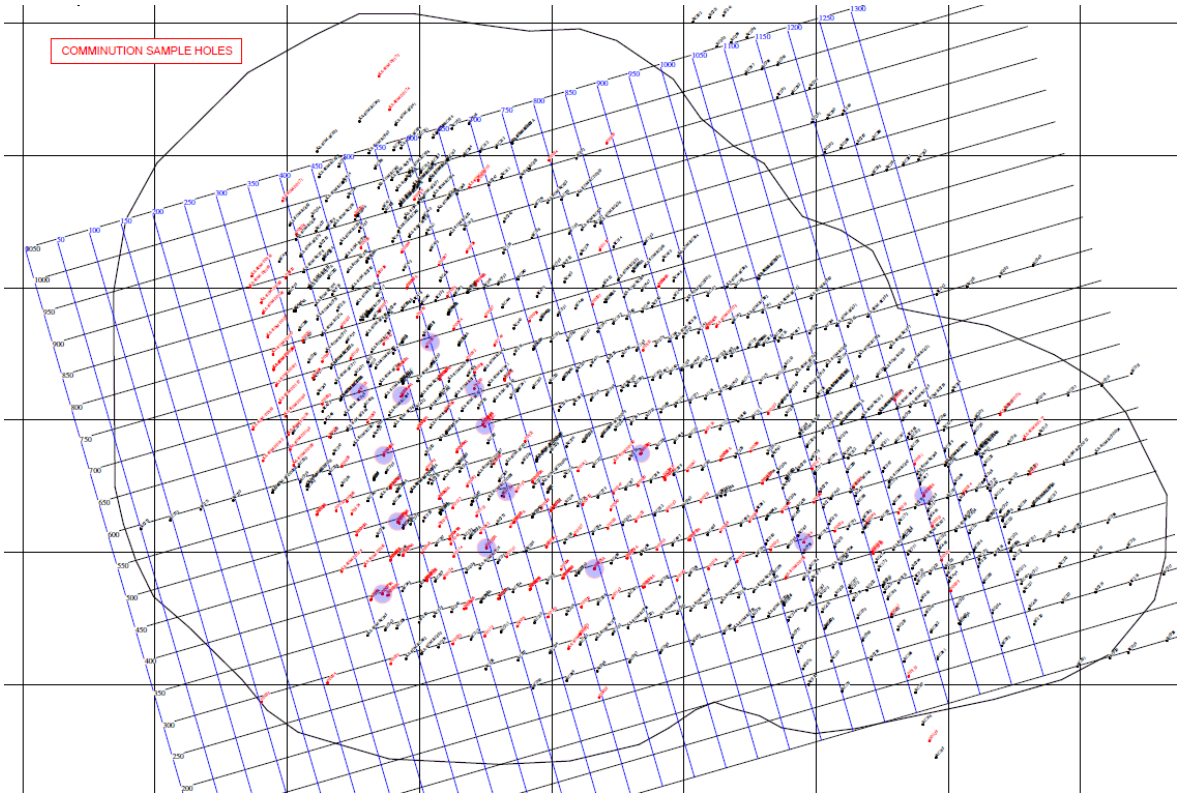
- SMC tests (@ -31.5+26.5 mm size fraction).
- Bond Abrasion Index (Ai) determination.
- Bond Ball Mill Work Index (BWi) determination.
- Modified Bond Ball Mill Work Index determination on all 'soft' material (pre-screen and remove all -106 µm material).

13.5.1 Sample Selection

Comminution testwork sample selection was based on the relative proportions of each individual weathered rock type and the overall contribution to the life of mine (LoM) ore tonnage.

An indication of the location of the comminution sample drill holes superimposed on the Kalana pit outline is shown in Figure 13.5.1.

Figure 13.5.1 Comminution Samples – Drill Hole Collars



A summary of the 14 samples selected for the comminution testwork programme is provided in Table 13.5-1.

Table 13.5-1 Comminution Testwork Samples

ALS Test No.	Hole ID	Appearance	Sample Mass (kg)	Lithology	Weathering	Au (g/t)
#1	KA-SEK-DD 44A	¼ HQ core	29.24	SDST	Primary	1.59
#2	KA-SEK-DD 71A	¼ HQ core	31.59	RHYTHM/SDST	Primary	1.34
#3	KA-SEK-DD 76B	¼ HQ core	27.54	SDST	Primary	2.28
#4	KA-SEK-DD 79A	¼ HQ core	30.11	SDST	Transition	1.04
#5	KA-SEK-DD 80A	¼ HQ core	30.84	SDST	Primary	1.48
#6	KA-SEK-DD 88C	¼ HQ core	26.19	SDST	Primary	0.60
#7	KA-SEK-DD 90	¼ HQ core	34.09	SDST	Oxide / Primary	0.64
#8	KA-SEK-DD 91	¼ HQ core	29.48	SDST	Primary	1.28
#9	KA-SEK-DD 101	¼ HQ core	28.77	SLST	Primary	5.72
#10	KA-SEK-DD 106A	Fine, broken core	30.98	DRT	Primary	0.86
#11	KA-SEK-DD 112A	Fine, broken core	29.25	DRT/SDST	Primary	1.02
#12	KA-SEK-DD 118	¼ HQ core	30.69	DRT/SDST	Oxide / Primary	4.85
#13	KA-SEK-DD 138A	¼ HQ core	26.51	SDST	Primary	1.60
#14	KA-SEK-DD 146	¼ HQ core	24.92	RHYTHM/SDST	Primary	1.11

13.5.2 SMC Tests

The drop weight index (DWi) results indicate that the transition ore (Sample #4) and primary siltstone (Sample #9) are the softest (DWi of 1.0 and 2.66 kWh/m^3), while the primary ore is significantly more competent, with DWi between 7.24 and 11.59 kWh/m^3.

With the exception of the transition ore (Sample #4) and primary siltstone (Sample #9) all samples tested yielded A x b values <40. The Kalana primary sandstone and diorite material is, therefore, considered to be very competent.

13.5.3 Bond Ball Mill Work Index Tests

Standard Bond Ball Mill Work Index (BBWi) tests were performed on 12 of the 14 comminution samples with closing screen sizes of 75 and 125 µm. (Sample #10 and Sample #11 contained excess fines and these were subjected to Truncated Bond Ball Mill Work Index tests only.) In general, the primary sandstone and dioritic samples returned high BBWi values (>16.1, max 19.1 kWh/t). In contrast the siltstone was significantly lower (12.6 kWh/t) while the lowest value was returned by the sandstone in the transition weathered phase (6.3 kWh/t).

The primary sandstone lithology contributes about 60% of the total tonnage to the overall Kalana mine reserve and thus the competency of this particular ore type has a dominant influence on the sizing of the milling circuit. Orway Mineral Consultants used the 85th percentile value of the BBWi data obtained from the closing screen size of 125 µm (i.e. to target a P_{80} of 90 µm) for mill design.

Standard Bond Abrasion Index (Ai) tests resulted in all the samples tested being categorised as Moderately Abrasive.

13.6 Initial Variability Testwork

The objective of the initial metallurgical variability tests was to compare the gold extraction for each sample under the same set of standard cyanidation leach conditions for each of the various lithologies and weathered rock types across the Kalana deposit. The results of this initial variability programme would then be used as a basis upon which to establish the Master Composites for the subsequent optimisation testwork programmes.

13.6.1 Initial Variability Testwork Results

There was a wide variation between the triplicate gold assays for some of the variability samples (notably Sample #8 and #11) suggesting an abundance of coarse gold particles. Potentially deleterious base metal elements were in low concentration although arsenic returned maximum and minimum concentrations across the 32 samples tested of 11,300 ppm and 65 ppm, respectively. The following provides a summary of the results:

- Gold extraction varied significantly across the samples, with 10 of the 32 initial variability tests returning gold extractions >95%, while 6 of the tests yielded gold extractions <80%.
- The highest tails gold assays corresponded with the highest S head assays.
- Gravity recoverable gold (GRG) from the 32 samples varied significantly, returning maximum and minimum values of 84.6% and 8.4%, respectively, with an average GRG of ~49%.
- Overall gold recovery from the eight Oxide ore samples tested was very high (93.4% minimum, 98.2% maximum, 95.7% average).
- Overall gold recovery from the three Transition ore samples was also high (82.7% minimum, 95.0% maximum, 88.5% average).
- Overall gold recovery from the 21 Primary ore samples was highly variable (19.0% minimum, 98.7% maximum, 78.2% average).

- Comparison of the gold recoveries based on primary ore lithology showed no discernible correlation. For example, the gold recovery from the three diorite primary ore samples ranged from 19.0% to 97.4%. Consequently, compilation of a Primary ore Master Composite, based on primary ore lithology, for the subsequent testwork phase was not possible based on these results alone.

13.7 Primary Ore LeachWELL Tests

One of the purposes of the Initial Variability testwork programme was to enable compilation of a number of Master Composites, based on lithology and metallurgical response, upon which to perform a subsequent programme of Optimisation testwork. However, the programme of Initial Variability tests failed to adequately identify a correlation between certain Primary ore types and metallurgical response, making it impossible to compile Master Composites for primary ore. As a consequence, a series of high intensity cyanide leach (LeachWELL) tests were performed on 56 primary ore samples across 21 diamond drill holes to facilitate compilation of Primary ore Master Composites based on cyanide-soluble Au recovery. (Note that the results of the LeachWELL tests were used for the purpose of characterising the various ore samples and are not a reflection of the anticipated gold recoveries from the commercial plant operation.)

Note also that the intensive nature of the LeachWELL tests ensures that all of the cyanide soluble gold is solubilised during the leach tests and none remains in the leach residue. The test does not, however, reflect the gold extraction performance that would be achieved under normal process plant operating conditions (assuming no pre-robbing mineral species are present).

Based on these Au recovery results, the samples have been categorised into high (>90%) and low (<90%) Au recovery and used to establish the two Primary ore Master Composites.

13.8 Optimisation Testwork Programme

Following completion of the comminution and initial variability testwork programmes, a separate optimisation testwork programme was implemented to optimise the grind size and subsequent cyanide leach conditions for the proposed CIL circuit. This optimisation programme also included oxygen uptake rate, preg-robbing index, standard carbon loading (kinetic), cyanide detoxification, arsenic removal, and slurry rheology tests.

13.8.1 Sample Selection

The optimisation testwork programme was implemented on five different Master Composites representing the different lithologies and gold grades within each weathered ore type of the Kalana deposit, and on a composite of the Kalanako deposit, viz:

- Master Composite #1 – Kalana Oxide ore.
- Master Composite #2 – Kalana Transition ore.
- Master Composite #3 – Kalanako Oxide ore.

- Master Composite #4 – Kalana Primary ore (high gold recovery).
- Master Composite #5 – Kalana Primary ore (low gold recovery).

13.8.2 Head Analysis

The results of the head analyses for each of the five Master Composites is shown in Table 13.8-1. The following comments can be made in reference to these results:

- There is considerable variability in the triplicate gold assays. This may indicate the presence of gold 'nuggets' (i.e. the nugget effect). To improve the reproducibility of the testwork and minimise the impact of nuggets on head assay determination, a gravity concentration step was adopted for all testwork. Preliminary testwork also highlighted the high proportion of gravity recoverable gold also verifying the requirement to use a gravity recovery stage in testwork to enhance reproducibility and potentially overall recovery.
- The average of the four Kalana head assays was 3.4 g Au/t which compares with the anticipated ROM grade of ~2.5 g Au/t.
- The Kalanako head assay of just 0.33 g Au/t was significantly lower than anticipated based on the assay data used to select the drill core in the first instance.
- Silver grades are consistently low for the oxide and transition material (below the detection limit of 0.2 g Ag/t) with higher values reported for the primary ore (0.9 g Ag/t). These low silver grades are unlikely to warrant special consideration in the design of the CIL and elution circuits.
- Mercury and cadmium concentrations are low (typically less than the assay detection limit of 0.1 ppm and 5 ppm respectively) and should not present an environmental or occupational health risk in the elution, electrowinning, or carbon regeneration circuits.
- Base metal concentrations for copper, lead, and zinc are all low and are thus unlikely to contribute to increased cyanide consumption and are also unlikely to adversely affect carbon performance.
- Organic carbon levels are low and preg-robbing should not be a problem.
- Arsenic concentrations are relatively high and a separate arsenic precipitation stage may need to be incorporated into the commercial plant flowsheet.

Table 13.8-1 Master Composite Head Assays

Element	Master Composite #1	Master Composite #2	Master Composite #3	Master Composite #4	Master Composite #5
Au (ppm)	8.43	2.35	0.27	2.03	1.01
Au (ppm)	10.6	3.70	0.51	1.79	1.38
Au (ppm)	12.2	5.35	0.22	0.59	1.10
Au av. (ppm)	10.41	3.80	0.33	1.47	1.16
Ag (ppm)	<2	<2	<2	0.9	0.9
Al (%)	9.44	8.48	10.5	8.16	8.2
As (ppm)	500	1,660	1,920	1,230	2,520
Ba (ppm)	760	640	600	600	700
Be (ppm)	<5	<5	<5	<5	<5
Bi (ppm)	<10	<10	<10	<10	<10
C (%)	0.09	0.21	0.48	0.60	0.69
C org (%)	<0.03	0.09	<0.03	<0.03	0.12
Ca (%)	0.065	0.200	0.640	1.10	1.10
Cd (ppm)	<5	<5	<5	<5	<5
Co (ppm)	20	35	25	15	20
Cr (ppm)	110	90	130	90	100
Cu (ppm)	62	50	78	48	74
Fe (%)	5.86	4.58	3.72	3.86	4.14
Hg (ppm)	0.1	<0.1	<0.1	<0.1	<0.1
K (%)	2.60	2.20	2.40	2.00	2.40
Li (ppm)	25	40	25	60	35
Mg (%)	0.56	1.28	0.68	1.56	1.56
Mn (ppm)	555	265	270	450	420
Mo (ppm)	<5	<5	<5	<5	<5
Na (%)	0.20	0.72	0.82	2.15	2.14
Ni (ppm)	65	85	80	50	50
P (ppm)	400	300	200	600	500
Pb (ppm)	<5	10	<5	15	<5
S (%)	0.78	0.34	0.34	0.64	0.78
S^{2-} (%)	<0.02	0.26	0.28	0.54	0.66
Sb (ppm)	2.3	2.9	1.8	1.5	3.4
SiO_2 (%)	61.2	63.2	60.8	65.4	63.6
Sr (ppm)	68	88	184	232	198
Te (ppm)	<0.2	<0.2	<0.2	0.6	0.4
Ti (ppm)	4,400	3,800	4,600	3,000	3,400
V (ppm)	128	112	130	92	98
Y (ppm)	<100	<100	<100	<100	<100
Zn (ppm)	82	106	84	80	106

The results of the tests performed on each of the Master Composites indicate that preg-robbing is unlikely to occur but, if required, the plant design can incorporate a pre-leach stage.

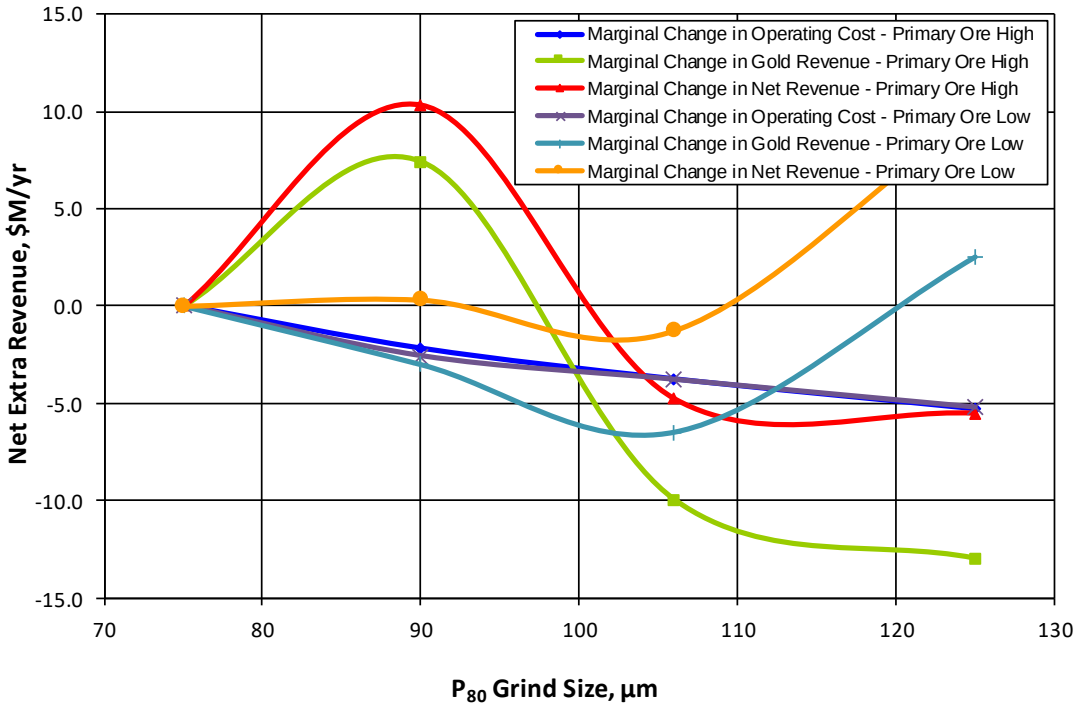
13.8.3 Grind Optimisation Tests

Cyanide leach and gravity concentration tests were performed on each of the five Master Composites at four different grind sizes (P_{80} 125 µm, 106 µm, 90 µm, 75 µm) under standard leach conditions and an optimum grind analysis was undertaken to determine the most economic grind size for the Kalana prospect primary ore.

The evaluation compared gold revenue against operating and capital expenditure for the grind sizes considered

A graphical summary of the evaluation for the Kalana Primary Ore is presented in Figure 13.8.1.

Figure 13.8.1 Grind Optimisation Summary



The results of the grind optimisation study indicate:

- Maximum revenue is realised for the high recovery primary ore at a grind size P_{80} of 90 µm, while maximum revenue is generated for the low recovery primary ore at a grind size P_{80} of 125 µm.

- The increase in gold revenue (recovery) with fineness of grind is offset by the increase in operating costs to achieve the finer grind sizes.

Given the significantly greater net revenue delta for the high recovery primary ore when comparing the P_{80} grind sizes of 90 µm and 125 µm (~$15.8M/year) versus a similar comparison for the low recovery primary ore (~$9.6M/year), the optimum grind size P_{80} for the Kalana primary ore is 90 µm.

13.8.4 Pulp Density and Cyanide Optimisation Tests

The results of pulp density and cyanide optimisation leach tests indicated that the bulk leach tests on the Master Composites should be performed at 45% w/w solids and 350 ppm NaCN starting concentration.

Diagnostic leach tests indicated that approximately two thirds of the unleached gold from Master Composite #5 is locked within sulphide minerals, confirming the refractory nature of this primary ore sample and the need to adopt a different processing route for this particular ore type if gold recoveries are to be improved. The remaining one third of the unleached gold is locked in silicates.

Although the oxygen uptake demand is low (i.e. significantly less than 0.15 mg/L.min for all values recorded), the addition of oxygen instead of air to the CIL tanks is proposed for the preliminary Kalana plant design to reduce cyanide consumption, improve leach kinetics, assist in oxidising reactive sulphides and reduce potential for HCN gas generation

The results of the viscosity tests indicate that while oxide ores should not be leached at pulp densities greater than 50% w/w solids, the primary ores are at risk of sanding in the CIL tanks at low solids densities. Consequently, a pre-leach thickener will be included ahead of the CIL circuit.

Bulk Cyanidation Testwork

Bulk gravity separation and cyanidation testwork was conducted on each of the five Master Composites under the optimised conditions in order to provide sufficient leach slurry for the subsequent carbon adsorption testwork. A summary of the bulk cyanidation testwork results is provided in Table 1.4-1 .

Table 13.8-2 Bulk Cyanidation Leach Results at Optimised Conditions

Master Composite	Grind Size P_{80} (µm)	% Solids (w/w)	Initial NaCN (ppm)	Calc. Head (g/t)	GRG (%)	Overall Recovery (%)		Reagent Consumption	
								NaCN (kg/t)	Lime (kg/t)
				Au	Au	Au	Ag		
Composite #1 [Kalana Oxide]	90	45	350	3.89	72.8	98.1	71.0	0.15	1.87
Composite #2 [Kalana Trans]	90	45	350	2.54	35.7	95.5	78.5	0.23	1.90
Composite #3 [Kalanako Oxide]	90	45	350	0.69	50.3	93.7	66.1	0.21	0.53
Composite #4 [Kalana Primary]	90	45	350	1.38	9.35	92.0	72.9	0.23	0.50
Composite #5 [Kalana Primary]	90	45	350	1.36	10.2	76.2	68.7	0.21	0.37

The carbon adsorption testwork results indicate good gold adsorption characteristics for most composites except the Kalanako Oxide composite which had a relatively low equilibrium carbon loading.

13.8.5 Cyanide and Arsenic Detoxification Tests

Based on the results of the CIL cyanidation leach tests, a programme of cyanide detoxification testwork was performed on the discharge slurries whose leaches were performed at an initial NaCN concentration of 350 ppm.

The cyanide detoxification testwork indicates the following:

- CN_{WAD} concentrations in the discharge liquor following cyanide detoxification for each of the Master Composites may be reduced to <3 ppm CN_{WAD} using either milk of lime or sodium hydroxide solution for pH control.
- The target CN_{WAD} value of <5 mg/L in the cyanide detoxification discharge liquor is able to be achieved with the addition of ~200% stoichiometric addition of SMBS for each of the Master Composites.
- It is necessary to supplement the cyanide detoxification reactions with copper sulphate solution for each of the Master Composites as there is insufficient soluble copper to catalyse the reaction following cyanide leaching.

A series of arsenic precipitation tests was performed on the discharge slurry from the cyanide detoxification tests. The results of the arsenic precipitation tests indicate the following:

- Arsenic concentrations in the final discharge solution of ~3 mg/L, <0.5 mg/L, ~1 mg/L and <0.5 mg/L are able to be achieved at pH 6.0 using ferrous sulphate addition to the cyanide detoxification discharge slurry from Master Composite #2 (Kalana Transition), Master Composite #3 (Kalanako Oxide), Master Composite #4 (Kalana Primary – High Au Recovery) and Master Composite #5 (Kalana Primary – Low Au Recovery), respectively.

- Arsenic concentration in the final discharge solution increases with increasing pH of the Scorodite precipitation process.

- There is a marginal decrease in final soluble arsenic concentration when the Fe:As ratio is increased from 5:1 to 10:1.

Average sulphuric acid consumption for the four composites tested at pH 6 is 3.3 kg/t. The results of these additional arsenic precipitation tests indicate the following:

- Arsenic concentrations in the final discharge solution of 1 mg/L, 0.9 mg/L, and 0.8 mg/L are able to be achieved at natural pH (i.e. no acidification) using ferrous sulphate addition to the cyanide detoxification discharge slurry from Master Composite #2 (Kalana Transition), Master Composite #3 (Kalanako Oxide), and Master Composite #4 (Kalana Primary – High Au Recovery), respectively.

- Arsenic concentrations in the final discharge solution improve with increasing Fe:As ratios and/or increasing DO concentrations.

- The results of the arsenic precipitation test programme have successfully demonstrated the ability to achieve low arsenic concentration in the final effluent using the Scorodite process.

Although these final arsenic concentrations still exceed the target value of <0.5 mg/L for the individual composites, it is likely that the commercial facility will be exposed to a blend of composites which, it is expected, will reduce the final solution arsenic concentration to <0.5 mg/L.

13.8.6 Intensive Cyanidation of Gravity Concentrate

These results demonstrate a consistently high recovery of gold from the gravity concentrate from each of the Master Composites when employing an intensive cyanidation leaching process.

13.9 Variability Metallurgical Testwork

Sample Selection

A programme of variability testwork was undertaken on twenty variability samples, selected to represent distance along strike and variations in depth and grade for the orebody. The results of variability testing indicate that:

- Gravity gold recoveries averaged 52% (range 15 to 91%). The samples with the highest calculated head grades typically have the highest gravity recoverable gold component.
- The average residue grade for the variability samples was 0.31 g Au/t and ranged from 0.02 to 1.64 g Au/t. This high residue value (1.64 g Au/t) is isolated and is associated with the highest sulphide sulphur feed content (2.50% w/w) which is indicative of a highly refractory sample.
- The average overall gold recovery (gravity + cyanide leach) across all variability samples was 88.9%.
- Leach rate curves show rapid gold dissolution in most cases with the majority of total cyanide recoverable gold leached in the first 2 to 4 hours.
- An average cyanide consumption of 0.13 kg/t was observed for the variability samples with a range of 0.06 to 0.25 kg/t. These results compare favourably with the testwork conducted on the Master Composites (0.10 kg/t to 0.22 kg/t) indicating relatively consistent metallurgical response throughout the orebody.
- An average lime consumption (60% available CaO) of 0.80 kg/t was observed for the variability samples with a range of 0.25 kg/t to 2.40 kg/t. These results compare favourably with testwork conducted on the Master Composites (0.30 kg/t to 1.85 kg/t).

13.10 Design Criteria Development

13.10.1 Selected Treatment Route

The metallurgical treatment route selected has been based on the results of the comprehensive 2018 comminution and metallurgical testwork programme with consideration of results from previous testwork programmes and may be summarised as follows:

- Single stage primary crushing.
- SABC (SAG mill, Ball mill, recycle pebble crushing) comminution circuit.
- Gravity concentration.
- CIL feed thickening.

- Carbon-in-leach (CIL).
- Split AARL stripping circuit.
- Air / SO_2 cyanide detoxification circuit.
- Arsenic precipitation circuit.

13.10.2 Key Process Design Parameters

The key process design parameters derived from the comminution and metallurgical testwork are summarised in Table 1.4-2.

Table 13.10-1 Key Process Design Parameters

Milling Circuit Design	
SMC A x b	26.7 (85th percentile)
Bond Ball Mill Work Index (BBWi)	18.7 kWh/t (85th percentile)
Crushing Work Index (CWi)	27.7 kWh/t (85th percentile)
Abrasion Index (Ai)	0.151 (85th percentile)
Primary Crushing Circuit Product Size, P_{80}	130 mm
Milling Circuit Product Size, P_{80}	90 µm
Gravity Gold Recovery	65%
Gravity Silver Recovery	15%
CIL Design	
Pre-leach Thickener Flux	1.0 t/m^2.h
Flocculant Addition	30 g/t
CIL Residence Time	36 hours (primary ore)
CIL Pulp Density	50%w/w solids (primary ore)
Cyanide Addition (nominal)	0.13 kg/t (oxide ore), 0.15 kg/t (primary ore)
Lime Addition (nominal)	1.27 kg/t (oxide ore), 0.67 kg/t (primary ore)
Aeration	Oxygen sparging
Operating pH	10.5
Carbon loading kinetic parameters, k and n values	182, 0.75 (gold); 108, 0.68 (silver)
Cyanide Destruction Design	
Cyanide Destruction Process	Air/SO_2
Residence Time	2 hours
SMBS Stoichiometric Requirement	200%
Operating pH	9.0
pH Control Reagent	Sodium Hydroxide Solution
Aeration	Oxygen sparging
Arsenic Precipitation Design	
Residence Time	1 hour
Iron Sulphate Addition, Fe:As	15:1
Operating pH	Natural (as received)
Aeration	Oxygen sparging

13.10.3 Metallurgical Recoveries

The range of metallurgical results for the leach tests conducted on each of the variability samples is summarised in Table 13.10-2. All samples that returned a gold head assay of less than 0.5 g/t Au have been excluded from the summary table as this low grade ore would not typically report to the commercial operation. Similarly all samples which have subsequently been found to fall outside of the preliminary mine pit shell model have also been excluded from the recovery calculation.

Table 13.10-2 Variability Tests – Gold Recovery

Rock Type	Au Extraction, %		
	Minimum	Maximum	Average
Oxide	90.6	99.1	96.0
Transition	82.7	95.0	90.0
Primary	44.1	99.6	88.6

Testwork Results Applied to the Mine Schedule

The results of the variability testwork programme suggest a poor correlation across all rock types within the Kalana prospect between the calculated gold head grade and corresponding gold residue grade.

An alternative approach to predicting the metallurgical recoveries for gold and silver is to take the average extractions obtained from the leach variability tests and then apply these to the mine schedule for the appropriate ore type to yield a weighted recovery value for each year.

Table 13.10-3 presents a summary of the preliminary mine schedule (developed for 3.0 Mtpa primary ore throughput), tonnage contribution from each ore type, and gold and silver extractions obtained from the variability testwork for each ore type. Included in this schedule is the retreatment of ~0.82 M tonnes of existing gravity circuit tailings

Table 13.10-3 Mine Schedule and Gold and Silver Extractions

Rock Type	LoM Au (g/t)	LoM Ag (g/t)	% of LoM Tonnage	LoM Tonnes (Mt)	Overall Au Extraction (%)	Ag Extraction (%)
Oxide	1.64	0.55	27.3	9.72	96.0	46.5
Transition	1.67	0.56	8.9	3.18	90.0	45.8
Primary	1.57	0.52	61.5	21.89	88.6	45.5
Tailings	1.67	0.56	2.3	0.82	74.8	45.8

Using this information, and assuming soluble gold and silver losses of 0.01 mg/L and 0.03 mg/L respectively and a CIL pulp density of 50% w/w solids, it is possible to predict the gold and silver recoveries for each year of the life-of-mine. These recoveries are summarised in Table 1.4-3.

Table 13.10-4 Predicted Life of Mine Metallurgical Recoveries

Mine Year	1	2	3	4	5	6	7	8	9	10	11
Au Extraction, %	94.1	92.8	91.7	90.4	89.0	88.9	92.1	88.8	88.6	88.6	88.6
Ag Extraction, %	46.4	46.2	46.0	45.9	45.6	45.5	46.0	45.5	45.5	45.5	45.5

This yields life-of-mine average gold and silver recoveries of 90.4% and 45.8%, respectively.

13.10.4 Reagent Consumption

Based on the revised lime purity, residual sodium cyanide concentration in the CIL discharge slurry, and the relative ore type contribution to the overall reserve, the life-of-mine sodium cyanide and lime consumptions for the selected treatment route may be estimated as summarised in Table 1.4-4.

Table 13.10-5 CIL Lime and Cyanide Consumption

Composite	NaCN Consumption kg/t	Lime[1] Consumption kg/t
Composite #1 (Kalana Oxide)	0.14	1.80
Composite #2 (Kalana Trans)	0.22	1.85
Composite #3 (Kalanako Oxide)	0.10	0.70
Composite #4 (Kalana Primary)	0.17	0.40
Composite #5 (Kalana Primary)	0.14	0.30

Note: 1. Lime consumption based on 90% available CaO.

13.11 Grind Sensitivity of Refractory Ore

The testwork results lead to the conclusion that there is little improvement in gold extraction for the Kalana refractory ore (Master Composite #5) unless much fine grinding to a P_{80} of 15 µm is performed, which is unlikely to be economic.

13.12 Thickening Testwork

The purpose of the thickening testwork was to determine the optimum size of a pre-leach thickener and to determine the flocculant type, flocculant addition rate, thickener overflow clarity, underflow density, and thickener underflow slurry yield stress. Slurry samples for each of the five Master Composites (milled to a target P_{80} of 90 µm) underwent a thickening testwork programme conducted by Outotec. Slurry pH was pre-adjusted to pH 10.5 using milk of lime to replicate the commercial plant operation. The results of the thickening testwork concluded that each of the samples can be successfully thickened to high densities over a range of flux rates whilst maintaining low (typically between 10 and 30 g/t) flocculant addition rates.

13.13 Tailings Testwork

A series of slurry samples generated from the metallurgical testwork programme were despatched to the Knight Piésold laboratory in East Perth to undertake a series of consolidation and geochemical tests.

Results of the consolidation testwork indicated that the rate of supernatant release is rapid, with the majority of water likely to be released from a tailings storage facility within one or two days. The results also suggested that the expected water release is high (40% to 60% of the water in the slurry).

14.0 MINERAL RESOURCE ESTIMATES

14.1 Informing Data

14.1.1 Kalana

The Kalana Resource Drill Hole Database, with an effective date of 12 July 2018, contains nearly 1,300 holes for approximately 218,000 m, see Table 14.1-1. The 246 drill holes drilled in 2017/18 are additional holes to the 2016 Technical Report. No drilling has occurred since July 2018.

Table 14.1-1 Kalana Deposit Resource Drill Hole Database, as of 12 July 2018

Company	Campaign	No. Holes	Type	Total Metres
Ashanti-JCI	1996	1	RCDD	294
SOMIKA	2008	41	RC	1,401
*IAMGOLD & SOMIKA	2010 - 2013	567	RC	79,388
		226	DD	53,004
	Subtotal	**834**		**133,793**
SOMIKA (Avnel)	2014	17	AC-RC	1,523
	2015	137	RC	21,405
		16	RCDD	3,950
		19	DD	2,811
		9	DDTT	1,976
	Subtotal	**198**		**31,665**
SOMIKA (Endeavour)	2017 - 2018	165	RC	24,914
		77	RCDD	25,841
		4	DD	1,347
	Subtotal	**246**		**52,102**
Total		**1,279**		**217,854**
*Includes 19 "H" holes which were UG DD GC holes drilled by SOMIKA in 2010.				

A breakdown of the assayed drill holes in the Resource Drill Hole Database is presented in Table 14.1-2.

Table 14.1-2 Kalana Deposit Resource Drill Hole Database by Type

Hole Type	Number	Assayed Length (m)
AC-RC	17	1,523
DDH	249	57,162
DDTT	9	1,976
RC	910	127,106
RC-DD	94	30,086

Total	**1,279**	**217,853**

Kalana underground drilling, channel and grab samples have not been used in the grade estimate. Geological and mineralisation qualitative data, however, has been used to aid the interpretations, as have the surveyed underground workings.

14.1.2 Kalanako

The Kalanako Resource Drill Hole Database, with an effective date of 7 February 2018, contains approximately 380 holes for approximately 46,500 m, see Table 14.1-3. The SOMIKA (Avnel) 2016 drill holes were drilled after the publication of the March 2016 MRE by Denny Jones. They were further supplemented by 66 holes for just over 8,000 m by SOMIKA (Endeavour) in late 2017/early 2018 (including two geotechnical triple tube diamond drill holes (DDT)).

Information from 27 historical Ashanti and 16 historical Russian drill holes was not used for the MRE as they have been superseded by modern holes. The database does not include the results of the 2019 Kalanako North Extension drilling campaign (27 holes) and reconnaissance exploration holes.

Table 14.1-3 Kalanako Resource Drill Hole Database, as of 7 February 2018

Company	Campaign	No. Holes	Type	Total Metres
SOMIKA (Avnel)	2010 - 12	29	DD	7,409
		206	RC	22,145
	Dec 2016	82	RC	8,635
SOMIKA (Endeavour)	2017 - 2018	63	RC	7,842
		1	DD	189
		2	DDT	350
Total		**383**		**46,570**

Drilling on the deposit is organised on west-east section lines on 25 mN centres. On these section lines, the drill hole spacing is typically 25 m and most drill holes dip -55˚ towards the west, with occasional drill holes scissored to the east. The general trend of the mineralisation runs diagonal to this drilling pattern, with lode strikes typically bearing 310˚ and occasionally 325˚. Optiro notes this means that the drill hole orientation is not optimal for the deposit, thus the effective along-strike drillhole spacing is typically in the range of 35 m to 45 m. The drilling direction is not perpendicular to the interpreted mineralisation planes and thus downhole continuity patterns are apparent thicknesses rather than close to true thickness.

14.1.3 TSFs

The volume of the Shaft No. 2 TSF as of June 2020 is modelled from drill hole logs, current topographic surfaces and historical partial survey data to be 128,000 m^3. Note - the interpreted volume may be conservative, inasmuch as it does not cover the full lateral footprint of the 2012 auger drilling and in some cases does not cover the full depth extent of the holes. It includes the depletion of 54,000 t reprocessed in 2016-17 by SOMIKA.

The TSF volume is estimated with consideration of the SOGEMORK estimated volume in 2004 by Snowden, preliminary dam design reports, the Denny Jones 2015 MRE, production plant discharge data (see Table 14.1-4) and various topographic surveys. It is estimated to be 653,000 t as of June 2020. Note, the total SOMIKA discharge estimated by the plant (643 kt) plus the 2004 survey volume (234 kt) equals 877 kt whereas the estimated volumes of the two TSFs is 845 kt, a slight difference of 32 kt or 4%; this is considered to be acceptable within the accuracy of an Indicated Resource. Endeavour undertook an order of magnitude volume / tonnage check by utilising the TSF design contour data and an extrapolation of the topography at the base of the TSF to calculate a TSF volume. Neither a final survey of the new TSF or a survey of the original topography was available; however, the TSF design footprint was tested by overlaying satellite imagery sourced from Google Earth and a good correspondence was observed. Working with an assumed dry bulk density of 1.5 t/m^3 and the calculated volume, a tonnage was produced that was compatible with the cumulative total tonnage calculated from the production statistics provided.

Table 14.1-4 SOMIKA Processing Plant Discharge to the TSFs

Year	Incremental Discharge		Cumulative Discharge	
	Tonnage	Grade (Au g/t)	Tonnage	Grade (Au g/t)
2004	35,410	2.95	35,410	2.95
2005	34,885	2.11	70,295	2.53
2006	27,743	2.75	98,038	2.59
2007	35,222	2.02	133,260	2.44
2008	48,262	1.9	181,522	2.30
2009	49,347	1.6	230,869	2.15
2010	50,238	1.08	281,107	1.96
2011	47,546	1.08	328,653	1.83
2012	49,484	0.97	378,137	1.72
2013	50,848	1.36	428,985	1.68
2014	49,513	1.41	478,498	1.65
2015	52,235	1.55	530,733	1.64
2016	54,043	1.66	584,776	1.64
2017	58,381	1.55	643,157	1.63
Total	**643,157**	**1.63**		

The Shaft No. 2 TSF was also sampled by some of the 2015 SOMIKA (Avnel) and 2017-18 SOMIKA (Endeavour) Kalana Deposit resource delineation drill holes (64 intervals from 29 holes with a length weighted average grade of 1.7 Au g/t).

Endeavour reports that approximately 54 kt of tailings were treated from the northern end of the Shaft No. 2 TSF in 2016 and 2017, and realised a gravity-recovered grade of 7.6 Au g/t. Endeavour notes that some of this material was treated in conjunction with underground ore and suggests that some coarse gold may have been retained in the plant circuit which should not be attributed to the TSF material. Endeavour concludes that the claimed recovered grade for the TSF material should be reduced to 6.0 g/t Au.

Note, some information pertaining to the TSFs including the original topographic surveys, actual collar elevation data for the 2012 auger data and results from a 2004 auger drilling programme are currently unavailable.

14.2 Resource Estimation

The resource estimation methodology applied for the Kalana and Kalanako deposits is very similar.

Preliminary data preparation included the following steps:

- Visual appraisal of imported data for consistency with expectations.
- Visual evaluation of spatial sample grade distribution.
- Evaluation of raw sample length.
- Application of codes to the drillhole data to discriminate the weathering, lithology and mineralisation conditions provided by the supplied interpretations.
- Review of grade distribution characteristics within the vein package interpretation.

An appraisal of downhole sample length demonstrated that over 99% of the drillhole data had sample lengths of one metre or less. Consequently, the drillhole data was composited to a target length of one metre within the constraints provided by the vein package, weathering and lithology interpretations. This composited data was used for all further analysis.

14.2.1 Mineralisation Distribution

Kalana

The Kalana vein package (VP) interpretation was compiled using a collection of conditions based on gold grade, presence of quartz, presence of visible gold and structural and lithological factors, as illustrated by Figure 14.2.1 to Figure 14.2.3. A nominal gold grade threshold of 0.2 g/t was applied during the interpretation process. While the vein package interpretation excludes a considerable proportion of samples with sub-0.2 Au g/t grades, the distribution of grades within the VPs shows that around 40% of the data is still below 0.2 Au g/t. Less than 2% of the VP data has grades exceeding 10 Au g/t. It is assumed that these elevated grades represent discrete larger gold-hosting quartz veins within the VPs.

Figure 14.2.1 Kalana Plan View Illustrating Drillhole Traces and VP Interpretation

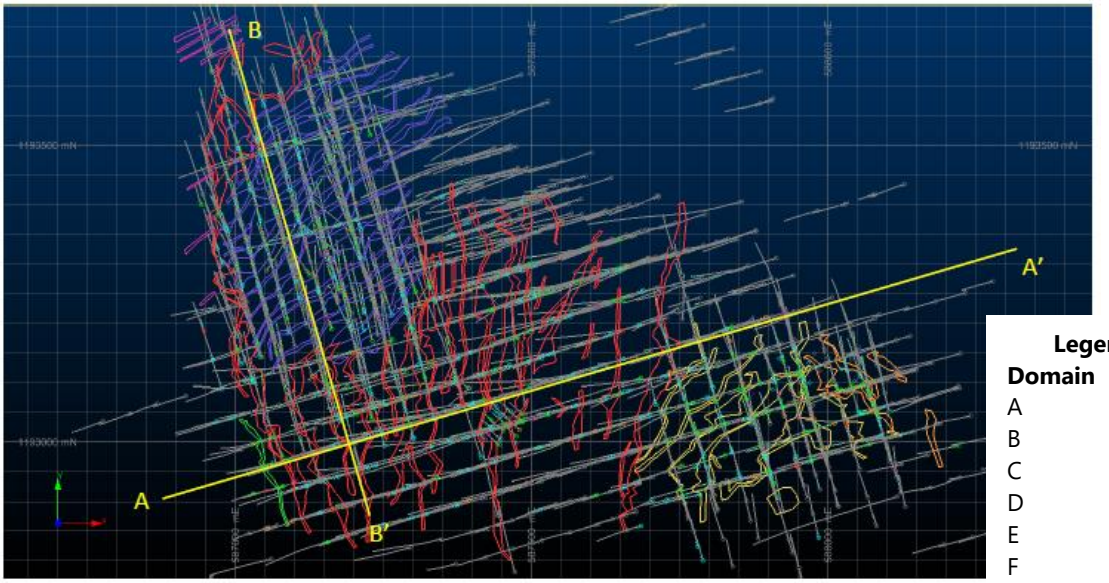


Figure 14.2.2 Kalana Cross-Section A-A' Showing Drillhole Traces, VP Interpretation, Topography and Weathering Surfaces

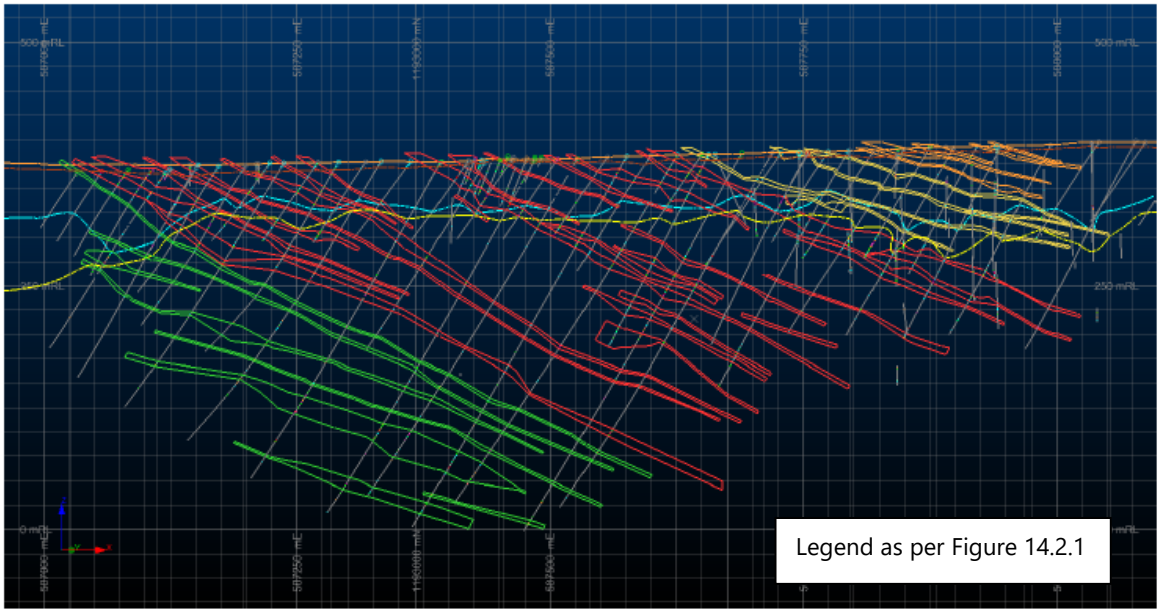
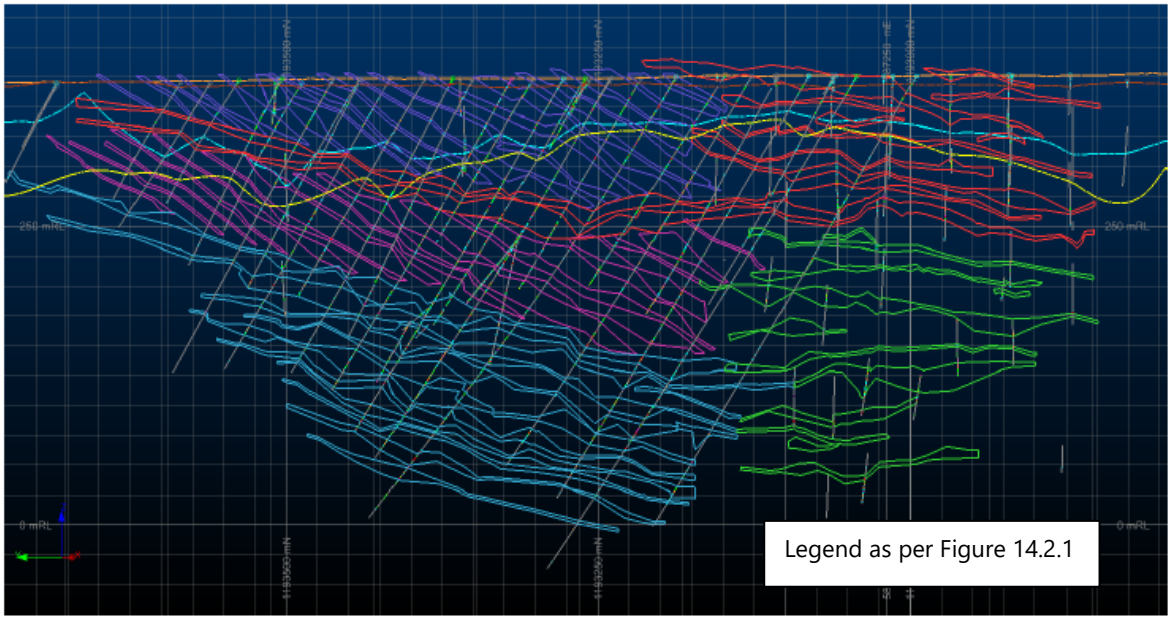
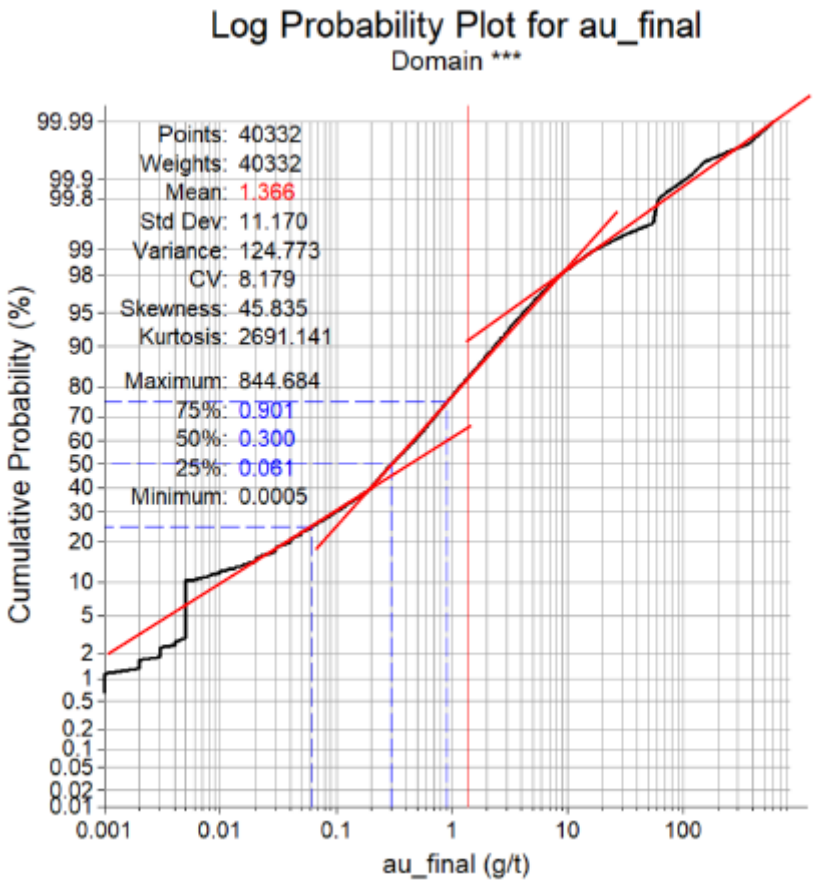


Figure 14.2.3 Kalana Cross-Section B-B' Showing Drillhole Traces, VP Interpretation, Topography and Weathering Surfaces



A review of the Kalana composited and flagged data located within the vein package interpretations shows a naïve average gold grade of 1.37 g/t and an extreme coefficient of variation (CV) that exceeds a value of eight. The grade distribution, as shown by a log-probability plot, demonstrates inflections at around 0.2 Au g/t and 10 Au g/t (Figure 14.2.4).

Figure 14.2.4 Gold Grade Distribution of Kalana Composites Contained Within the Vein Package Interpretation (Excluding the Regolith) Showing Potential Inflection Points

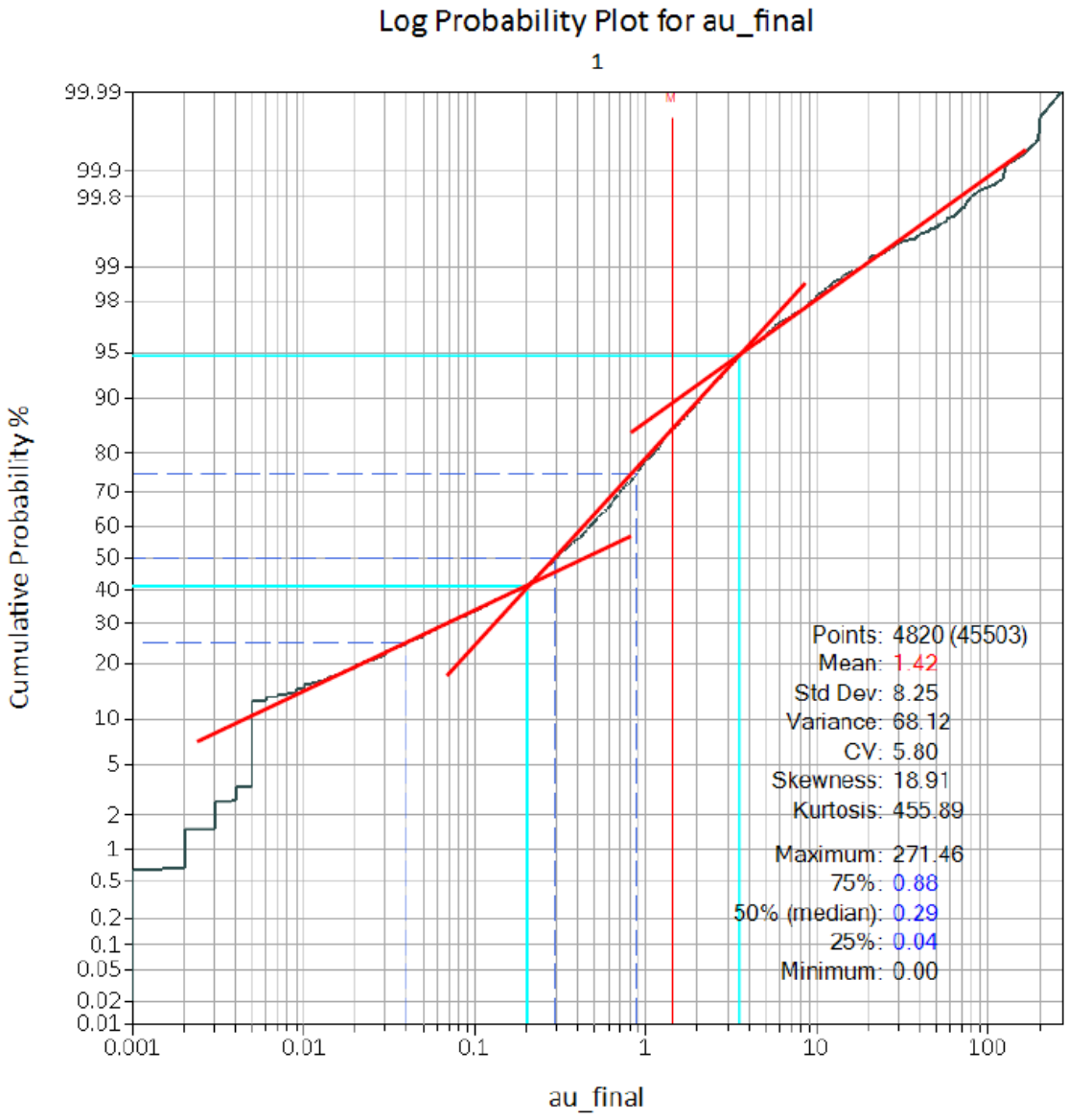


Kalanako

Visual appraisal of the Kalanako sample grades greater than 0.2 Au g/t reveals several spatial patterns within the drilling data that support the orientations of the interpreted lodes.

A review of the Kalanako composited and flagged data located within the mineralisation interpretations shows a naïve average gold grade of 1.42 g/t and an extreme CV that exceeds a value of five. The grade distribution, as shown by a log-probability plot, demonstrates inflections at around 0.2 Au g/t and 3.5 Au g/t (Figure 14.2.5).

Figure 14.2.5 Gold Grade Distribution of Kalanako Composites Contained Within the Mineralisation Lode Interpretation Showing Inflection Points Defining the Grade Categories



The Kalankao mineralisation interpretation was also compiled using a combination of criteria based on gold grade, the presence of quartz, the presence of visible gold, structural and lithological factors. Optiro understands that a nominal gold grade threshold of 0.2 g/t was applied during the lode interpretation process. While the mineralisation interpretation excludes a considerable proportion of samples with sub-0.2 g/t grades, the distribution of grades within the interpretation shows that around 41% of the data is still below 0.2 g/t.

Slightly more than 5% of the Kalanako mineralisation data has grades exceeding 3.5 Au g/t. It is assumed that these elevated grades represent discrete larger gold-hosting structures within the broader mineralisation domains.

Due to the grade distribution characteristics described above and the very high CV (variability) of the grade population within the mineralisation (VP) interpretations from both deposits, it was decided to use categorical indicator methods to divide the mineralisation volumes into grade categories based on the two grade inflection points.

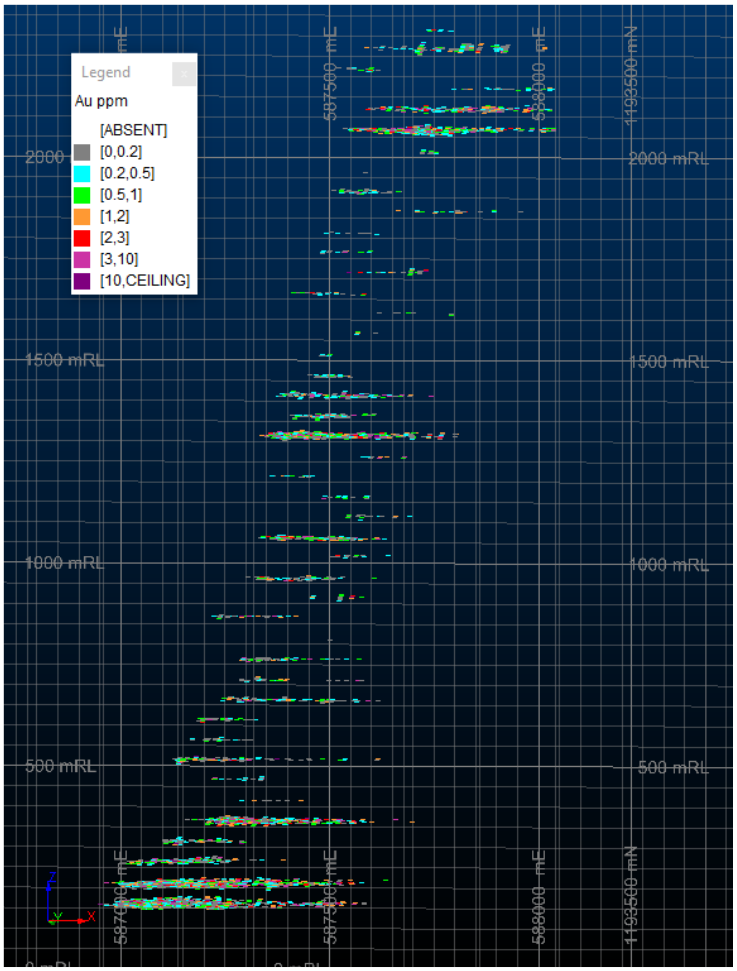
14.2.2 Estimation in Projected Space

Kalana

The geometry of the Kalana VPs led to the adoption of a projection (co-ordinate transformation) methodology as part of all estimation processes. In all cases, individual vein packages were projected onto a horizontal plane for variography analysis and block grade estimation. The process employed the centreline of each vein package as a reference and projected the centreline onto a horizontal plane using a localised vertical offset at the resolution of the smallest subcell size employed for volume modelling (5 mE by 5 mN by 1.25 mRL). The offset process was applied during block estimation to both the blocks representing a VP and the composite data located within the vein package. Thus, the X and Y co-ordinates are preserved but a local Z coordinate is established.

All variography was completed with the VP composites projected onto a horizontal plane using the same transformation process. When vein packages were combined to improve statistical support for variography within any of the structural domains (A to G), the data within each vein package was first projected to a horizontal plane. The projected data was then combined after applying a different vertical offset to each vein package as illustrated for Domain A in Figure 14.2.6. This allowed the projected data to be combined without any spatial mingling between individual vein packages. All variography analysis was then completed within the horizontal plane and vertical continuity was only evaluated over distances that did not exceed the vertical separation applied to individual vein package.

Figure 14.2.6 Kalana Domain A Composites After Flattening and Stacking for Variography Analysis



Kalanako

The geometrical patterns exhibited by the Kalanako mineralisation lodes, combined with their narrow across-plane widths compared to their strike and dip extents, led to the decision to use Datamine Studio RM's dynamic anisotropy tools during all estimation processes. The application of dynamic anisotropy means that each individual block in the model is assigned a unique dip and dip direction (± plunge) which is utilised during the estimation processes applied to each block to orientate the continuity and search ellipsoids. This contrasts with a 'normal' estimation process, whereby orientation data is fixed for all blocks within a given domain condition.

Orientation information was assigned to each block using the following process applied to each lode:

- Calculate the centreline location of the lode on a regular grid (5 m along strike, 2.5 m down dip).
- Review and remove any unwanted edge effect artefacts from the centreline data.
- Create a DTM wireframe from the centreline data.
- Calculate the dip and dip direction of each wireframe triangle in the DTM.
- Estimate the local dip and dip direction from the above orientation data for each block within the lode using a dedicated radial estimation process (which understands that the average of two bearings of 350˚ and 010˚ is a bearing of 000˚).
- Visually validate the orientation data assigned to the model blocks.

Both the categorical and grade estimation processes that follow used this localised orientation data.

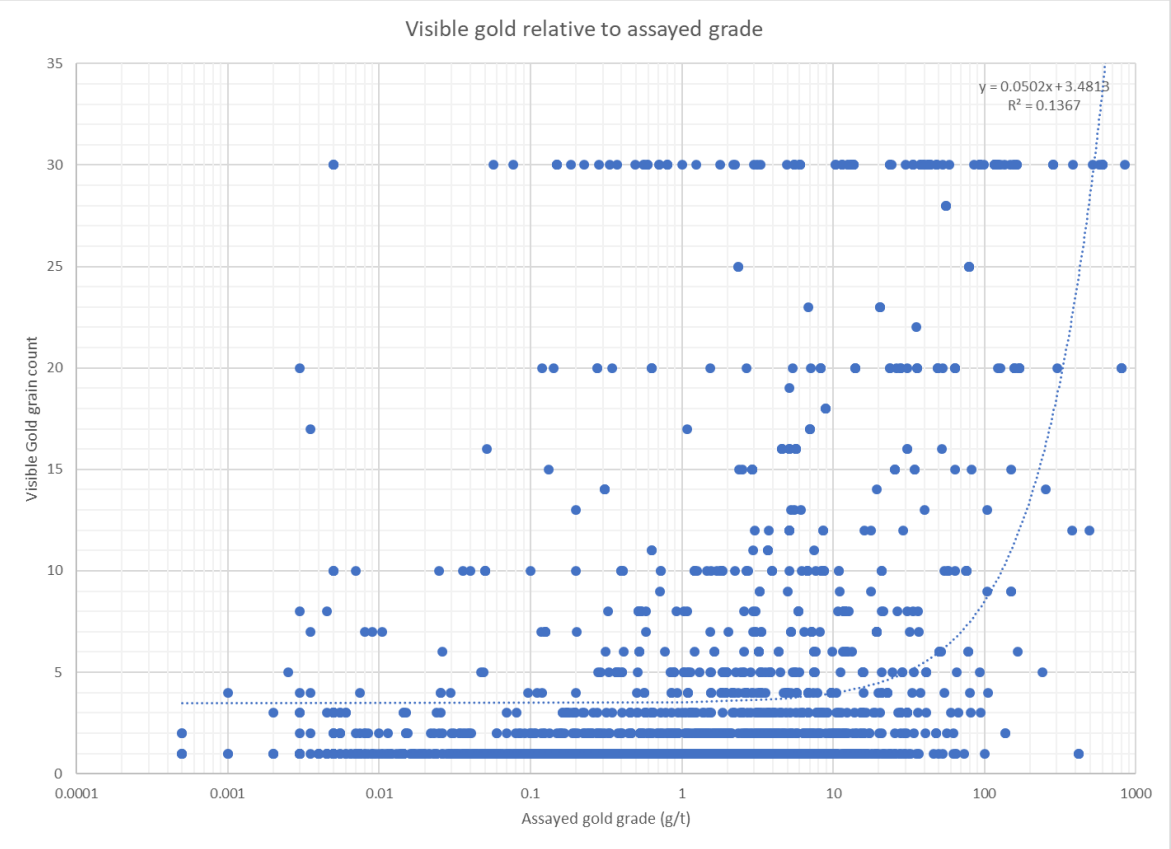
14.2.3 Sub-Domains Within Vein Packages

Categorical indicators (0/1) were created for the two grade thresholds in both deposits and spatially analysed with variograms within each of the structural domains using all the composite data contained within the vein packages.

Kalana

In addition to the applied grade threshold, any composite logged as containing visible gold was included in the 0.2 Au g/t threshold category if it had not already been assigned to either of the grade categories based on assayed grade. Analysis demonstrates that the assayed gold grade correlates very poorly with the visible gold grain count; however, it was decided that the presence of visible gold was adequate to support assignment to at least the lower grade mineralisation category, Figure 14.2.7. This choice increased the composites assigned to the lower grade threshold by 1.3%.

Figure 14.2.7 Relationship Between Visible Gold Grain Count and Assayed Gold Grade at Kalana



Indicator continuity models were successfully generated for each structural domain (as illustrated in Figure 14.2.1 to Figure 14.2.3) at the 0.2 Au g/t indicator threshold; however, grade continuity modelling at the 10 Au g/t threshold was unsuccessful due to the small proportion of data that exceeds this threshold.

The 0.2 indicator variography varied between the structural domains. The following general comments apply:

- The data variograms were not robust and required considerable interpretation.
- The nugget effect represented between 32% to 57% of the total grade variability. Typical values were around 50%.
- The first structure range defined most of the grade variability in most cases. In-plane continuity rarely exceeded 25 m for this structure. Across plane, ranges typically varied between 5 m and 10 m.

- Maximum ranges generally only represented minor components of the total grade variability. Modelled ranges could extend to several hundred metres, but these are not considered to be robust.

- Directions to the east-southeast (100° to 120°) were favoured for the best continuity, but this outcome is not considered robust.

For categorical modelling purposes, the 0.2 Au g/t threshold continuity models were also applied to the 10 Au g/t threshold to estimate the probability that a location was likely to exceed the relevant threshold grade.

The categorical estimation was conducted on 5 m by 5 m by 1.25 m blocks located within the vein packages and below the regolith surface. Each indicator threshold was estimated using ordinary kriging to provide a value in each block (between 0 and 1) representing the probability that the block exceeded either of the grade thresholds.

An iterative process was then undertaken to identify probability thresholds that identified blocks most likely to host the higher-grade conditions represented by the indicator thresholds. Once a probability threshold was selected, it was used to assign each block to one of three categories, representing grades of less than 0.2 Au g/t, between 0.2 Au g/t and 10.0 Au g/t and above 10.0 Au g/t. Once each block was assigned to a category, the associated drillhole composites were assigned to one of the three grade categories by assigning the category of the block in which they were located. If any composite with a grade greater than 10.0 Au g/t was not assigned to the highest-grade category, the probability process was overridden, and the composite was assigned to the highest-grade category. If this occurred, the block in which the composite was located was also reassigned to the highest-grade category. This physically restricted the influence of an isolated high-grade composite to a single 5 m by 5 m by 1.25 m volume, thus implying it represents a less continuous vein.

The process applied to the regolith differed slightly from the vein package sub-domaining. In this case, the vein package interpretation was not applied (even though it extended above the surface topography) as spatial grade patterns suggested a sub-horizontal control. Grade thresholds of 0.3 Au g/t and 3.0 Au g/t were selected as indicator thresholds based on grade inflections shown by the regolith composites (excluding any data located within the TSF). Thereafter, the process was the same as that applied within the vein packages, except that no projection was required, thus all variography and estimation were completed in normal 3D space. Continuity models were successfully developed for both indicator grades.

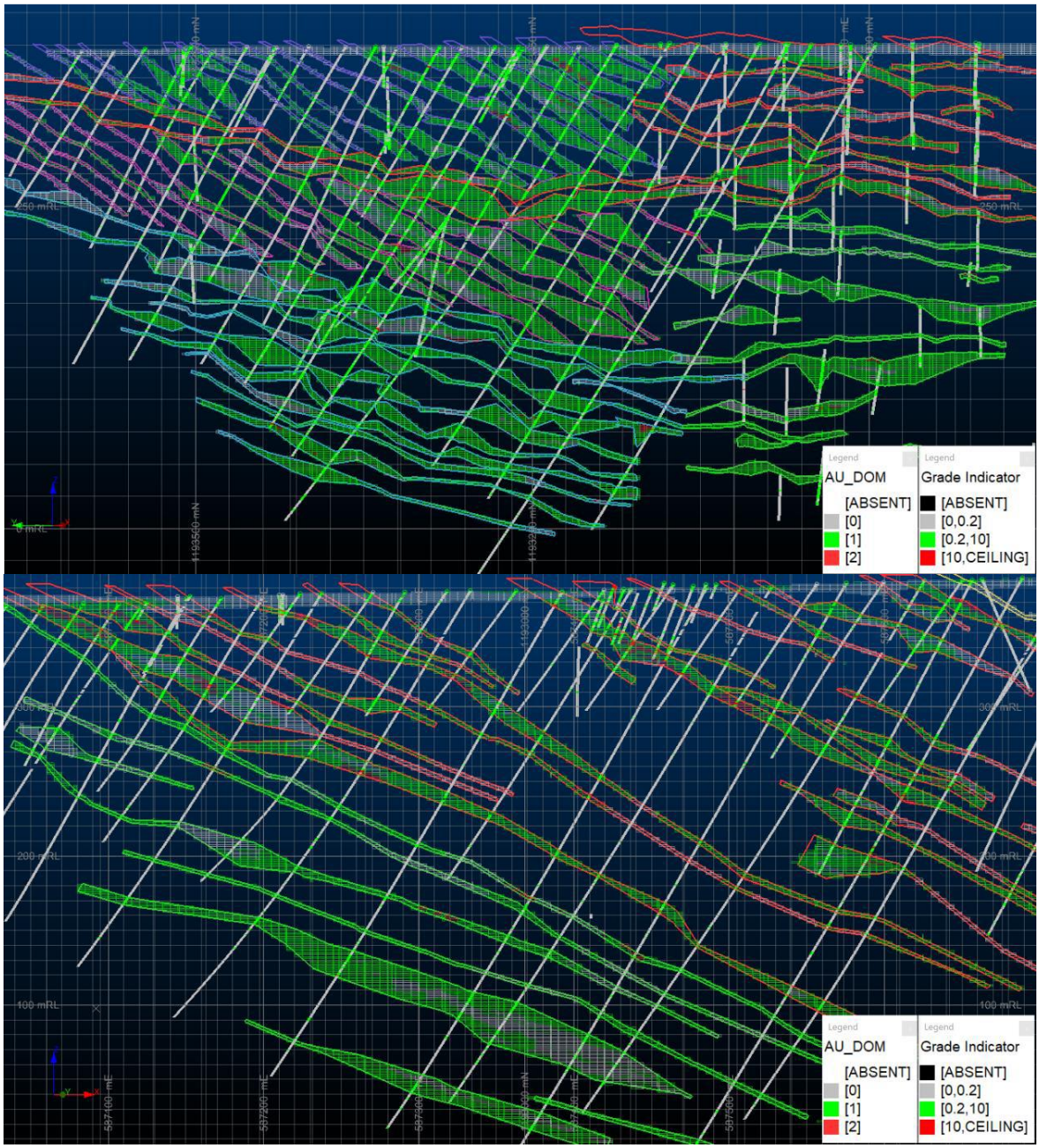
A factor impacting the regolith categorical modelling was that a number of drillholes had collar locations that placed them above the provided topography DTM. This had the impact of excluding that data from the process due to the narrow vertical search restriction applied during estimation. Optiro understands that the collar survey data provides superior resolution to that of the satellite data from which the DTM was created. This incompatibility should be resolved in future resource estimation cycles. The current situation most likely leads to a degree of conservatism in the regolith mineralisation volume.

The model and data resulting from the categorical indicator process included a code identifying the category to which each block and composite was assigned (Table 14.2-1). The product of the categorical process is illustrated by Figure 14.2.8, which shows cross sections of the drillhole data coded by gold grade (i.e. before final category assignment) and the block model coded by sub-domain category (AU_DOM).

Table 14.2-1 Kalana Indicator Based Sub-Domain Category Assignment to the Block Model and Composite Data

AU_DOM	Category	
	Vein Package	Regolith
0	0.0 to 0.2 Au g/t	0.0 to 0.3 Au g/t
1	0.2 to 10.0 Au g/t	0.3 to 3.0 Au g/t
2	>= 10.0 Au g/t	>= 3.0 Au g/t
99	Outside vein packages below regolith	

Figure 14.2.8 Kalana Example Cross Sections showing Sub-domaining of Vein packages and Regolith (A-A′ top, B-B′ bottom)



Kalanako

Indicator continuity models were successfully generated for the Kalanako mineralisation at the 0.2 Au g/t indicator threshold, where maximum ranges in the order of 120 m were detected in the mineralisation plane. Indicator continuity for the 3.5 Au g/t threshold was not as well defined by the available data, and a maximum range of 20 m was used within the mineralisation plane.

The following general comments apply:

- The across-plane variograms were not robust due to the combination of narrow across-plane geometry and the section orientation relative to the strike and dip of the mineralisation. This led to across-plane ranges being calculated from the downhole continuity by applying a true thickness calculation based on average mineralisation plane and drilling direction orientations. The geometrical relationship resulted in the across-plane ranges being approximately one third of the downhole ranges.

- The nugget component of the 0.2 Au g/t threshold represented 43% of the total indicator variability. For the 3.5 Au g/t threshold, the nugget effect represented 69% of the total indicator variability.

- The first structure range for the 0.2 Au g/t threshold of around 20 m represented around 27% of the total grade variability. The 3.5 Au g/t threshold was modelled using a single structure model. For the across-plane direction, the 0.2 Au g/t threshold range was around 15 m, while the 3.5 Au g/t threshold range was less than 2 m.

The categorical estimation was conducted on 1.25 m by 5 m by 2.5 m blocks (across strike, along strike, vertical directions) located within the mineralisation lodes. Each indicator threshold was estimated using ordinary kriging to provide a value in each block (between 0 and 1) representing the probability that the block exceeded either of the grade thresholds.

An iterative process was then undertaken to identify probability thresholds that identified blocks most likely to host the higher-grade conditions represented by the indicator thresholds. Once a probability threshold was selected, it was used to assign each block to one of three categories, representing grades of less than 0.2 Au g/t, between 0.2 Au g/t and 3.5 Au g/t and above 3.5 Au g/t. Once each block was assigned to a category, the associated drillhole composites were assigned to one of the three grade categories by allocating the category of the block in which the composites were located. If any composite with a grade greater than 3.5 Au g/t was not assigned to the highest-grade category, the probability process was overridden, and the composite was assigned to the highest-grade category. If this occurred, the block in which the composite was located was also reassigned to the highest- grade category. This physically restricted the influence of an isolated high-grade composite to a single 1.25 m by 5 m by 2.5 m volume, thus implying that it represents a less continuous vein.

The model and data resulting from the categorical indicator process included a code identifying the category to which each block and composite was assigned (Table 14.2-2).

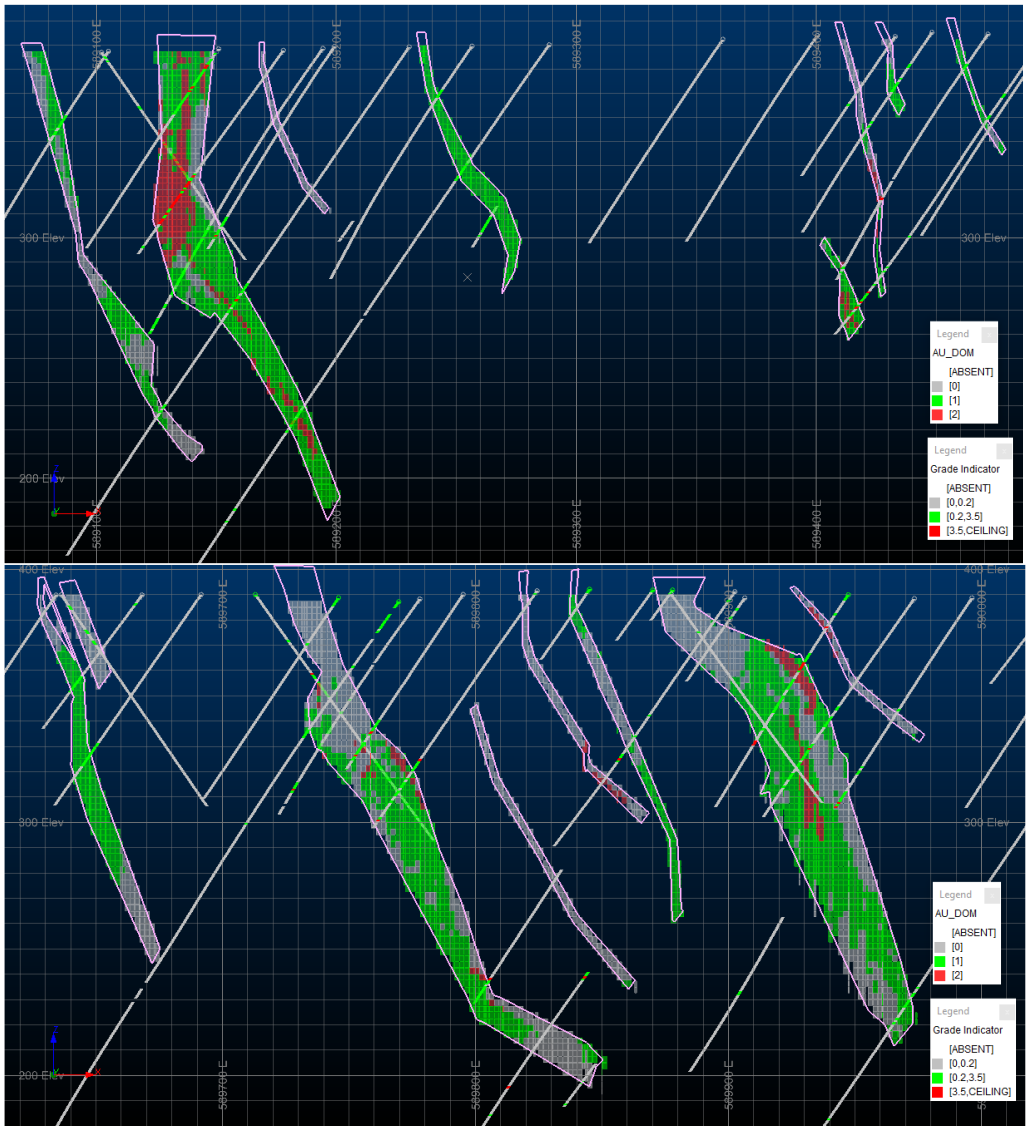
The product of the categorical process is illustrated by

Figure 14.2.9 which shows cross sections of the drillhole data coded by gold grade (i.e. before final category assignment) and the block model coded by sub-domain category (AU_DOM).

Table 14.2-2 Indicator Based Sub-domain Category Assignment to the Kalanako Block Model and Composite Data

AU_DOM	Grade Category
0	0.0 to 0.2 Au g/t
1	0.2 to 3.5 Au g/t
2	>= 3.5 Au g/t

Figure 14.2.9 Example Cross Sections Showing Sub-Domaining Of Kalanako Mineralisation Lodes (1,194,900 mN Top, 1,195,250 mN Bottom)



14.2.4 Gold Grade Estimation

After completion of the categorical processes used to generate the vein package sub-domains, the grade estimation process followed a traditional path. The analysis process that was followed utilised the data sub-divisions provided by the structural domains, vein packages and sub-domains. Grade statistics and continuity were assessed by structural domain and by sub-domain for the data formed by combining the individual structural domain vein packages.

Kalana Grade Statistics and Top-Cuts

Un-cut and top-cut, declustered gold grade statistics are provided in Table 14.2-3 and Table 14.2-4, respectively, and example log-histogram plots are presented for Domain A prior to top-cuts being applied in Figure 14.2.10. Declustering was applied by structural domain and vein package sub-domain.

Top-cuts were determined using a grade population disintegration method that targets high grade outliers in the grade distribution. The following broad comments can be made regarding top-cutting:

- Top-cuts of between 1 Au g/t and 4 Au g/t were applied to the lowest grade category (AU_DOM=0).
- No top-cuts were applied to the middle grade category (AU_DOM=1).
- Top-cuts for the highest-grade category (AU_DOM=2) were generally set at 150 Au g/t although one structural domain was not top-cut and two others had top-cuts of 65 Au g/t and 120 Au g/t applied.

After top-cutting, CVs were acceptable for the application of ordinary kriging for grade estimation; however, the CVs for the lowest grade sub-domain remained high. This factor was ignored largely as the average grade was low at 0.11 Au g/t to 0.15 Au g/t depending on structural domain.

In terms of number of composites, Domain A is significantly larger than the other structural domains. Domains E, F and G are the next largest and contain roughly one-third of the composites of domain A. Domains B, C, and D contain between 13% and 22% of the data contained in Domain A.

The middle grade category (AU_DOM=1) comprises 52% to 67% of the total vein package data depending on structural domain. Domain A has the highest proportion in this category and also one of the highest average grades for this category. Average grades in this category vary from 0.91 Au g/t to 1.21 Au g/t.

The highest-grade category contains between one and four percent of the composites contained within each structural domain. The highest proportion is hosted by Domain A followed by Domain B. Average grades vary between 17.26 Au g/t and 23.16 Au g/t.

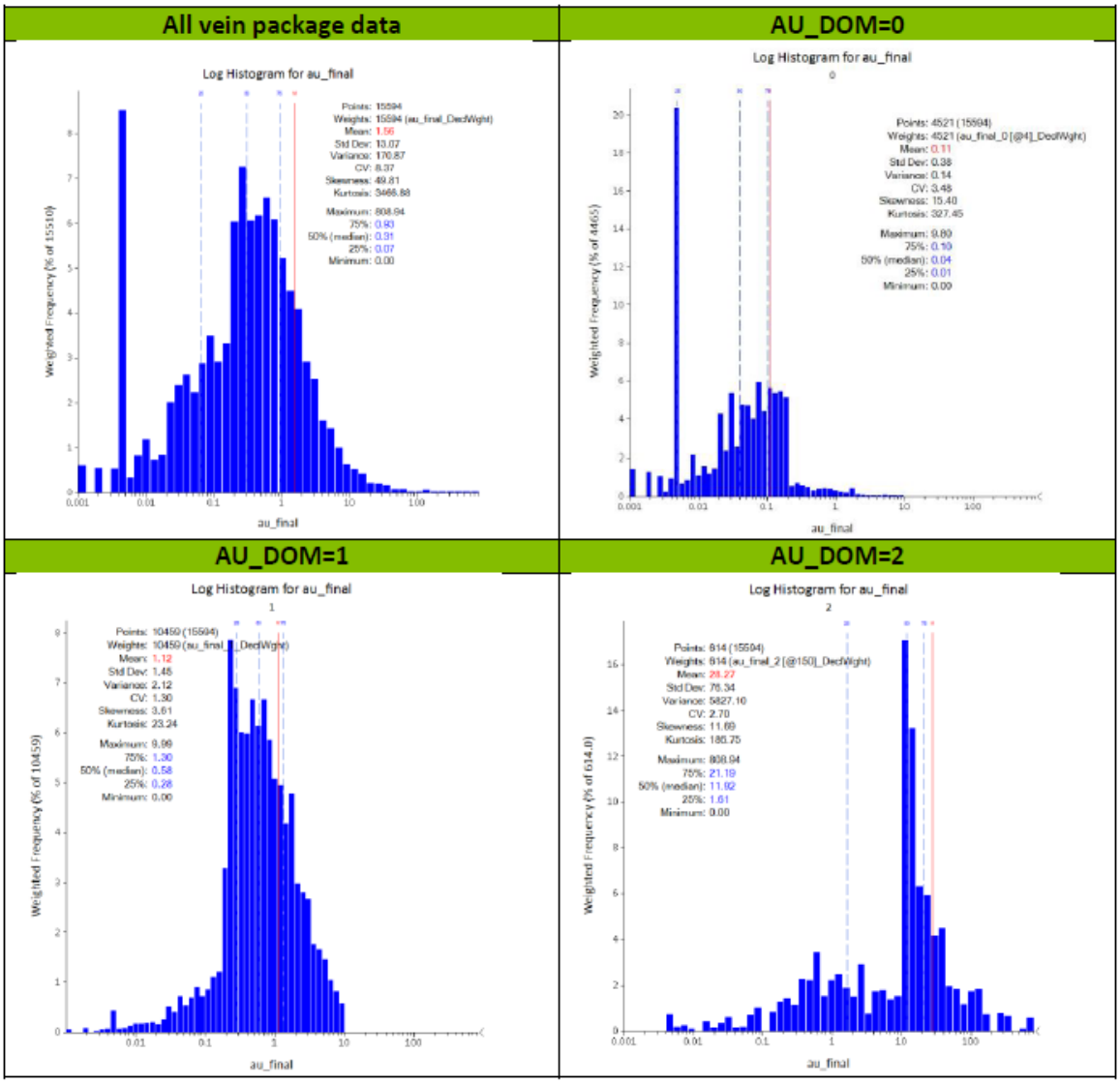
Table 14.2-3 Kalana Vein Package Declustered, Top-Cut Gold Grade Statistics

Region		A			B			C			D			E			F			G		
Sub-domain (AU_DOM)		**0**	**1**	**2**	**0**	**1**	**2**	**0**	**1**	**2**	**0**	**1**	**2**	**0**	**1**	**2**	**0**	**1**	**2**	**0**	**1**	**2**
Samples		4,521	10,459	614	853	1,156	55	1,608	1,807	34	934	1,466	59	1,665	3,507	122	1,845	3,669	126	2,113	3,618	70
Number top-cut		9	0	14	20	0	2	18	0	0	12	0	1	8	0	2	15	0	2	14	0	2
Minimum		0.0005	0.0005	0.0005	0.0005	0.01	0.03	0.0005	0.0005	0.3	0.0005	0.0005	0.02	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005
Maximum		4.00	9.99	150.00	1.00	9.35	150.00	2.00	9.80	79.84	1.50	9.96	150.00	3.00	9.70	150.00	2.00	9.93	150.00	3.00	9.74	120.00
Mean		**0.10**	**1.12**	**21.53**	**0.09**	**1.09**	**20.59**	**0.12**	**0.92**	**17.26**	**0.09**	**1.21**	**23.16**	**0.10**	**1.11**	**21.63**	**0.11**	**1.01**	**19.97**	**0.12**	**0.91**	**23.09**
Standard deviation		0.30	1.45	32.83	0.16	1.47	28.92	0.29	1.25	20.39	0.21	1.51	27.86	0.29	1.47	31.85	0.26	1.33	25.29	0.34	1.13	32.14
CV		2.89	1.30	1.52	1.76	1.34	1.40	2.48	1.36	1.18	2.36	1.25	1.20	2.97	1.33	1.47	2.44	1.32	1.27	2.84	1.24	1.39
Variance		0.09	2.12	1077.54	0.03	2.15	836.22	0.08	1.56	415.75	0.04	2.27	776.42	0.08	2.16	1014.35	0.07	1.76	639.72	0.12	1.28	1033.19
Percentile	10%	0.01	0.18	0.33	0.01	0.21	0.34	0.01	0.16	0.43	0.01	0.13	0.18	0.00	0.11	0.15	0.01	0.20	0.51	0.00	0.13	0.06
	20%	0.01	0.25	0.92	0.01	0.25	4.01	0.01	0.24	1.40	0.01	0.25	0.98	0.01	0.23	0.89	0.01	0.24	1.72	0.01	0.24	0.57
	30%	0.01	0.32	2.89	0.02	0.32	11.00	0.01	0.30	4.64	0.01	0.35	3.52	0.01	0.31	4.65	0.01	0.31	2.91	0.01	0.30	1.29
	40%	0.02	0.44	10.32	0.04	0.42	12.04	0.02	0.36	9.32	0.01	0.48	10.60	0.01	0.44	10.76	0.02	0.41	10.52	0.01	0.42	10.81
	50%	0.04	0.58	11.92	0.04	0.53	12.64	0.03	0.46	11.41	0.02	0.62	15.87	0.02	0.57	12.04	0.04	0.54	12.30	0.03	0.55	12.55
	60%	0.06	0.77	14.38	0.06	0.65	13.56	0.05	0.62	12.73	0.05	0.84	22.72	0.04	0.74	13.86	0.06	0.72	16.09	0.05	0.70	15.68
	70%	0.09	1.09	17.49	0.09	0.95	17.87	0.08	0.84	13.55	0.07	1.28	28.05	0.06	1.05	16.85	0.08	0.96	21.65	0.08	0.91	19.77
	80%	0.12	1.62	26.50	0.13	1.51	20.20	0.12	1.29	24.39	0.11	1.89	31.96	0.11	1.61	29.87	0.12	1.35	29.49	0.13	1.31	30.93
	90%	0.18	2.75	52.04	0.17	2.80	35.14	0.20	2.17	45.90	0.17	2.95	52.38	0.17	2.83	52.31	0.18	2.34	47.02	0.19	2.09	62.13
	95%	0.26	4.18	105.79	0.30	4.29	68.52	0.43	3.17	55.91	0.31	4.55	71.73	0.32	4.21	78.51	0.33	3.81	63.71	0.43	3.23	112.94
	97.50%	0.65	5.69	150.00	0.67	5.86	138.46	1.06	4.62	76.84	0.85	5.62	92.64	0.81	5.70	135.47	0.87	4.98	93.58	0.97	4.31	120.00
	99%	1.59	7.60	150.00	1.00	7.38	150.00	2.00	7.03	78.64	1.40	7.49	129.56	1.49	7.64	150.00	1.84	6.82	112.08	2.08	6.10	120.00

Table 14.2-4 Kalana Composite Data Top-Cuts Applied (Naïve Grade Statistics Reported)

Domain	Sub-domain (AU_DOM)	Num Samples	Uncut Data			Top-cut Data					% Difference	
			Max	Mean	CV	Num Cut	Top-Cut	Max	Mean	CV	Mean	CV
A	0	4,532	9.80	0.11	3.34	9	4.00	4.00	0.11	2.81	-3.9%	-15.9%
	1	10,468	9.99	1.21	1.28	0	N/A	9.99	1.21	1.28	0.0%	0.0%
	2	614	808.94	23.90	2.73	14	150.00	150.00	18.98	1.62	-20.6%	-40.8%
B	0	853	9.92	0.15	4.32	20	1.00	1.00	0.10	1.82	-35.9%	-57.8%
	1	1,157	9.35	1.18	1.28	0	N/A	9.35	1.18	1.28	0.0%	0.0%
	2	55	495.51	28.41	2.49	2	150.00	150.00	22.12	1.55	-22.2%	-37.6%
C	0	1,609	9.90	0.14	3.85	18	2.00	2.00	0.11	2.49	-16.7%	-35.3%
	1	1,808	9.80	0.94	1.38	0	N/A	9.80	0.94	1.38	0.0%	0.0%
	2	34	79.84	15.28	1.29	0	N/A	79.84	15.28	1.29	0.0%	0.0%
D	0	934	7.25	0.13	3.93	12	1.50	1.50	0.10	2.46	-21.5%	-37.5%
	1	1,468	9.96	1.15	1.29	0	N/A	9.96	1.15	1.29	0.0%	0.0%
	2	59	366.10	24.29	2.09	1	150.00	150.00	20.63	1.39	-15.1%	-33.4%
E	0	1,665	8.13	0.12	3.94	8	3.00	3.00	0.11	2.98	-10.6%	-24.4%
	1	3,508	9.70	1.03	1.31	0	N/A	9.70	1.03	1.31	0.0%	0.0%
	2	122	600.46	23.30	2.87	2	150.00	150.00	17.41	1.50	-25.3%	-48.0%
F	0	1,845	6.74	0.12	3.45	15	2.00	2.00	0.10	2.46	-13.2%	-28.6%
	1	3,671	9.93	1.01	1.28	0	N/A	9.93	1.01	1.28	0.0%	0.0%
	2	126	844.68	23.82	3.28	2	150.00	150.00	18.19	1.55	-23.6%	-52.9%
G	0	2,115	5.62	0.14	3.09	14	3.00	3.00	0.13	2.74	-5.2%	-11.3%
	1	3,619	9.74	0.91	1.28	0	N/A	9.74	0.91	1.28	0.0%	0.0%
	2	70	170.72	19.07	1.68	2	120.00	120.00	17.83	1.49	-6.5%	-10.8%
Regolith	0	5,733	2.95	0.12	1.39	17	1.50	1.50	0.12	1.21	-1.7%	-13.1%
	1	1,128	2.99	0.72	0.78	0	N/A	2.99	0.72	0.78	0.0%	0.0%
	2	99	574.05	16.24	3.60	5	65.00	65.00	10.78	1.33	-33.6%	-63.0%

Figure 14.2.10 Gold Grade Log Histogram Graphs For Kalana Domain A Vein Packages – All Data And By Sub-Domains (AU_DOM)



Kalanako Grade Statistics and Top-Cuts

Kalanako declustering was applied within the mineralised data using a fixed 25 m by 50 m by 50 m cell size.

Top-cuts were determined using a grade population disintegration method that targets high grade outliers in the grade distribution (Table 14.2-5). Top-cuts were applied to the lowest and highest grade sub-domains. No top-cutting was required within the middle grade domain. After top-cutting, the CV remained high for the lowest and highest-grade sub-domains (Table 14.2-6). The planned grade estimation method was ordinary kriging and this magnitude of CV is not optimal for ordinary kriging. This high variability issue was mitigated by the very low-grade nature of the lowest grade sub-domain, which is not material to the reported resource and, within the highest-grade sub-domain, by the volumetric constraint provided by the categorical domaining process.

Based on the distribution of composites between the sub-domains, the lowest grade sub-domain comprises 37% of the total mineralised data set, the middle sub-domain contains 52%, while the highest-grade sub-domain contains 10% of the data. This implies that a considerable portion of the interpreted mineralisation volume will have estimated grades less than 0.2 g/t Au.

Log-histogram plots were generated for the mineralised sample data prior to top-cuts being applied (Figure 14.2.11); log-probability plots are displayed in Figure 14.2.12. These demonstrate the degree of separation achieved by the categorical process, which has an underlying resolution related to the block size (1.25 mE by 5 mN by 2.5 mRL) utilised for the process. For example, around 25% of the data in the highest-grade sub-domain contains gold grades less than 3.5 g/t. This is largely a consequence of the generally narrow higher-grade intercepts and the intermixing downhole of higher and lower grades, as well as the resolution provided by the block size. In contrast, when the middle grade sub-domain grade distribution is reviewed, only around 11% of the included composites have grades less than 0.2 Au g/t. This largely reflects the thicker drillhole intercepts with grades in this category. This degree of misclassification is not material.

Table 14.2-5 Kalanako Mineralisation Sub-Domains Declustered, Top-Cut Gold Grade Statistics

Statistic		AU_DOM		
		0	1	2
Samples		1,796	2,529	495
Minimum		0.001	0.002	0.001
Maximum		1.00	3.49	160.00
Mean		0.07	0.73	9.01
Variance		0.02	0.46	469.87
Standard Deviation		0.15	0.67	21.68
CV		2.07	0.92	2.41
Percentile	5%	0.003	0.077	0.084
	10%	0.005	0.173	0.293

	20%	0.005	0.251	0.679
	25%	0.005	0.288	0.869
	30%	0.005	0.331	1.131
	40%	0.01	0.437	2.018
	50%	0.02	0.561	3.645
	60%	0.035	0.714	4.666
	70%	0.06	0.929	5.833
	75%	0.075	1.06	7.321
	80%	0.095	1.259	9.114
	90%	0.17	1.892	17.891
	95%	0.318	2.376	38.454

Table 14.2-6 Kalanako Composite Data Top-Cuts Applied (Naïve Grade Statistics Reported)

Domain	Num. Sams	Uncut Data			Top-Cut Data					% Difference	
		Max	Mean	CV	Num Cut	Top-Cut	Max	Mean	CV	Mean	CV
0	1,796	3.45	0.08	2.63	15	1.00	1.00	0.07	2.02	-8.1%	-23.1%
1	2,529	3.49	0.80	0.89	0	3.49	3.49	0.89	0.89	0.0%	0.0%
2	495	271.46	9.45	2.57	4	160.00	160.00	9.06	2.31	-4.2%	-10.0%

Figure 14.2.11 Kalanako Mineralisation Gold Grade Log Histogram Graphs – All Data and by Sub-Domain (AU_DOM)

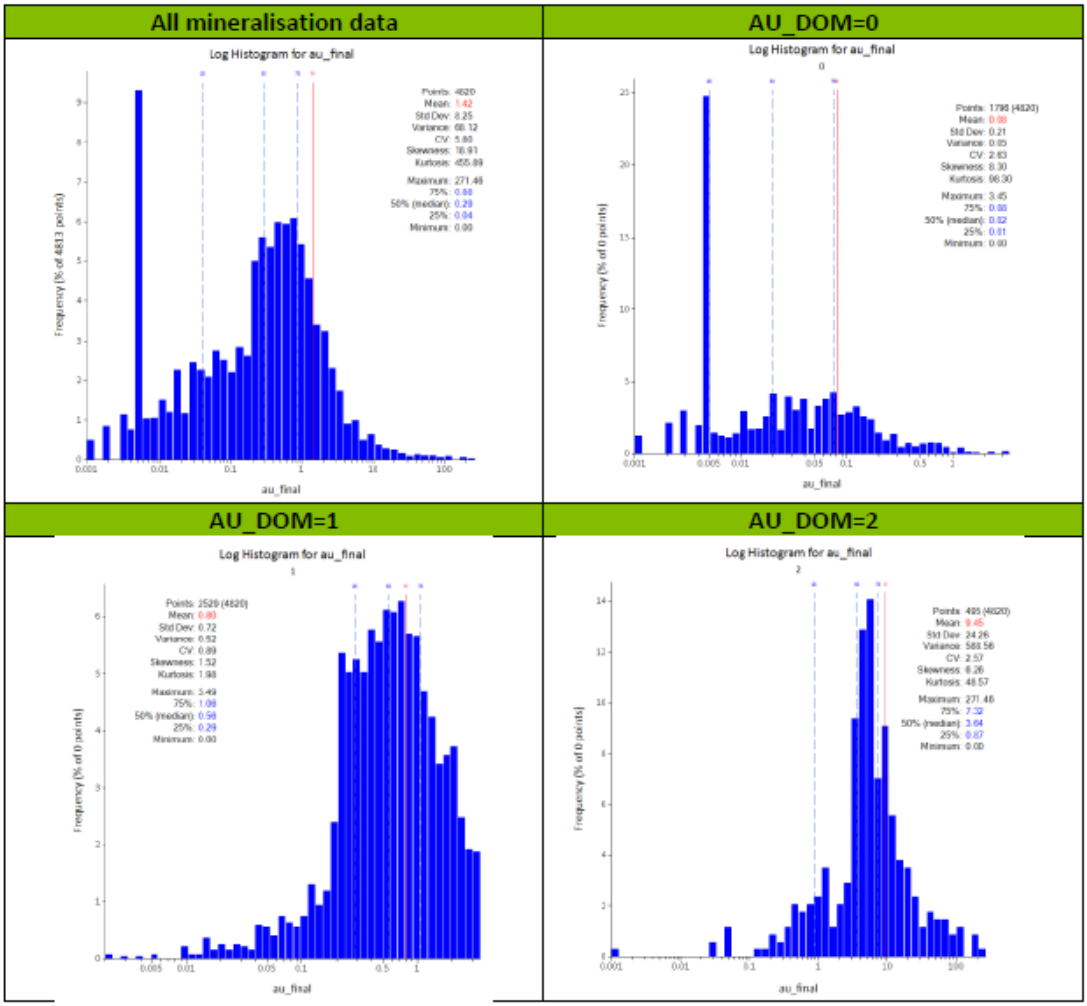
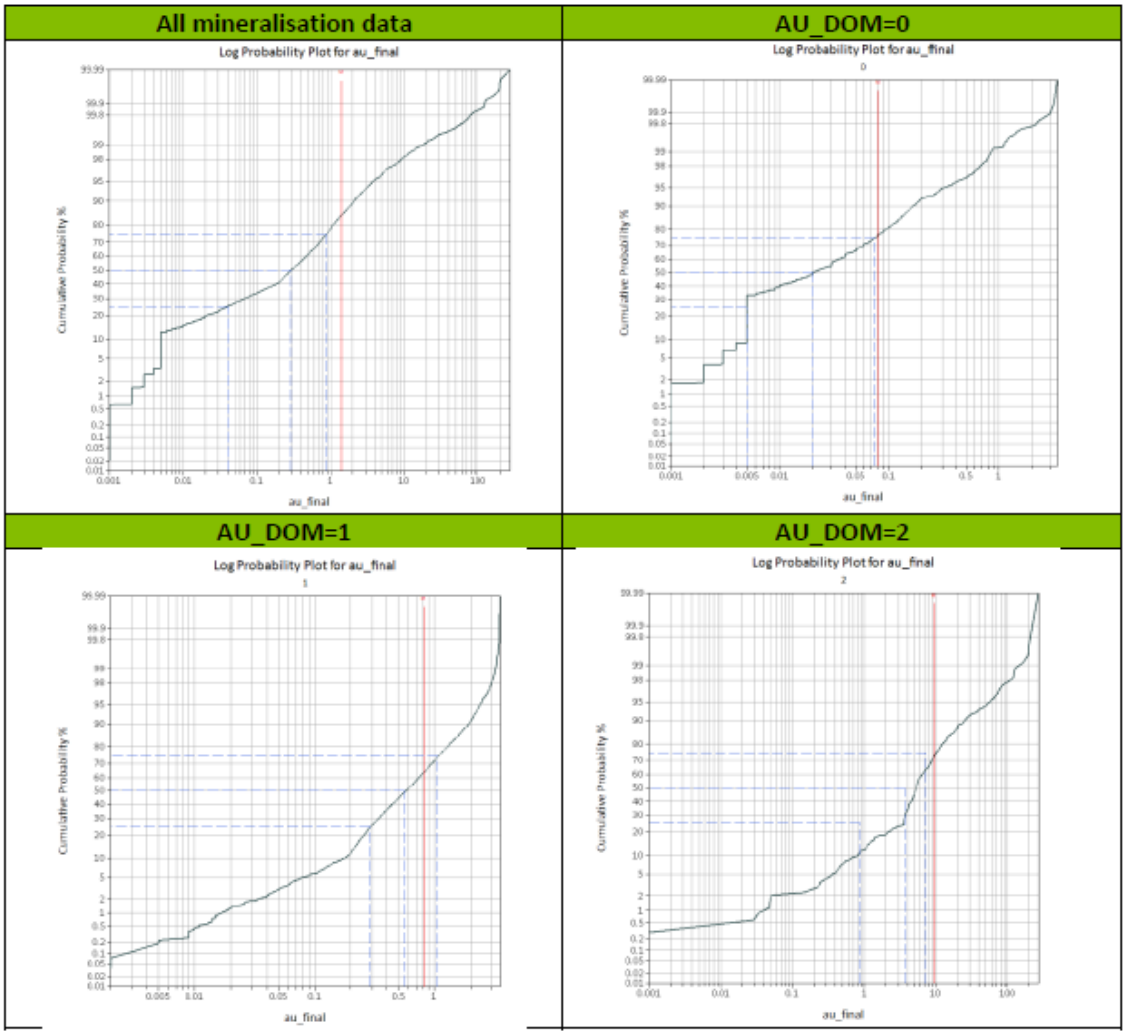


Figure 14.2.12 Kalanako Mineralisation Gold Grade Log-Probability Graphs – All Data and by Sub-Domains (AU_DOM)



14.2.5 Grade Continuity

Gold grade continuity was assessed for the middle and high-grade sub-domains within each structural domain using normal score variograms. The amount of data in the highest-grade sub-domain (AU_DOM=2) was insufficient to support continuity modelling, thus the analysis concentrated on the middle grade category (AU_DOM=1). Even within this category, grade continuity was not well-defined.

Kalana

Based on nominal drillhole spacings of 25 m and 50 m at Kalana, a typical scenario during continuity modelling within the vein package plane was to have a variogram data point at a distance of 20 m that was slightly below the sill of the variogram, while the next point at 50 m was on the sill. This indicates that a small component of the grade continuity was defined at 25 m while no grade continuity remains at 50 m.

An example of the experimental variograms and the fitted spherical models is provided for AU_DOM=1 from Domain A in Figure 14.2.13. The back-transformed continuity models for all AU_DOM=1 sub-domains are provided in Table 14.2-7.

The grade continuity models all have nugget effect components that represent around two-thirds of the total grade variability. This indicates a mineralisation style that includes a large component of random gold grade variability - a feature that is expected of gold deposits that contain a lot of coarse, visible gold.

All structural domains, with the exception of the steeper dipping F and G Domains, were modelled with a preferred primary axis direction towards the southeast. The variography analysis provided weak and variable support for this direction; however, it was maintained as it had also been observed during the categorical indicator analysis.

Variography for Domains F and G provided no directional guidance, so the east and north directions were modelled. The maximum continuity ranges in these two domains were less than the other domains, even though they tended to be covered by closer-spaced drilling. Moreover, the data does not preclude the possibility that the nugget component is much higher than represented by the fitted models. The presence of the sub-vertical veining in these domains may be adversely impacting the continuity analysis.

It is possible that more close-spaced drilling in the other structural domains would result in shorter range continuity models.

Figure 14.2.13 Gold Grade Continuity Experimental Variograms And Fitted Spherical Models For Kalana Domain A, AU_DOM=1

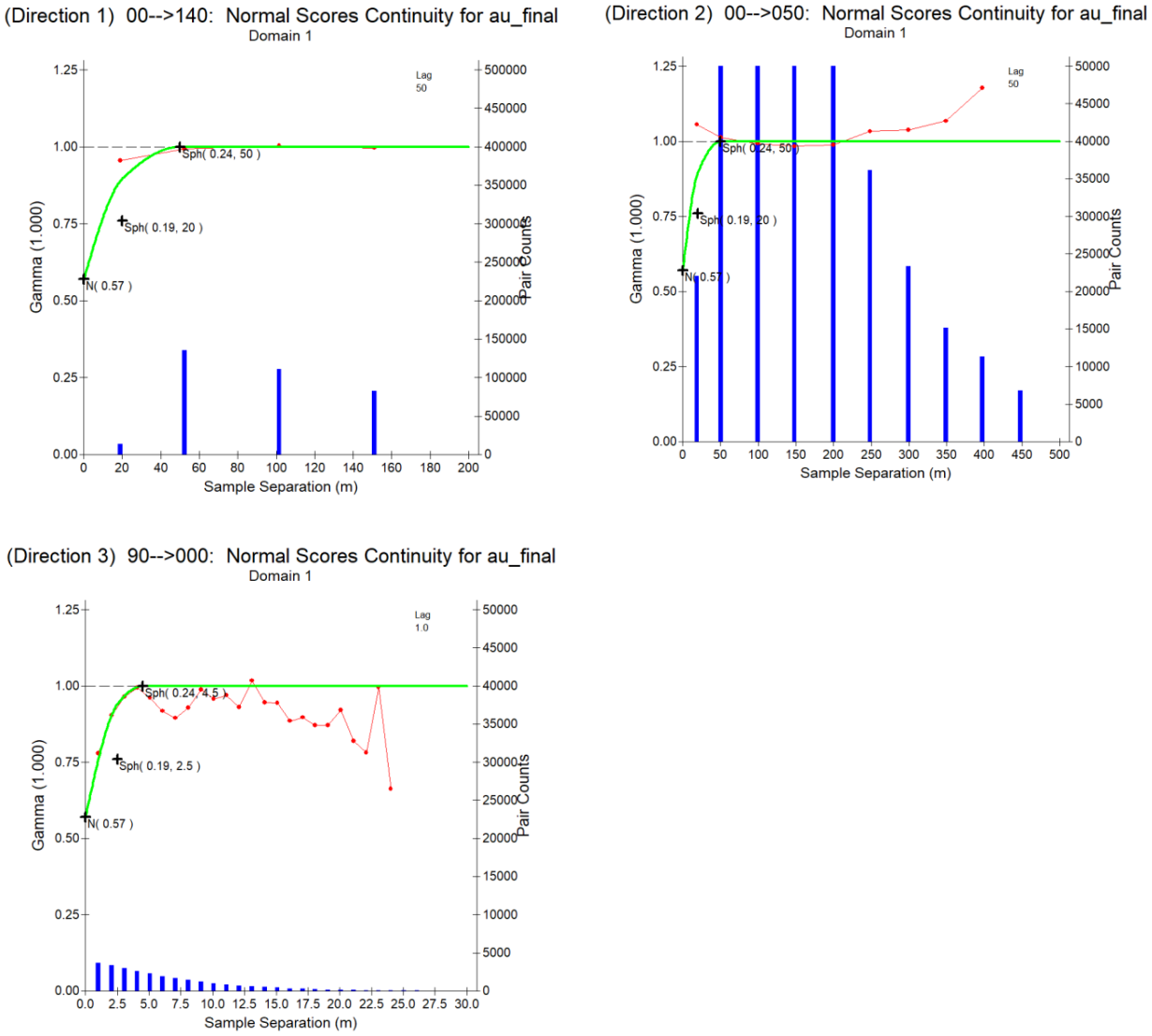


Table 14.2-7 Fitted Back-Transformed Spherical Parameter Grade Continuity Models For Kalana AU_DOM=1

Domain	Direction	C_0	C_1	A_1	C_2	A_2
A	00 – 140	0.65	0.18	20	0.17	50
	00 – 050			20		50
	Vertical			2.5		4.5
B	00 – 130	0.65	0.21	20	0.14	50
	00 – 040			20		50
	Vertical			3		3
C	00 – 120	0.68	0.24	20	0.08	50
	00 – 030			20		50
	Vertical			3		3
D	00 -120	0.65	0.17	20	0.18	50
	00 – 0.30			20		75
	Vertical			3		3
E	00 – 120	0.66	0.19	20	0.15	70
	00 – 030			20		100
	Vertical			4		4
F	00 – 000	0.66	0.20	15	0.14	15
	00 – 270			15		15
	Vertical			1		3
G	00 – 000	0.66	0.27	8	0.07	20
	00 – 270			8		20
	Vertical			1		3

Kalanako

Based on the nominal drill hole spacing along strike of around 40 m and slightly shorter down dip at Kalanako, only a single variogram data point was below the sill of the variogram, while the next point was on the sill. This indicates that grades are likely to vary considerably between drill holes on the current drill grid, and that the modelling process will be unable to fully account for the grade variability between sample data.

The experimental variograms and the fitted spherical models are presented for AU_DOM=1 in Figure 14.2.14. The across-plane continuity model was calculated from the downhole variogram and is also presented in Figure 14.2.14. The continuity models for all AU_DOM=1 sub-domains are shown in Table 14.2.8.

Figure 14.2.14 Kalanako Gold Grade Continuity Experimental Variograms and Downhole Variogram with Fitted Spherical Models for AU_DOM=1

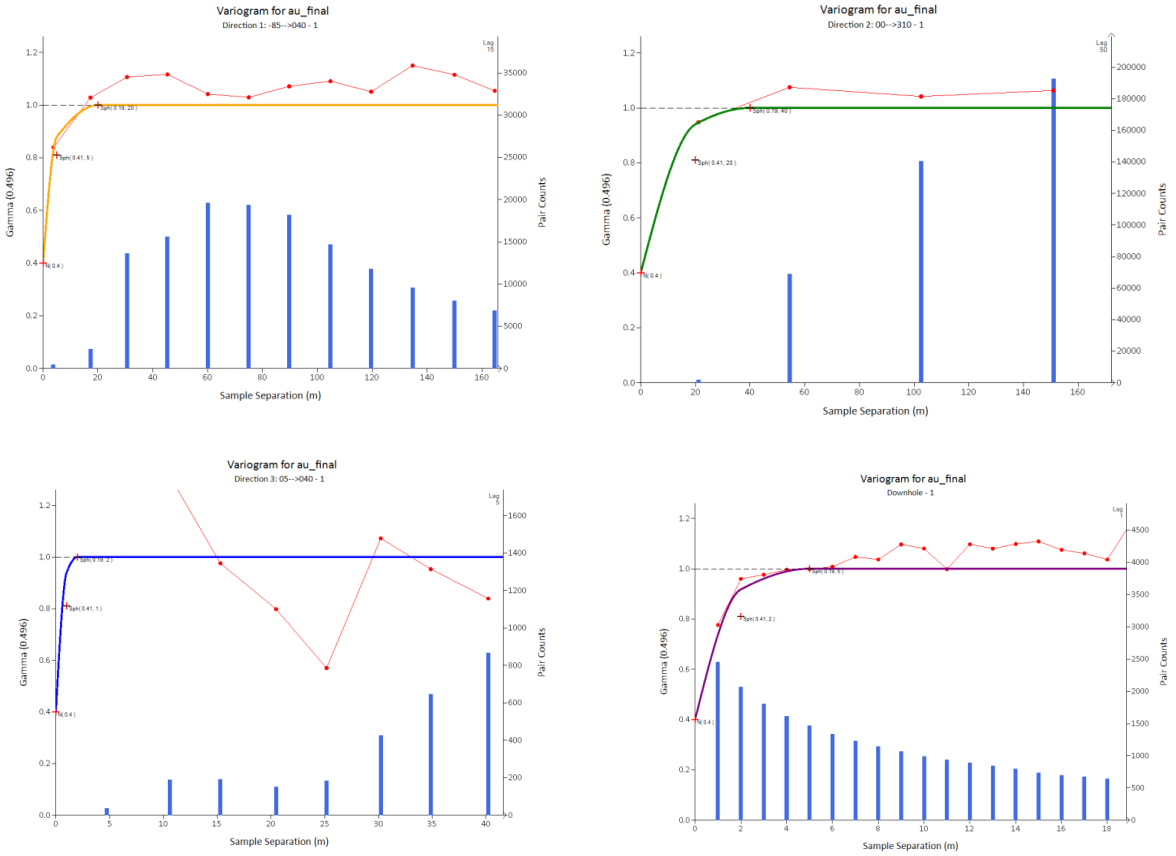


Table 14.2-8 Fitted Spherical Grade Continuity Models for Kalanako AU_DOM

Direction	C_0	C_1	A_1	C_2	A_2
-85° – 040°	0.40	0.41	5	0.19	20
00° – 310°			20		40

The grade continuity model has a nugget effect component that represents 40% of the total grade variability, which represents a moderate component of inherently random behaviour within the middle grade sub-domain. The grade continuity ranges are relatively short and are arguably dictated by the drillhole spacing separation rather than continuity characteristics that are inherent to the mineralisation. It would be prudent to undertake some close spaced drilling within a 'typical' region of the deposit to improve the definition of grade continuity and to improve confidence in modelling the continuity of grade. Any such drilling should be based on section lines that are perpendicular to the overall deposit strike and separated by no more than 20 m.

14.2.6 Kriging Neighbourhood Analysis

A basic kriging neighbourhood analysis (KNA) was completed on the vein package data located within Kalana Domain A and the Kalanako middle grade domain. The Kalana KNA did not reveal high quality estimation conditions due to the combined impact of high nugget effect and short grade continuity ranges, combined with the relatively narrow, tabular presentation of the vein packages. The Kalanako KNA performance was not expected to reveal high quality estimation conditions due to the combined impact of the high nugget variance and the short grade continuity ranges, combined with the relatively narrow, tabular presentation of the mineralisation lodes. The analysis confirmed this expectation.

The block size related behaviour (Figure 14.2.15 and Figure 14.2.16) of the Slope of Regression (SoR) and Kriging Efficiency (KE) suggests that smaller blocks provided better relative performance than larger block configuration; however, the absolute performance of all block sizes tested was poor at Kalana and very poor at Kalanako, which indicates that conditional bias will be present in the grade estimates. A parent block size of 10 mE by 10 mN by 5 mRL was selected on this basis for Kalana. A parent block size of 2.5 mE by 20 mN by 10 mRL was selected at Kalanako, mainly on the basis that this roughly reflected half the drill grid spacing in the mineralisation plane and catered for the relatively narrow lode configuration across the mineralisation plane.

KNA for the number of sample testing indicated that using between 12 and 30 data points at Kalana and between 10 and 20 at Kalanako provided reasonable relative performance (Figure 14.2.17 and Figure 14.2.18). The assessment of search radii suggested that estimation performance declined as search ranges increased at Kalana (Figure 14.2.19). The assessment of search volumes suggested that estimation performance was relatively insensitive to the search criteria applied at Kalanako (Figure 14.2.20).

Figure 14.2.15 Kalana Domain A KNA Block Size Performance

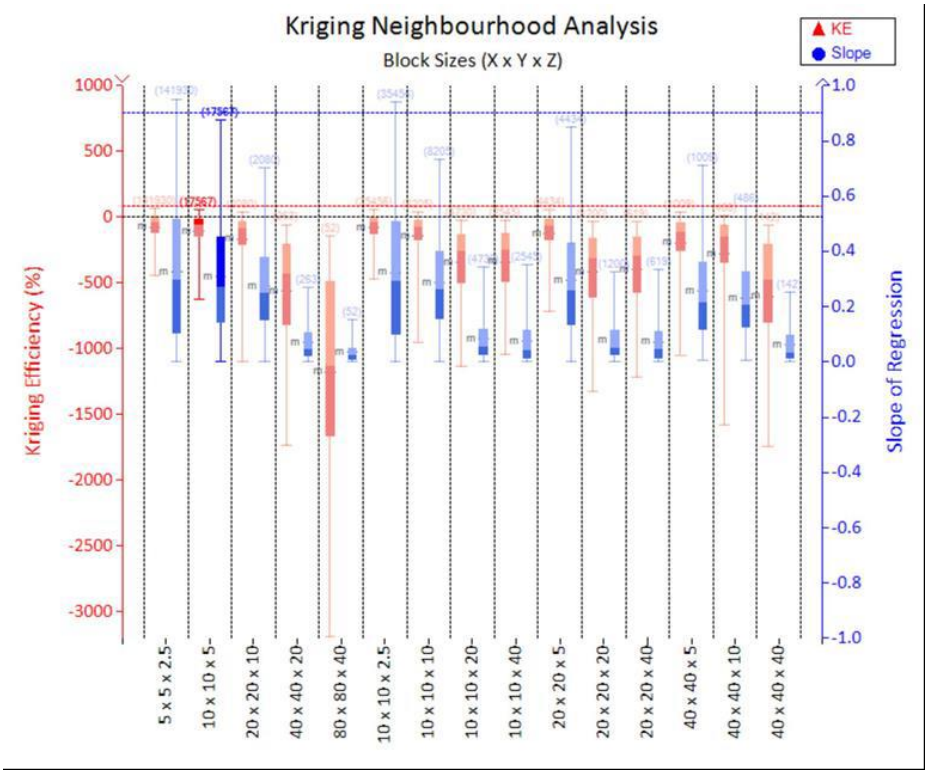


Figure 14.2.16 Kalanako KNA Block Size Performance

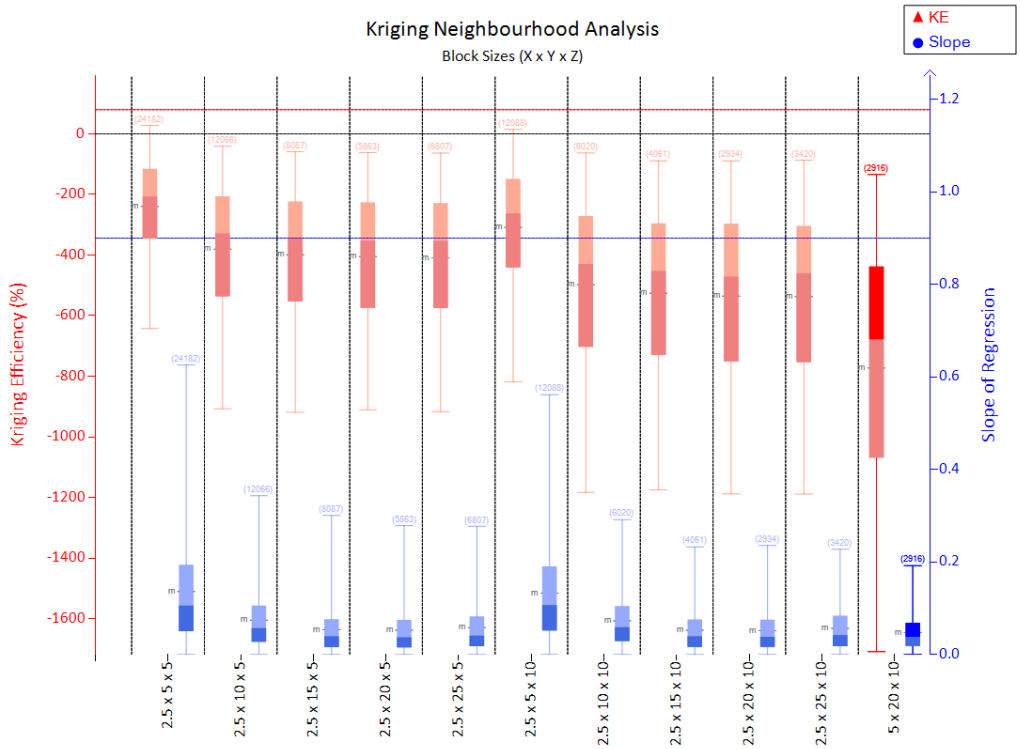


Figure 14.2.17 Kalana Domain A KNA Number Of Samples Performance

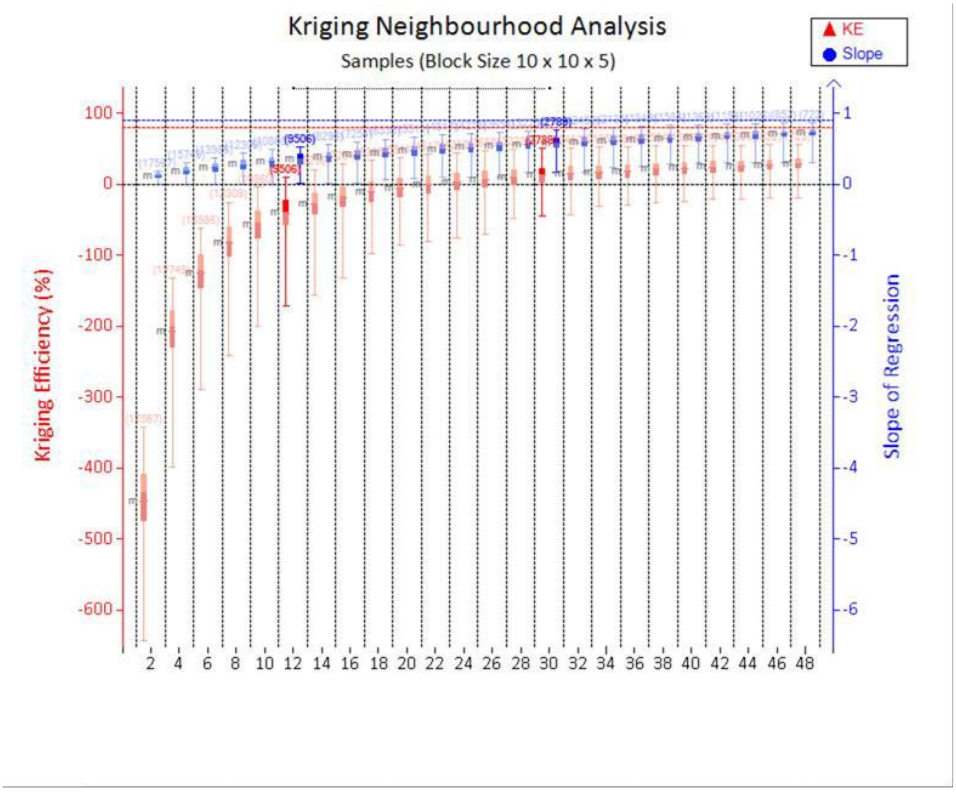


Figure 14.2.18 Kalanako KNA Number of Samples Performance

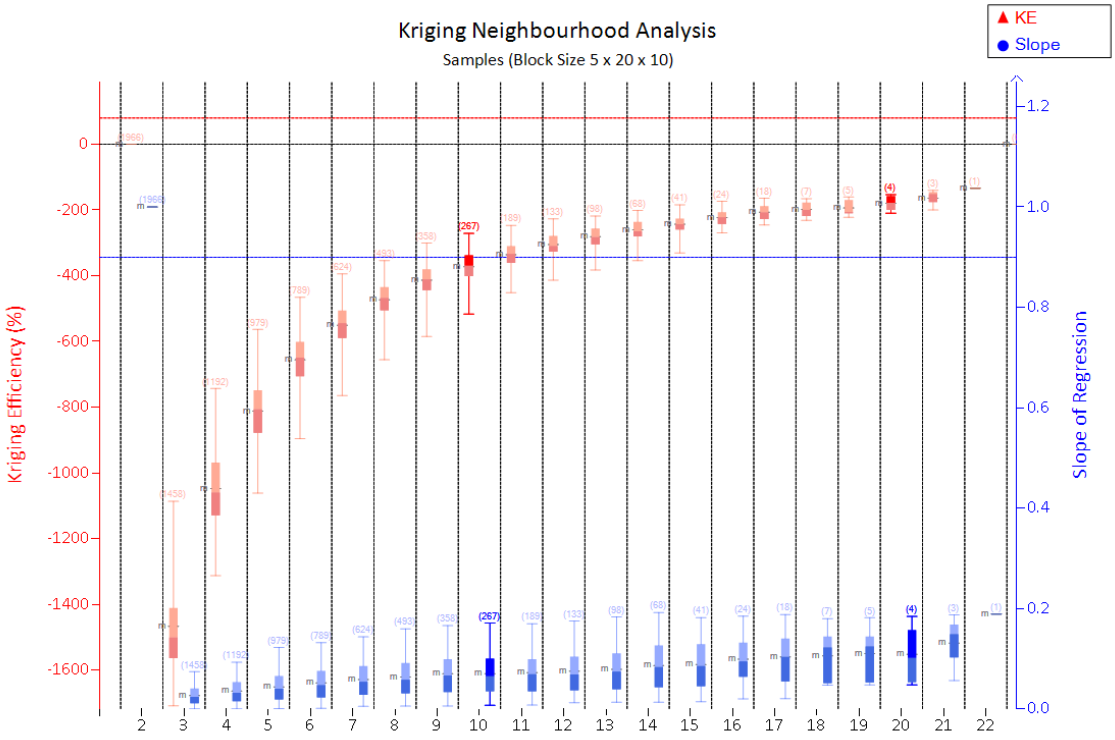


Figure 14.2.19 Kalana Domain A KNA Search Radii Performance

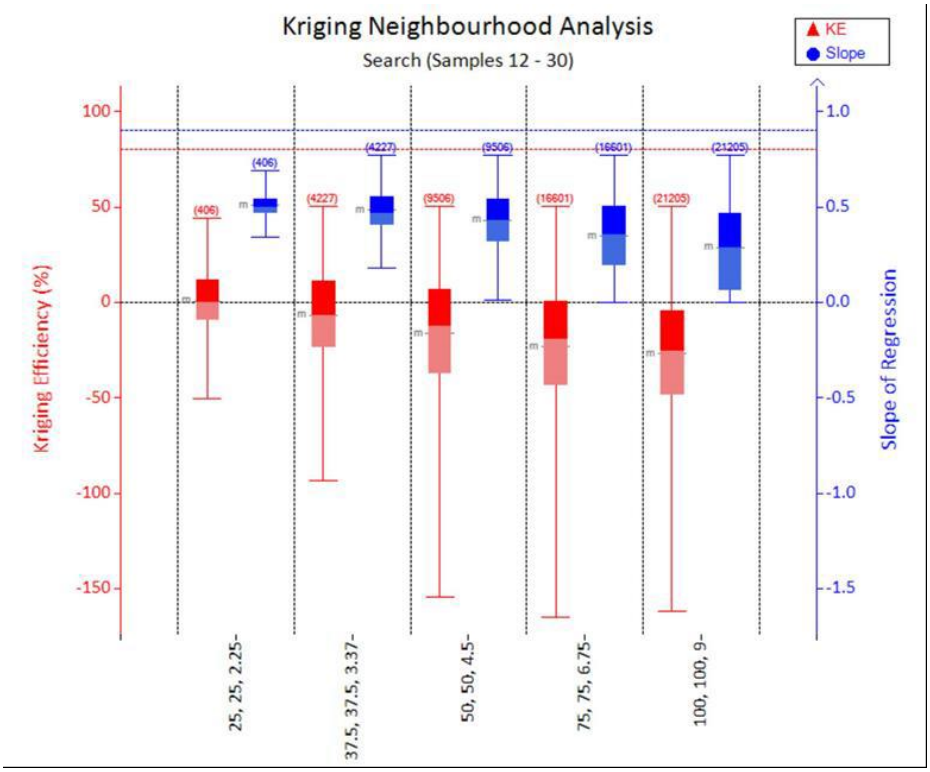
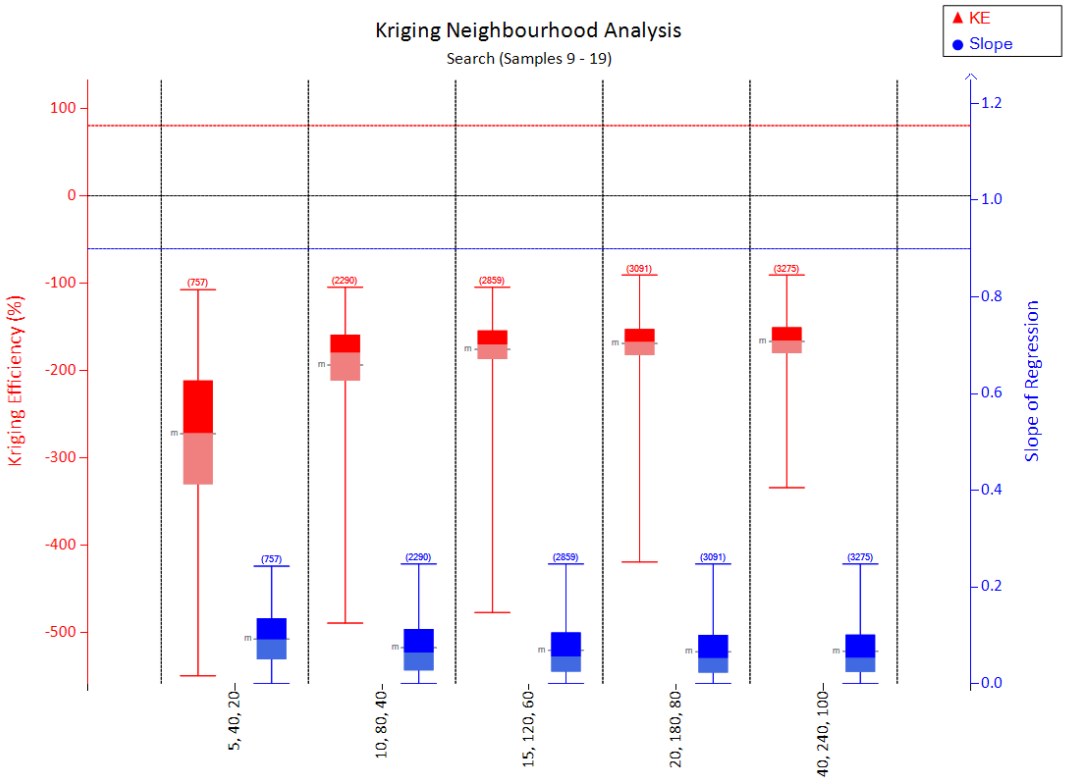


Figure 14.2.20 Kalanako KNA Search Radii Performance



14.2.7 Block Model Creation

A block model was created with topography, weathering, lithology, vein package and TSF wireframes for the Kalana deposit. The wireframes were represented by 10 mE by 10 mN by 5 mRL parent blocks with 5 mE by 5 mN by 1.25 mRL subcells at boundaries. The geographical extent of the Kalana block model is summarised in Table 14.2-9. Unique numeric codes for each vein package volume within each structural domain were created. Model blocks were assigned the codes described in Table 14.2-10 and illustrated in Figure 14.2.21 and Figure 14.2.22.

Table 14.2-9 Kalana Block Model Extents and Block Size

Axis	Minimum	Maximum	Range	Parent Block Size	Number of Blocks
Easting	586,610	588,520	1,910	10	191
Northing	1,192,480	1,194,150	1,670	10	167
Elevation	-100	420	520	5	104

Table 14.2-10 Kalana Block Model Non-Standard Fields

Field	Description
ASS	AU_CUT is assigned (1) rather than estimated (0)

AU_CUT	Estimated gold grade at panel scale
AU_DOM	Vein package sub-domain
DENSITY	Density value
LITHDOM	Lithology domain
MINDOM	Vein package structural domain
MINDOM2	Vein package number (MINDOM2 = MINDOM + (Surpac Object Number) + (Surpac Trisolation Number)/100)
OK_MD	Ordinary kriging minimum distance ratio
OK_NS	Number of composites employed for block grade estimate
OK_SV	Search pass applicable to block grade estimate (1,2,3)
OK_VAR	Ordinary kriging variance
RESCAT	Preliminary resource classification code (2 - Indicated,3 - Inferred)
RPEEE	Within RPEEE limit (1) or outside (0)
SMUGRADE	Estimated gold grade at SMU scale
TSF	Block is within (1) or outside (0) tailings storage facility
UG	Block is located within (1) or outside (0) underground development or stoping
WEATHDOM	Weathering domain

Figure 14.2.21 Kalana Cross-Section A-A′ Showing WEATHDOM Coding

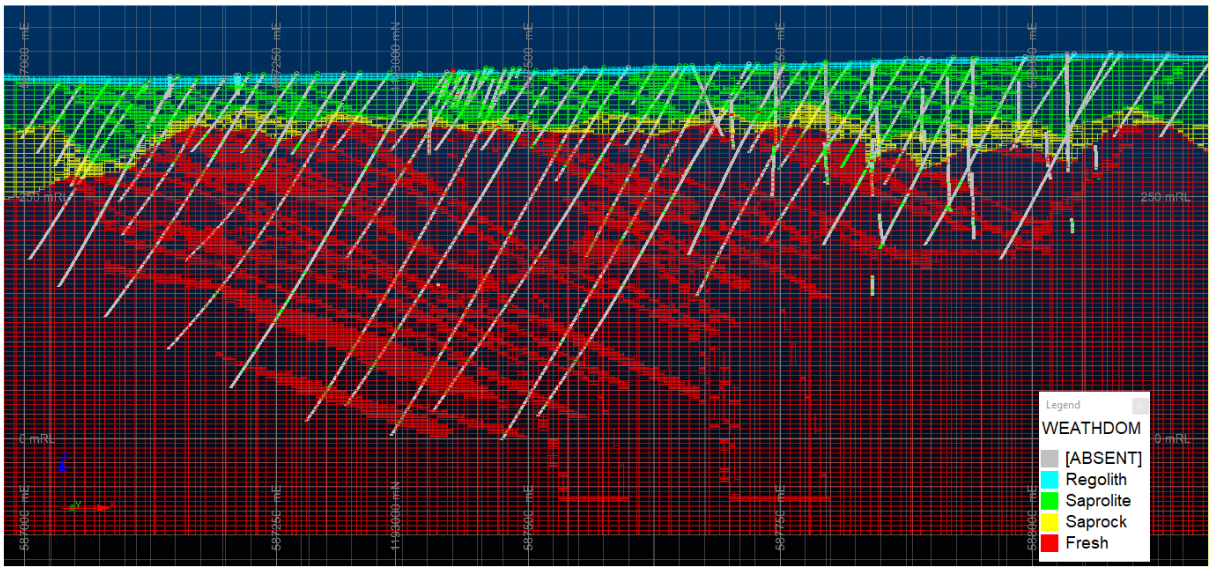
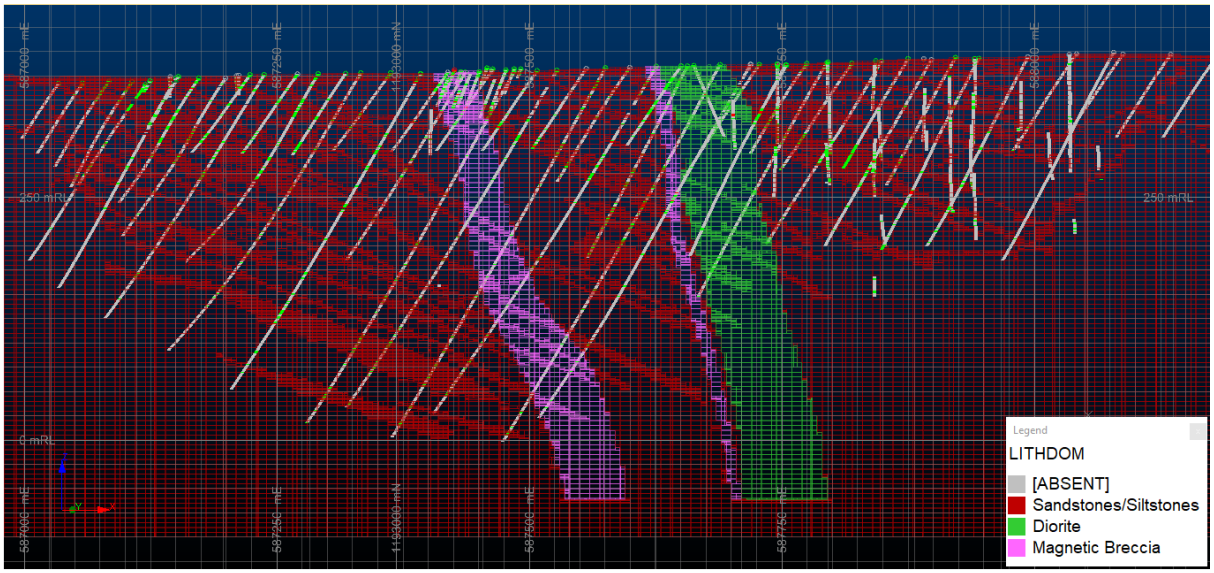


Figure 14.2.22 Kalana Cross-Section A-A′ Showing LITHDOM Coding



An azimuth rotated (310˚) block model was created for the Kalanako deposit using the supplied topography, weathering, oxidation, graphitic and mineralisation wireframes. The geographical extent of the block model is summarised in Table 14.2-11. Brief descriptions of the non-standard block model fields are provided in Table 14.2-12. The wireframes were represented by 5 mX by 20 mY by 10 mRL parent blocks with 1.25 mX by 5 mY by 2.5 mRL subcells at boundaries. This block coding is illustrated in section in Figure 14.2.23 and Figure 14.2.24.

Table 14.2-11 Kalanako Block Model Extents and Block Size

Axis	Bottom Left -Hand Corner and Rotation Pont	Range (m)	Parent Block Size (m)	Number of Blocks
X	590,000	640	2.5	256
Y	1,194,270	1,860	20	93
Elevation	120	310	10	31

Table 14.2-12 Kalanako Block Model Non-Standard Fields

Field	Description
ASS	AU_CUT is assigned (1) rather than estimated (0)
AU_CUT	Estimated gold grade at panel scale
AU_DOM	Vein package sub-domain
DENSITY	Density value
GRAPH	Graphitic domain code
MIN	Inside mineralisation interpretation (1) or outside (0)
MINDOM	Mineralisation domain number
AU_NS	Number of composites employed for block grade estimate
AU_SV	Search pass applicable to block grade estimate (1,2,3)
AU_VAR	Ordinary kriging variance
OXIDE	Oxidation domain code
RESCAT	Preliminary resource classification code (2 - Indicated, 3 - Inferred, 4 - Unclassified)
RPEEE	Within RPEEE limit (1) or outside (0)
NO_RPEEE	RESCAT setting prior to applying RPEEE

Figure 14.2.23 Kalanako Section 1,194,900 mN Showing Weathering Coding

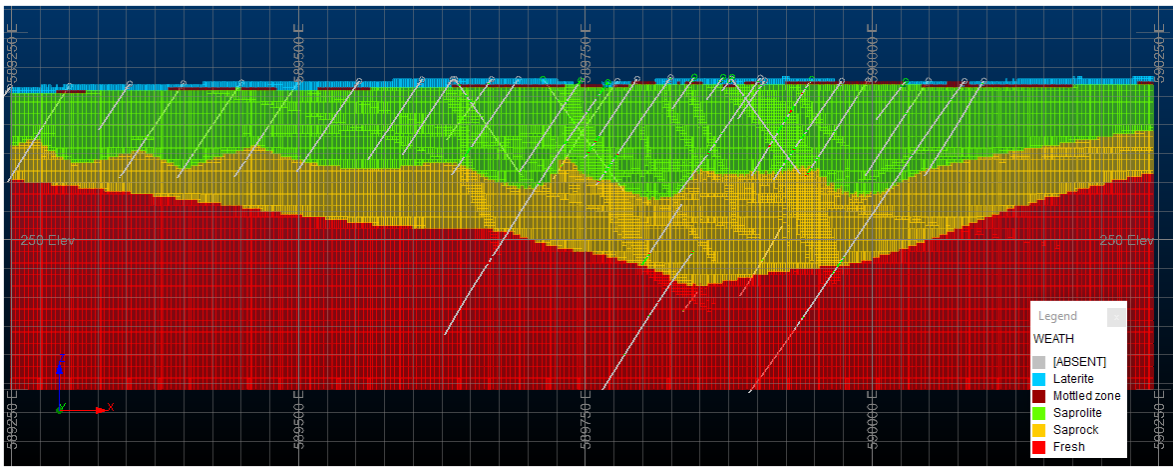
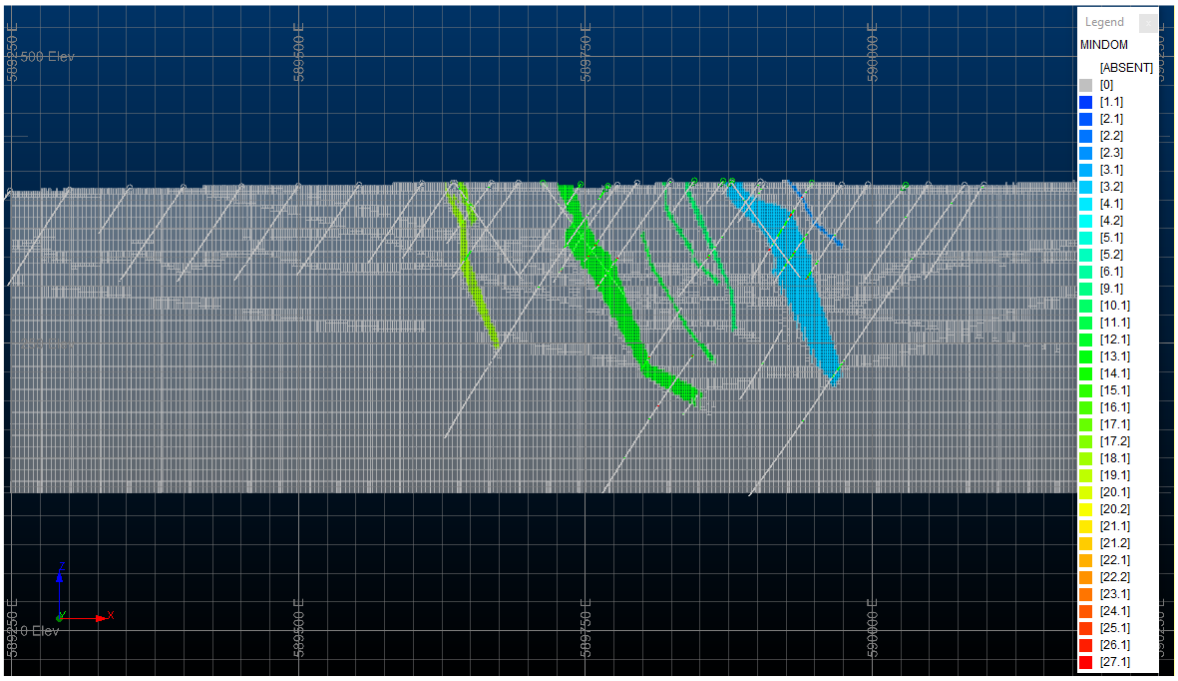


Figure 14.2.24 Kalanako Section 1,194,900 mN Showing MINDOM Coding



14.2.8 Parent Block Grade Estimation

Parent block grade estimation was undertaken using ordinary kriging (OK) of the top-cut, one metre downhole composite samples.

Kalana

The Kalana estimation process was constrained within the vein package sub-domains and undertaken with the vein package projected to a horizontal plane. The OK used the continuity parameters detailed in Table 14.2-7, which were derived from the mid-grade sub-domain data and applied to all sub-domain categories. These variograms have a total sill which was normalised to a value of one.

A multiple pass search strategy was applied in the estimation process. For the lower and mid-grade sub-domains (AU_DOM=0 or 1), the primary search was set at 40 m by 40 m in the horizontal plane and three meters across the plane. For the highest-grade sub-domain (AU_DOM=2), the primary search was reduced to 20 m by 20 m by 3 m. A second search pass doubled these search ranges, while a final third pass search multiplied the primary ranges by a factor of ten to inform any peripheral blocks.

Between 12 and 30 samples were required to inform the primary and secondary search passes for the lower and mid-grade sub-domains. The minimum number of samples required for the highest-grade domain was reduced to six, primarily as this domain was informed by fewer samples and preliminary estimation runs had struggled to estimate blocks. For the third search pass, these minimum sample numbers were reduced to six and two respectively. In all cases, individual drillholes were constrained to provide a maximum of three samples per drillhole.

Discretisation was set at 2 by 2 by 1. Block grade estimation was not particularly sensitive to the discretisation selected.

Figure 14.2.25 Kalana Cross-Section A-A′ Showing Estimated Parent Block Grades

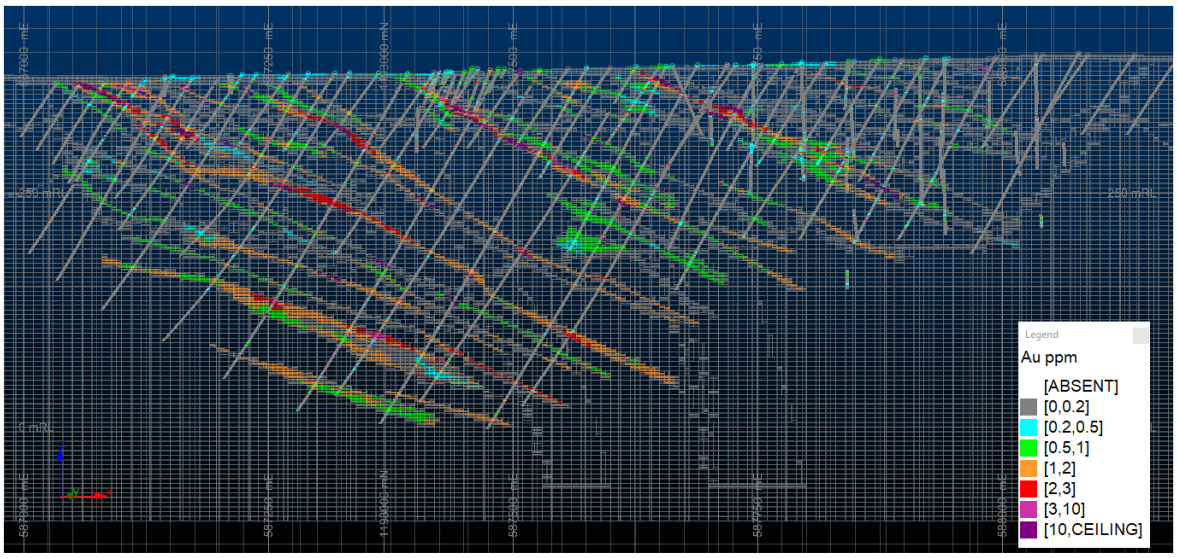
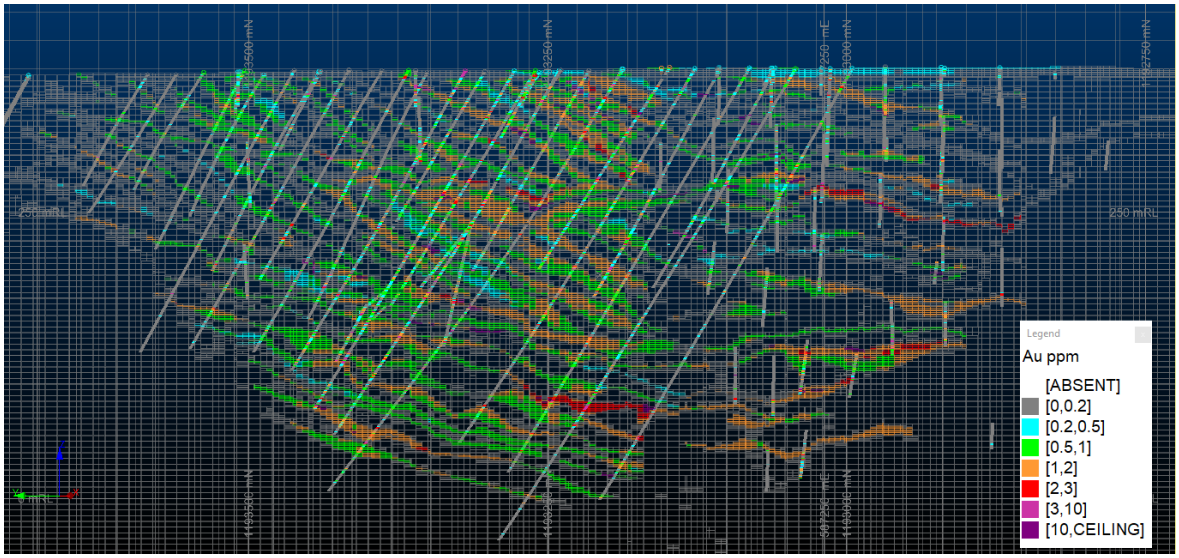


Figure 14.2.26 Kalana Cross-Section B-B′ Showing Estimated Parent Block Grades



An analysis of the proportion of blocks informed in each search pass shows that the lower and mid-grade sub-domains were largely estimated during the second search pass. The highest-grade sub-domain was largely estimated during the third search pass. This is a consequence of the low number of composite samples hosted by this sub-domain, particularly when the estimation process is constrained to only those sub-domain composites located within individual vein packages.

A small proportion of the sub-domain blocks did not receive a grade estimate, largely relating to the low- and high-grade sub-domains. In both cases, a typical cause of this situation is extended distances from drillholes relative to the search radii and the minimum sample number requirements applied. Where no block grade was estimated during the kriging process, the average grade of the estimated sub-domain grades was assigned to the uninformed blocks within the pertinent structural domain. Assigned grades varied from 0.09 Au g/t to 0.12 Au g/t for the low grade sub-domain, 0.91 Au g/t and 1.21 Au g/t for the medium grade sub-domain and 17.26 Au g/t to 23.16 Au g/t for the high grade sub-domain.

Kalanako

The estimation process was constrained within the mineralisation domains (MINDOM) and the grade sub-domains (AU_DOM) and employed dynamic anisotropy to locally orientate the data search and variogram model, as explained earlier. The OK used the continuity parameters detailed in Table 14.2-8, which were derived from the mid-grade sub-domain data and applied to all sub-domain categories. These variograms have a total sill which was normalised to a value of one.

A multiple pass search strategy was applied in the estimation process. The primary search was set at 40 m along strike, 20 m down dip and 5 m across the plane. A second search pass doubled these search ranges, while a final third pass search multiplied the primary ranges by a factor of 15 to inform any peripheral blocks.

Between 10 and 20 samples were required to inform the primary and secondary search passes for the lower and mid-grade sub-domains. The minimum number of samples required for the highest-grade domain was reduced to two, primarily as this domain was much narrower, informed by fewer samples and the observation that preliminary estimation runs had struggled to estimate block grades. For the third search pass, these minimum sample numbers were reduced to two. In all cases, individual drillholes were constrained to provide a maximum of five samples per drillhole. Discretisation was set at 1 by 8 by 4 to represent the block configuration.

The results of the grade estimation are illustrated in Figure 14.2.27 and Figure 14.2.28. It should be noted that the less than optimal drilling direction compared to the mineralisation strike direction rapidly causes visual mismatches between block and drillhole grades if the drillhole is located any distance off the nominal section line location. For instance, in Figure 14.2.24, the drillhole running down dip in the larger lode on the right of the section is around 11 m off the section line and is actually largely outside the footwall of the lode as the lode has moved to the east. In the sections provided, the section corridor for drillhole visualisation is ±12.5 m.

Figure 14.2.27 Kalanako Section 1,194,900 mN Showing Estimated Parent Block Grades

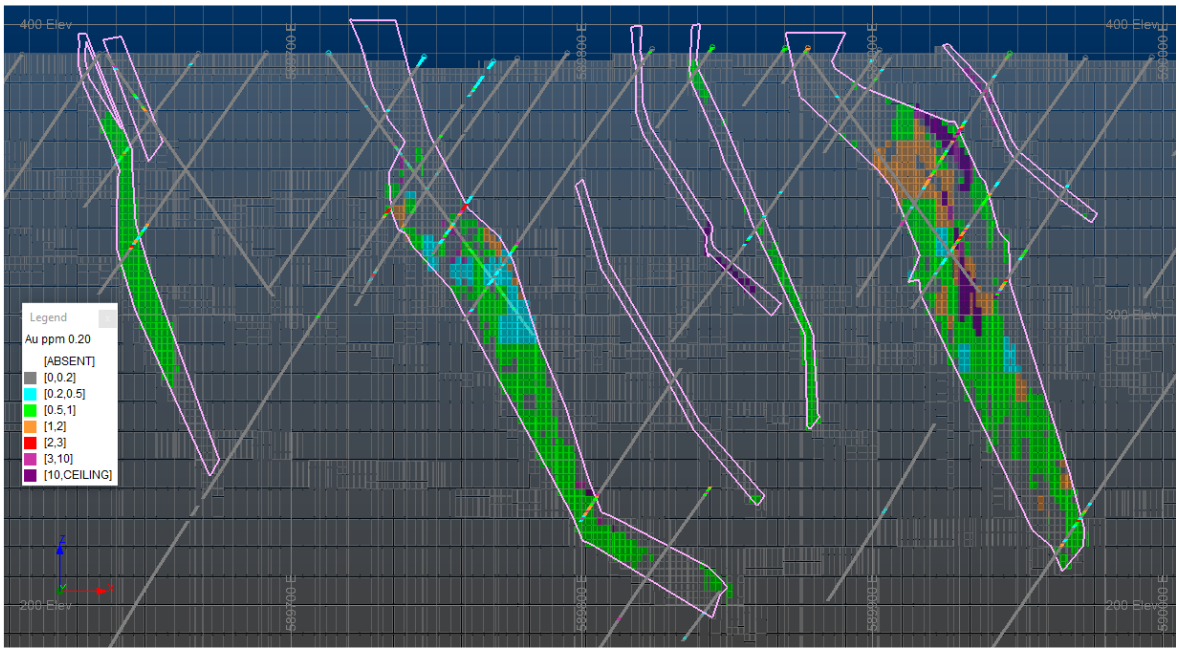
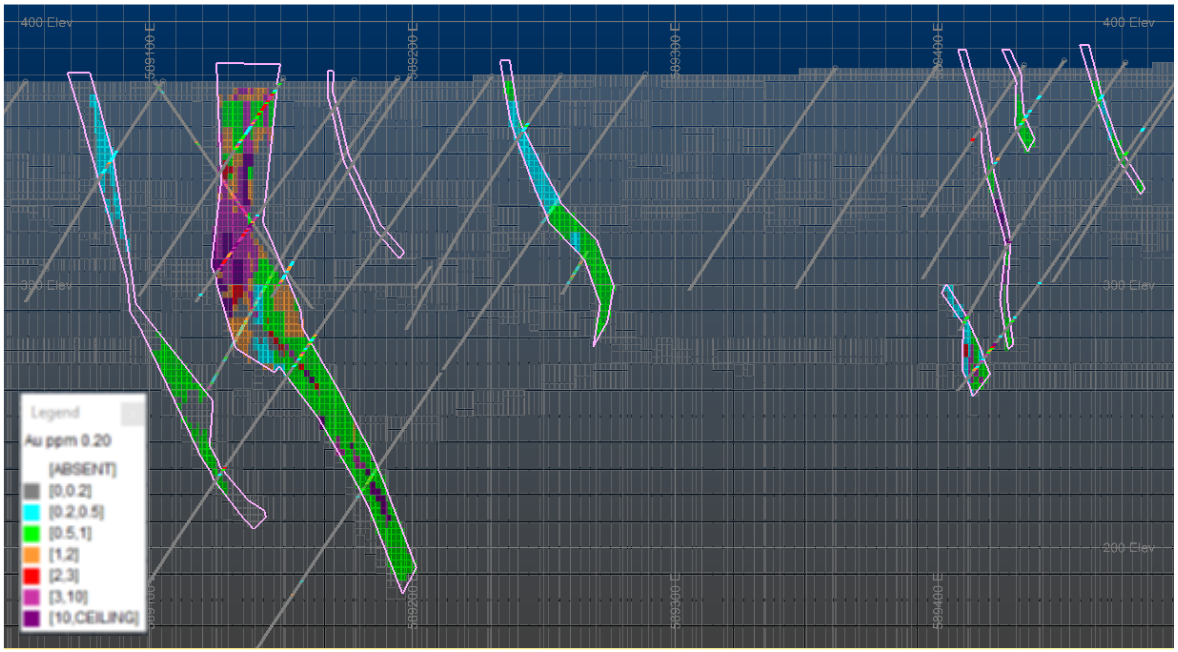


Figure 14.2.28 Kalanako Section 1,195,250 mN Showing Estimated Parent Block Grades



An analysis of the proportion of blocks informed in each search pass shows that the lower and mid-grade sub-domains were largely estimated during the third search pass (Table 14.2-13). The highest-grade sub-domain was largely estimated during the first search pass. These patterns reflect several factors. For the lower and mid-grade sub-domains, material issues are:

- The drillhole spacing and orientation to the mineralisation strike.
- The reduction in demonstrated continuity between the 0.2 Au g/t threshold mineralisation continuity and the mid-grade sub-domain grade continuity.
- Sub-domain geometry.
- The minimum number of samples (10) required to inform a block in the first two search passes.
- The narrow width of the smaller lodes
- The limit on the number of samples (5) that can be sourced from a single drill hole.

In the case of the highest-grade sub-domain, more blocks were estimated in the first search due to:

- Closer alignment between the mineralisation continuity applied and the subsequent grade continuity model.
- The relaxed constraint that blocks could be informed by as few as two samples in the first search.

Section examples of the search pass informing blocks are provided in Figure 14.2.29 and Figure 14.2.30.

Table 14.2-13 Kalankao Search Pass Statistics

AU_DOM	Search Pass	Volume	Proportion
0	Not estimated	94	0%
	1	173,266	15%
	2	107,578	10%
	3	850,031	75%
1	Not estimated	-	0%
	1	464,750	23%
	2	229,344	11%
	3	1,342,031	66%
2	Not estimated	2,422	1%
	1	160,188	88%
	2	9,000	5%
	3	9,813	5%

Figure 14.2.29 Kalanako Section 1,194,900 mN Showing Grade Estimation Search Pass

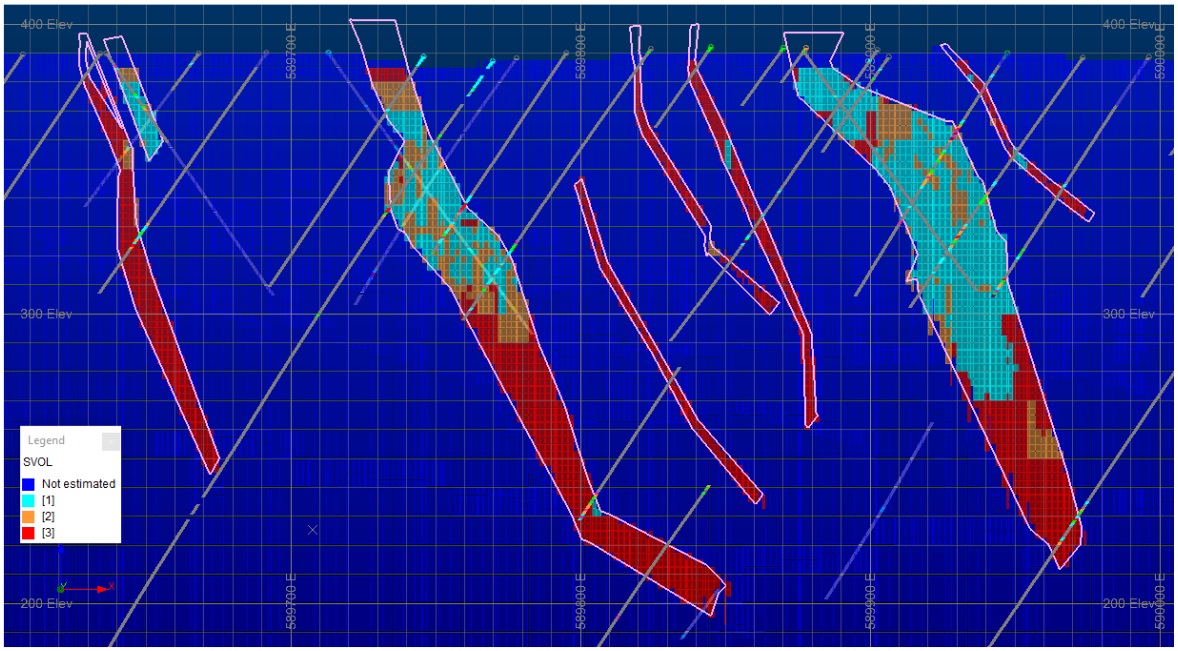
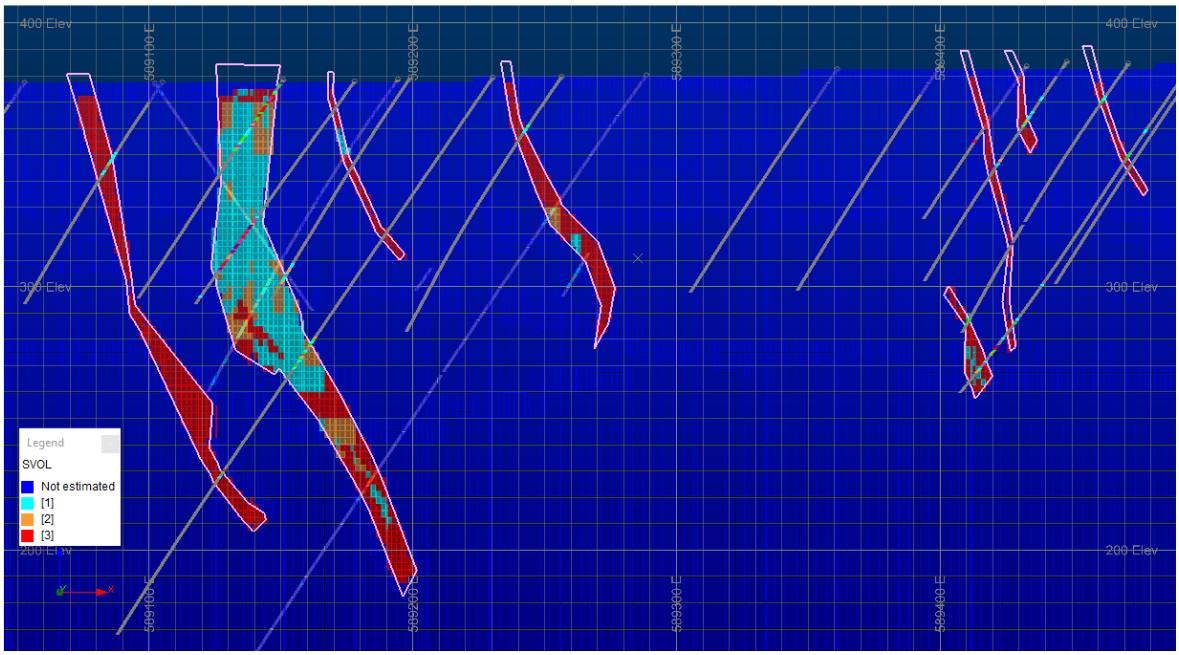


Figure 14.2.30 Kalanako Section 1,195,250 mN Showing Grade Estimation Search Pass



A small proportion of the sub-domain blocks did not receive a grade estimate, mainly within the high-grade sub-domain. Where no block grade was estimated during the kriging process, the average grade of the input declustered drill hole composites within the mineralisation lode and sub-domain combination was assigned to the uninformed blocks. Assigned grades varied from 4.67 Au g/t to 18.26 Au g/t for the high-grade sub-domain.

14.2.9 Model Validation

The block grade estimates were visually validated on screen by comparing the estimated block grades and the input composites. Visual appraisal indicates that the block grades are responding to local drillhole grades; however, the parent block grades are very smooth as a consequence of the high nugget effect and short continuity ranges revealed by the variography analysis.

The block estimates were statistically validated against the naïve and declustered, top-cut informing composites on a whole-of-domain basis, working within the sub-domains within each structural domain at Kalana and within the sub-domains for each mineralisation lode at Kalanako.

Kalana

The Kalana block estimate results obtained show excellent performance within the mid-grade sub-domain (AU_DOM=1), adequate performance within the low-grade sub-domain (AU_DOM=0) and variable performance within the high-grade sub-domain. Within the high-grade sub-domain, the two poorest performing domains (B and C) are both small in terms of domain tonnage and have been informed by the lowest amounts of data. On this basis, no further work was done on these sub-domains. In the case of Domains E and G, the, the average block grade sits between the declustered and naïve mean grades which is an acceptable outcome.

Grade trend profile plots were constructed for the grouped vein packages within each structural domain for each sub-domain to test for any global bias and to compare the average grade of the block estimates with the average of the declustered top-cut composited input samples for slices through the models. The plots assist in the assessment of the reproduction of local mean grades and to validate grade trends in the model. The grade profile plots demonstrate good conformance between the composited sample grades trends and the block grades for almost all structural domains and sub-domains. The most obvious departures between the input data and output model grades occur for the low-grade subdomain in Domain D. In this case, the visual comparison supports the whole-of-domain statistics, which indicate that the model grades are higher than anticipated when compared to the input data. The grade patterns in the profile plots suggest that this over-estimation of grade is a consequence of extrapolation of grade into the outer edges of the domain. As the average grade of this domain is in the order of 0.1 Au g/t, no further action was taken.

Kalanako

The Kalanako block estimate results obtained from the highest-grade sub-domain (AU_DOM=2) exhibit several patterns. These domains are all volumetrically small to very small and informed by low numbers of composites. Several of the smallest domains are based on only a single higher-grade sample, which prevented grade estimation using the dynamic anisotropy method. Some of the domains depart from the average grade of the input data. Further appraisal of these domains showed in all cases that the domain geometry and sample locations had resulted in greater emphasis being placed on certain samples which caused the reported grade difference. All outcomes were considered reasonable, although there is more risk associated with these estimates.

The middle-grade sub-domain results are generally adequate, particularly for the larger domains. All domains which were subject to additional review showed that departures between the input data and model result were caused by more poorly informed regions of the domains that were informed by sparser data which was either higher or lower than the average data grade.

The lowest grade sub-domain exhibits a lot of contrasts with the input data. These domains are all very low grade, resulting in small absolute grade differences which in turn lead to large relative differences. As such, no further work was done on these sub-domains and the grade estimates were deemed acceptable.

Grade trend profile plots (swath plots) were reviewed for the sub-domains within the larger six mineralisation lodes. The profile plots show that there is generally good local conformance with both the naïve and declustered profile grades. There are some examples of overly smoothed block grades and there is some evidence of extrapolation issues at each end of the lode's strike range and at depth. These patterns suggest, overall, that the declustered grade may be locally misleading rather than any issue with the block grade estimates. Overall, the conformance between the input data and the block estimates is considered to be adequate.

Similar patterns were noted for all of the larger domains, and generally for the smaller domains, although these tended to be more erratic due to small domain volumes and low numbers of informing samples.

14.2.10 Density Assignment

Density measurements were taken from diamond core samples using a water displacement (Archimedes) method as per Section 11.7.

Density values were assigned to the resource models based on the interpreted weathering domain as summarised in Table 14.2-14.

Table 14.2-14 Block Model Assigned Densities

Rock Type	Kalana	Kalanako
Laterite/Mottled Zone/Regolith	1.67	1.70
Saprolite	1.64	1.76
Sap-rock	2.15	2.09
Fresh	2.68	2.64

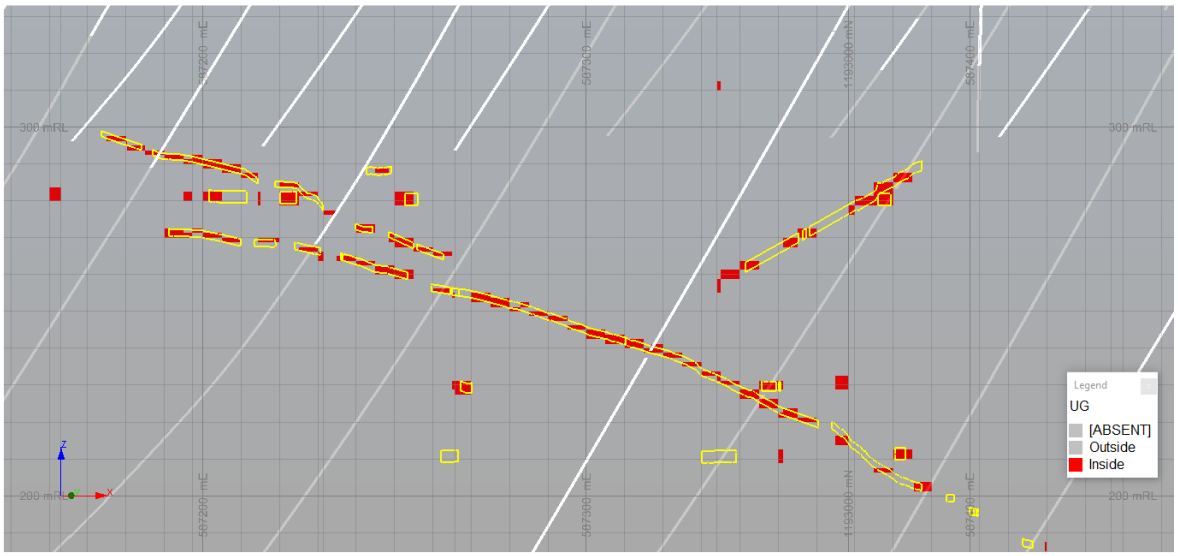
14.2.11 Underground Depletion

The Kalana model has been depleted for underground using a supplied stope and development wireframe model at a resolution of 5 mE by 5 mN by 1.25 mRL. The representation of the underground volumes in the block model is a relatively low resolution at a local scale as many of the modelled voids are very narrow (Figure 14.2.31). Blocks identified as being inside an underground void have been assigned a density value of zero.

It should be noted that the current model is not expected to reconcile with the reported production from underground mining due to:

- The vein package interpretation is not designed to represent individual veins; rather, it represents the mineralisation that may be accessed by less selective open pit mining methods.
- The model block size is not configured to represent discrete narrow veins.
- A significant portion of the drillhole database comprises drilling that occurred after completion of underground mining, thus the holes do not include any sampling from within the mined high-grade veins.

Figure 14.2.31 Representation of Underground Voids in Block Model, Cross Section A-A'



(Yellow perimeters are wireframed voids. Red blocks are coded as inside those voids.)

14.2.12 TSF Estimation

In 2004, the SOGEMORK tailings in the TSF were evaluated by Snowden and a NI 43-101 (CIM) compliant Measured Resource of 234 Kt at 1.9 Au g/t for 14 Koz of gold reported (Snowden, 2005). The following points are considered by Endeavour to be pertinent:

- The TSF was sampled in May 2004 on a 10 m by 10 m grid which collected one metre vertical downhole samples from auger drilling – this data cannot be located.
- No survey volume for the TSF was possible as there was no survey record of the pre-depositional topography of the base of the dam.
- The tonnage in the tailings dam was estimated from historical production records and combined with records for the tailings resulting from ore milling for the period from January to June 2004.

Since 2004, the tonnage and grade of tailings deposited in the TSFs were measured and recorded by the SOMIKA processing plant. In 2015, Denny Jones compiled the production data from between June 2004 and September 2015 to derive a resource estimate of 494 kt at 1.68 g/t Au for 26.7 koz. The overall tailings estimate was a simple weighted average of the 2004 SOGEMORK tailings estimate and the June 2004 to September 2015 SOMIKA production data to result in a Measured Resource of 728 kt at a grade of 1.75 Au g/t for 40,000 oz gold (Denny Jones, 2015). The Shaft No. 2 TSF has an assumed tonnage of 192 kt based on its geometry (including the 2016-17 depletion). Since 2015, an additional 102 kt of material has been added to the TSF.

Endeavour estimates the TSF proper as of June 2020 to contain 653 kt at 1.8 Au g/t containing 37.3 koz Au. The Shaft No. 2 TSF is estimated by its volume (128,000 m^3) multiplied by a dry bulk density (1.5) at a grade of 1.7 Au g/t to contain 192 kt at 1.7 Au g/t containing 10.4 koz Au. Table 14.2-15 summarises the assumptions used. The Total TSF Indicated Resource is estimated to be 845 kt @ 1.8 Au g/t containing 48 koz Au.

Table 14.2-15 TSF Assumptions

Parameter/MRE	Dam	kt	Grade (Au g/t)	koz
Pre1991 SOGEMORK Production (Snowden 2005 MRE)	TSF Proper	234	1.9	14.0
2004 - 15 SOMIKA Production Tailings	TSF Proper	302	1.68	16.3
2004 - 09 SOMIKA Production Tailings	Shaft No. 2	192	1.68	10.4
2004 - 15 Subtotal Production Tailings		**494**	**1.68**	**26.7**
2015 Snowden MRE	Both	728	1.75	40.0
2016-17 SOMIKA Production Tailings	TSF Proper	117		
	Total	**845**		
June 2020 MRE	TSF Proper	653	1.8	37.3
	Shaft No. 2 TSF	192	1.7	10.4
	Total	**845**	**1.8**	**47.6**

14.3 Mineral Resource Classification and Reporting

The Mineral Resource classification system used is consistent with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards 2014. Under the CIM classification system, a Mineral Resource is defined as:

…"a concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction".

Resources are classified into Measured, Indicated and Inferred categories based upon increasing geological confidence. Resource classification within the mineralised vein wireframes are generally based on drill hole spacing, grade continuity and overall geological continuity. The distance to the nearest composite and the number of drill holes are also considered in the classification.

As with any non-rigidly defined classification system, there will always be some blocks within categories that depart from defined criteria. The final outcome reflects a practical combination of geological knowledge and estimation quality parameters that may be more numerical in nature.

14.3.1 Kalana

The Kalana Mineral Resource has been classified using drillhole spacing as the primary criterion. This was achieved by generating a dedicated kriging estimate that accessed all data without any domain controls and utilised a search protocol that required three drillholes to be within a 25 m horizontal radius to satisfy the first search pass, three drill holes to be within a 50 m horizontal radius to satisfy the second search pass and two drillholes within a 125 m horizontal radius to satisfy the third search pass.

The search volume indexing was used to assist in the delineation of parts of the deposit that were informed largely by closer spaced drilling. The preliminary classification assigned an Indicated resource category if a location was largely informed by at least three drillholes within a 50 m radius.

No Measured category material was assigned, largely because of the coarse gold character of the deposit, the high nugget effect component and the relatively poor grade continuity definition provided by the drilling data.

A 'reasonable prospect of eventual economic extraction' (RPEEE) limit was developed using pit optimisation methods to define a pit shell. The RPEEE pit shell was developed using the 10 mE by 10 mN by 5 mRL panel grade values and a $1,500/oz gold price combined with the parameters detailed in Table 14.3-1. The Kalana RPEEE pit shell and mineralisation wireframes are graphically illustrated in Figure 14.3.1.

Table 14.3-1 Kalana RPEEE Parameters

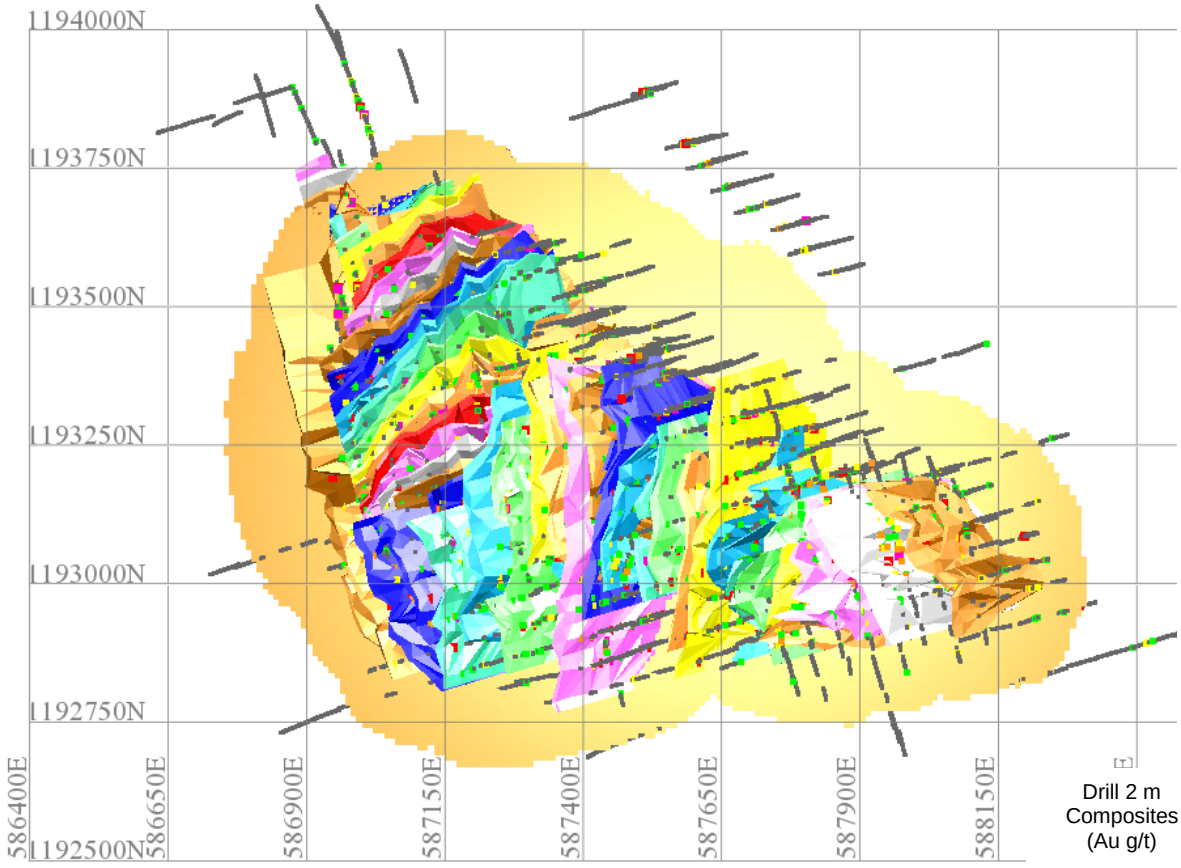
	Units	Laterite	Oxide	Trans	Fresh
Gold Price	$/oz	1,500	1,500	1,500	1,500
Royalty	%	3.6	3.6	3.6	3.6
Discount Rate	%	5.0	5.0	5.0	5.0
Transport & Refinery	$/oz	4.0	4.0	4.0	4.0
Processing Cost - Total	**$/t ore**	**19.82**	**19.96**	**24.52**	**23.00**
Milling	$/t ore	11.54	11.54	15.95	14.26
G&A	$/t ore	6.25	6.25	6.25	6.25
Ore/Waste Differentials	$/t ore	-0.48	-0.34	-0.18	-0.02
Ore Haulage	$/t ore	0.80	0.80	0.80	0.80
Rehandling	$/t ore	0.40	0.40	0.40	0.40
GC	$/t ore	1.31	1.31	1.31	1.31
Recovery	%	95.4	95.4	92.4	92.7
Mining Cost Average Estimated	**$/t mined**	**1.64**	**1.8**	**2.08**	**2.95**
Starting Top Level	mRL	400	405	365	340
Regression - Constant	b	1.62	1.73	1.99	2.44
Change in every 10 m	a	0.016	0.021	0.030	0.056
Geotechnical Slopes - West	deg	28.5	28.5	42	48
Geotechnical Slopes - East	deg	28.5	28.5	45	52
LG Cut-off Grade	Au g/t	0.4	0.5	0.6	0.5
SG Cut-off Grades (40% G&A)	Au g/t	0.4	0.5	0.5	0.4
Plant Throughput			3 Mtpa		

The Kalana MRE is predominately located in the fresh rock, see Table 14.3-2, above a 0.5 Au g/t cut-off grade.

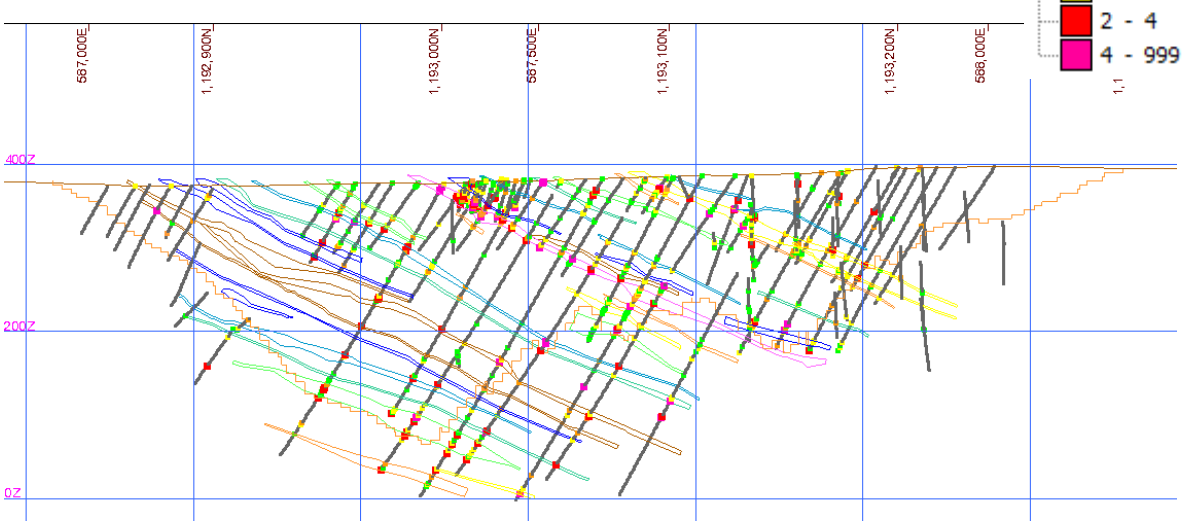
Table 14.3-2 Kalana MRE, June 2020 By Weathering

Rock Type	Indicated			Inferred		
	Tonnes (kT)	Grade (Au g/t)	Gold (kOz)	Tonnes (kT)	Grade (Au g/t)	Gold (kOz)
Regolith/Laterite	665	0.91	20	1	0.62	0
Saprolite	7,720	1.67	415	12	0.96	0
Transition	3,212	1.62	168	8	1.38	0
Fresh	31,931	1.50	1,540	4,188	1.64	221
Total	**43,527**	**1.54**	**2,158**	**4,210**	**1.64**	**222**

Figure 14.3.1 Kalana RPEEE Pit Shell and Mineralisation Wireframes with Drill Hole Two Metre DTH Au Composites



Cross Section, Line A – A



14.3.2 Kalanako

Kalanako has been classified using drillhole spacing and mineralisation lode size (i.e. overall geological confidence) as the primary criteria. This was achieved by referencing the search pass index which was based on multiples of the maximum grade continuity ranges derived from the middle grade category sub-domain. Only the six largest domains were considered as candidates for classification in the Indicated class. Indicated was considered to be a reasonable classification if the local block grades were largely informed during the first search pass completed by the kriging process, meaning that there was adequate data located either 40 m along strike or 20 m down dip.

No Measured category material was assigned, largely because of the relatively poor grade continuity definition provided by the drilling data and its relatively wide spacing.

A RPEEE limit was developed using pit optimisation methods to define a pit shell. Any mineralisation located outside the selected pit shell was not classified. The RPEEE pit shell was developed using the 2.5 mE by 20 mN by 10 mRL panel grade values and a $1,500/oz gold price, combined with the parameters provided by EDV and detailed in Table 14.3-3 and presented in Figure 14.3.2.

Table 14.3-3 Kalanako RPEEE Parameters

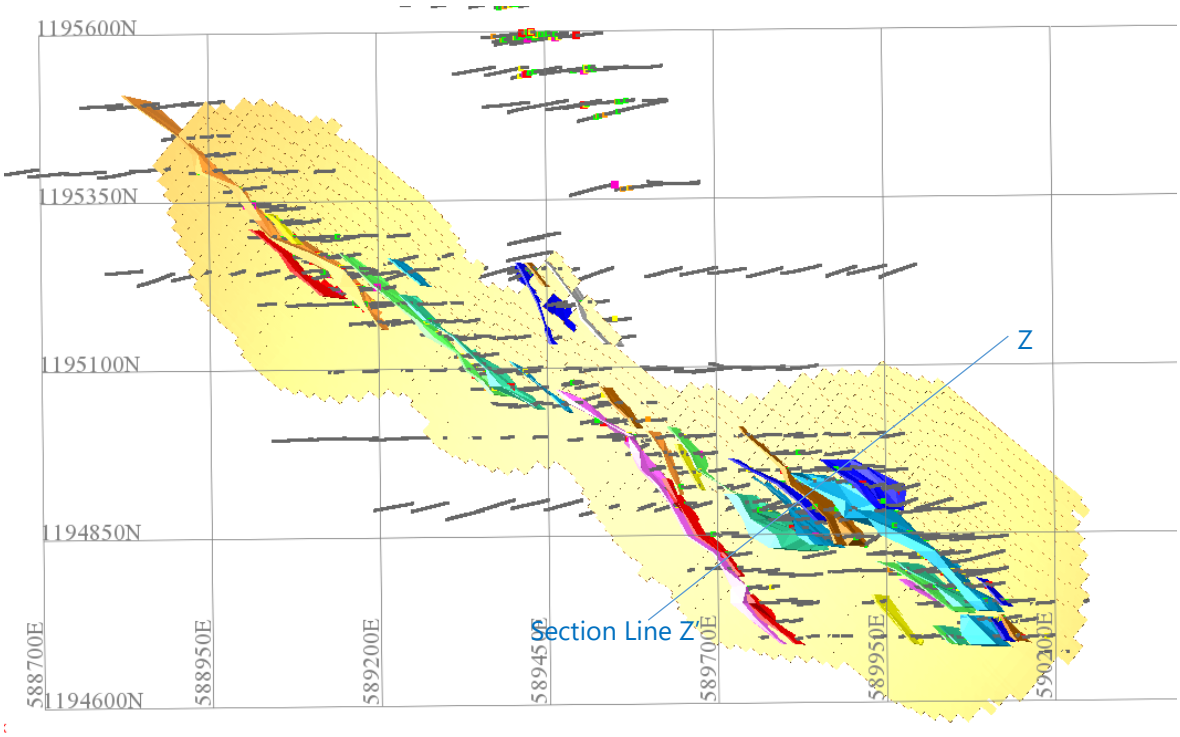
	Units	Laterite	Oxide	Transition	Fresh
Gold Price	$/oz	1,500	1,500	1,500	1,500
Royalty	%	3.60%	3.60%	3.60%	3.60%
Discount Rate	%	5.00%	5.00%	5.00%	5.00%
Transport & Refinery	$/oz	4	4	4	4
Troy Ounce to Grams Conversion	factor	31.1	31.1	31.1	31.1
Processing Cost - total	**$/t ore**	**20.19**	**20.26**	**24.78**	**23.41**
Milling	$/t ore	11.22	11.22	15.77	14.32
G&A	$/t ore	6.18	6.18	6.18	6.18
Ore/Waste Differentials	$/t ore	0.25	0.31	0.29	0.37
Ore Haulage	$/t ore	0.8	0.8	0.8	0.8
Rehandling	$/t ore	0.42	0.42	0.42	0.42
GC	$/t ore	1.31	1.31	1.31	1.31
Recovery	%	96 .0	96.0	88.5	89.0
Mining Cost Waste Average Estimated	**$/t mined**	**1.53**	**1.68**	**2.11**	**2.82**
Starting Top Level	mRL	0	385	360	295
Regression - Constant	b	1.53	3.495	3.424	4.02
Change in every 10 m	a	0	-0.0051	-0.0045	-0.0053
Geotechnical Slopes – West	deg	28.5	28.5	42	48
Geotechnical Slopes – East	deg	28.5	28.5	45	52
LG – Cut-off Grades	Au g/t	0.5	0.5	0.6	0.6
MO Cut-off Grades - 40% GA	Au g/t	0.4	0.4	0.5	0.5

A feature of the Kalanako deposit is that a large portion of the resource is hosted within oxidised material, as shown in Table 14.3-4 at a 0.5 Au g/t reporting cut-off grade.

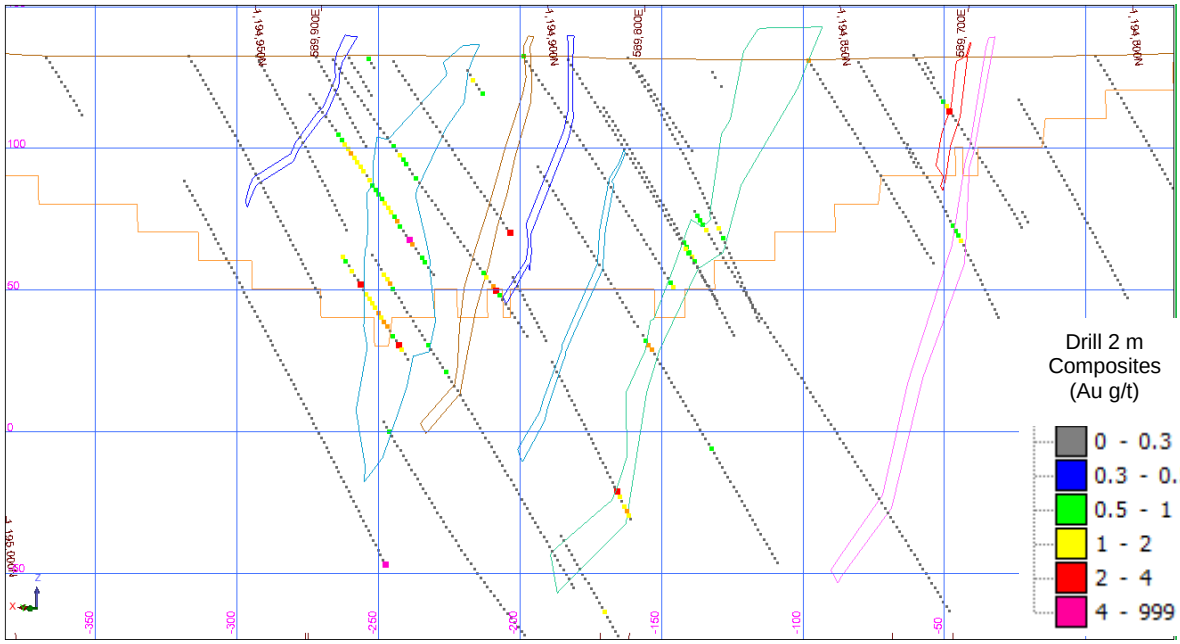
Table 14.3-4 Kalanako MRE, June 2020 By Weathering

Rock Type	Indicated			Inferred		
	Tonnes (kT)	Grade (Au g/t)	Gold (kOz)	Tonnes (kT)	Grade (Au g/t)	Gold (kOz)
Regolith/Mottled Zone/Laterite	16	1.80	1	13	1.75	1
Saprolite	1,219	2.23	87	314	1.61	16
Transition	318	1.85	19	20	8.54	5
Fresh	61	2.18	4	4	0.83	0
Total	**1,614**	**2.15**	**112**	**351**	**1.99**	**23**

Figure 14.3.2 Kalanako RPEEE Pit Shell and Mineralisation Wireframes with Drill Hole Two Metre DTH Au Composites



Cross Section, Line Z – Z'



14.3.3 TSFs

Endeavour considers there to be enough uncertainties, associated primarily with data loss, sample quality and the inability to accurately define the base of the tailings dams that the Mineral Resource classification should be reduced to Indicated (rather than Denny Jones' Measured classification).

14.4 Statement

The Kalana Project is composed of two primary deposits, Kalana and Kalanako, and two TSFs. The Kalana deposit has an Indicated Mineral Resource of 43.5 Mt at a grade of 1.54 Au g/t containing 2.16 Moz gold and an Inferred Mineral Resource of 4.2 Mt at a grade of 1.64 Au g/t containing 222 koz gold.

Kalanako has an Indicated Mineral Resource of 1.6 Mt at a grade of 2.15 Au g/t containing 112 koz gold and an Inferred Mineral Resource of 351 kt at a grade of 1.99 Au g/t containing 23 koz gold.

The two TSFs have a combined Indicated Mineral Resource of 845 kt at a grade of 1.8 Au g/t containing 48 koz gold. No cut-off grade has been applied and the entire resource would be mined.

The Kalana Project has an Indicated Mineral Resource of 46.0 Mt at a grade of 1.57 Au g/t containing 2.32 Moz gold and an Inferred Mineral Resource of 4.6 Mt at a grade of 1.67 Au g/t containing 245 koz gold. The effective date of the Kalana Project MRE is 30 June 2020 and is presented in Table 14.4-1.

Table 14.4-1 Kalana Project Mineral Resource Estimate, as of 30 June, 2020

Deposit	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Gold (koz)	Tonnes (kt)	Grade (Au g/t)	Gold (koz)
Kalana	43,527	1.54	2,158	4,210	1.64	222
Kalanako	1,614	2.15	112	351	1.99	23
TSFs	845	1.8	48	-	-	-
Total	**45,986**	**1.57**	**2,318**	**4,561**	**1.67**	**245**

Notes:

1. Mineral Resource estimates follow the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition standards for Mineral Resources and Reserves and have been completed in accordance with the Standards of Disclosure for Mineral Projects as defined by National Instrument 43-101.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Kalana and Kalanako Mineral Resources are constrained by MII $1,500/oz Pit Shell and based on a cut-off of 0.5 Au g/t.
4. The TSF Mineral Resource does not have a cut-off grade applied.
5. Reported tonnage and grade figures have been rounded from raw estimates to reflect the relative accuracy of the estimate. Minor variations may occur during the addition of rounded numbers.
6. The Qualified Person for the exploration data and geological interpretation is Helen Oliver MSc FGS CGeol, Endeavour Mining Group Resource Geologist. The Qualified Person for the Tailings Mineral Resource Estimate is Ms. Oliver.
7. The Qualified Person for the Kalana and Kalanako Mineral Resource Estimate is Paul Blackney BSc Hons MAusIMM MAIG, Optiro Pty. Ltd.
8. The cut-off date for the Kalana drill hole database is July 1, 2018 and February 7, 2018 for the Kalanako drill hole database. The effective date of the MRE is June 30,2020.

14.5 Comparison with Historical Mineral Resource Estimates

Denny Jones reported a NI 43-101 compliant Mineral Resource estimate for Kalana in March 2016 at a 0.9 Au g/t cut-off grade. It contained a Measured Resource of 9.5 Mt at 4.20 Au g/t containing 1.28 Moz Au, an Indicated Resource of 13.5 Mt at 4.10 Au g/t for 1.77 Moz Au and an Inferred Resource of 1.7 Mt at 4.51 Au g/t for 0.24 Moz Au within a RPEEE pit shell generated using a $1,400 gold price (Denny Jones, 2016).

Optiro notes the 2016 Kalana resource estimate was classified as Measured and Indicated with very little Inferred. The Measured and Indicated was split roughly 50:50. In Optiro's opinion, it is not possible to apply a better than Indicated classification to a coarse gold deposit defined by drilling on the grid spacings available at Kalana. It may be argued that 25 m by 25 m drilling is adequate to support a Measured category; however, this ignores issues related to sample representivity, and Optiro observes that the closer spaced drilling appears to have been carried out as a response to greater mineralisation complexity in the areas where it occurs.

The changes in the tonnage and grade estimates between the 2016 and 2020 MERs can be attributed to multiple factors including:

- Different styles of interpretation, i.e. vein packages and proportions (acting as soft boundaries) in 2016 compared to specific veins and hard boundaries in 2020.
- Differing emphasis on high grade continuity.
- No top-cuts were employed in the 2016 Estimate but were in the 2020 Estimate.
- Different algorithms were used to estimate the grade (i.e. Multiple Indicator Kriging in 2016 and Categorical Ordinary Kriging in 2020).

Optiro and Endeavour are of the opinion that 2020 methodology is a more pragmatic approach (which can be replicated and updated as and when required) and is a robust and realistic estimate of Kalana and Kalanako's gold content. Optiro and Endeavour acknowledge that the MRE presented here-in is the base case and potential upside exists due to the currently intangible effect of the coarse gold and nugget effect.

Denny Jones reported a NI 43-101 compliant Mineral Resource estimate for Kalanako in July 2017 at a 0.9 Au g/t cut-off grade of 0.77 Mt at 4.61 Au g/t for 114,000 oz of gold reported within a RPEEE pit shell generated using a $1,400 gold price (Denny Jones, 2017).

Optiro notes if the 2017 cut-off grade of 0.9 Au g/t was applied to the 2020 Kalanako MRE, it would yield 0.80 Mt at 3.67 Au g/t for 94,000 oz of gold. Compared to the 2017 reporting, the current model represents 4% more tonnage at 20% less grade for 17% less metal. The causes of these differences have not been pursued in detail but are likely to relate to a combination of the modified interpretation and estimation methodology applied to generate the current resource estimate, compared to the preceding methodology. Optiro considers the magnitude of the differences to be commensurate with the largely Indicated classification applied to the resource estimate.

Endeavour considers there to be enough uncertainties, associated primarily with data loss, sample quality and the inability to accurately define the base of the tailings dam, that the Mineral Resource classification should be reduced to Indicated, rather than Denny Jones' Measured classification.

The 2020 TSF MRE estimate is of similar magnitude than the 2015 Estimate which confers confidence on the estimate.

15.0 MINERAL RESERVE ESTIMATE

15.1 Mining and Mineral Reserve Estimation Approach

The Mineral Reserve statement for the Kalana Gold Deposit dated December 2020 is supported by the engineering designs and modifying factors in the Pre-Feasibility Study. The reported Mineral Reserve include the following sources:

- Kalana open pit
- Kalanako open pit and
- Historical tailings produced from the Kalana underground mine.

The Mineral Reserve estimation was achieved by:

- Modification of the Resource block model.
- Open pit optimisation to estimate the potential pit limits.
- Engineering pit design of ultimate pit and stage designs.
- Mine plan scheduling.
- Reporting.

15.2 Key Assumptions and Basis of Estimate

15.2.1 Resource Classification

Only Measured and Indicated were considered as potential ore blocks with Inferred and unclassified resource categories treated as waste.

15.2.2 Initial Surface

The initial surface for the pit optimisations was the topography coded into the Resource models. The Kalana model also included a field indicating the location of underground workings which were assigned a zero density to ensure gold already mined was not included in the pit optimisation.

15.2.3 Dilution and Mining Recovery

No modifications were made to the Kalana Resource model for mining dilution or ore loss. At the proposed cut-offs, the Resource has about a 14% increase in mass but a 6% increase in metal (i.e. overall 7% reduction in grade).

The Kalanako model was re-blocked to combine ore sub-blocks with adjacent waste sub-blocks into a single block of 5 mE x 5 mN x 2.5 mRL. This formed a suitable selective mining unit (SMU) for the anticipated equipment. At the proposed cut-offs, this equates to a mass reduction of about 4% and metal reduction of 7% (i.e. 3% reduction in grade).

15.2.4 Slope angles

Table 15.2-1 summarises the slope angles used in the pit optimisation by weathering. These were based on the inter-ramp pit design parameters with a reduction to account for potential ramp locations.

Table 15.2-1 Pit Optimisation Slope Angles by Weathering

Weathering	Footwall (°)	Hangingwall (°)
Laterite	28.5	28.5
Saprolite	28.5	28.5
Saprock	42.0	45.0
Fresh	48.0	52.0

15.2.5 Processing Rate and Recovery

Table 15.2-2 summarises the processing rates and metallurgical recoveries used in the pit optimisation.

Table 15.2-2 Pit Optimisation Processing Rate and Recovery by Weathering

Weathering	Rate (Mdt/a)	Recovery (%)
Laterite	4.0	96.0
Saprolite	4.0	96.0
Saprock	3.6	90.0
Fresh	3.0	88.6

15.2.6 Mining Costs

Table 15.2-3 summarises the regression formulas used to calculate the mining cost. The mining costs were based on costs supplied by Endeavour. For Kalana the costs were split at a reference level and broken into two regressions to more accurately define the curve.

Table 15.2-3 Pit Optimisation Mining Costs by Weathering

Weathering	Above Kalana Reference Level (US$/t rock)	Kalana Reference Level (mRL)	Below Kalana Reference Level ($/t rock)	Kalanako (US$/t rock)
Laterite	0.005597 * level - 0.279385	380	-0.003435 * level + 3.192512	1.53
Saprolite	0.003042 * level + 0.680004	370	-0.002915 * level + 2.925194	-0.0051 * level + 3.495
Saprock	0.004182 * level + 0.784695	310	-0.006834 * level + 4.176246	-0.0045 * level + 3.424
Fresh	-0.0026 * level + 3.557655	210	-0.005382 * level + 4.325476	-0.0053 * level + 4.02

15.2.7 Ore Costs

Table 15.2-4 summarises the ore costs applied in the pit optimisation. Costs that differ for Kalanako are shown in brackets.

Table 15.2-4 Pit Optimisation Ore Costs by Weathering

Weathering	Laterite (US$/t)	Saprolite (US$/t)	Saprock (US$/t)	Fresh (US$/t)
Processing	11.55/ (11.30)	11.55 / (11.30)	14.20	15.89
Incremental ore cost	-0.48 / (0.25)	-0.34 / (0.31)	-0.18 / (0.29)	-0.02 / (0.37)
Ore haulage	0.80	0.80	0.80	0.80
ROM rehandle	0.42	0.42	0.42	0.42
Ore control	1.31	1.31	1.31	1.31
Process G&A	2.27	2.27	2.52	3.02
Non-process G&A	2.42 /(2.3)	2.42 / (2.37)	2.69	3.23
Total ore cost	**18.29 / (18.72)**	**18.43 / (18.78)**	**21.76 / (22.23)**	**24.65 / (25.04)**

15.2.8 Other Costs

A rehabilitation cost of US$0.06/dmt was included for closure.

A royalty of 3% and stamp duty of 0.6% was applied to all sales based on the price.

A transport and refining charge of US$4/oz was also applied.

15.2.9 Price and Discounting

A gold price of US$1,500/oz was used.

A discount rate of 5% was applied with revenue and cost being discounted by depth using a vertical advance rate of 40 m per year.

15.2.10 Cut-off Grades

Using the parameters and modifying factors outlined above, Table 15.2-5 shows the marginal cut-off grade by weathering for both deposits.

Table 15.2-5 Cut-off Grade by Weathering

Weathering	Kalana (g/t)	Kalanako (g/t)
Laterite	0.41	0.42
Saprolite	0.41	0.42
Saprock	0.52	0.53
Fresh	0.60	0.61

15.3 Pit Optimisation

Pit optimisations were run separately on Kalana and Kalanako using the standard Lerch-Grossman (LG) algorithm to produce incremental pit shells (nested) based on varying the input price. These nested pit shells were used in selecting the ultimate pits and guiding the location of pushbacks / stages for both deposits.

Figure 15.3.1 and Figure 15.3.2 summarise the pit optimisation physicals by the input price for Kalana and Kalanako respectively. There is a noticeable drop in mass around the US$1,400/oz for Kalana and around the US$1,100/oz for both deposits.

Figure 15.3.1 Kalana Pit Optimisation Physicals

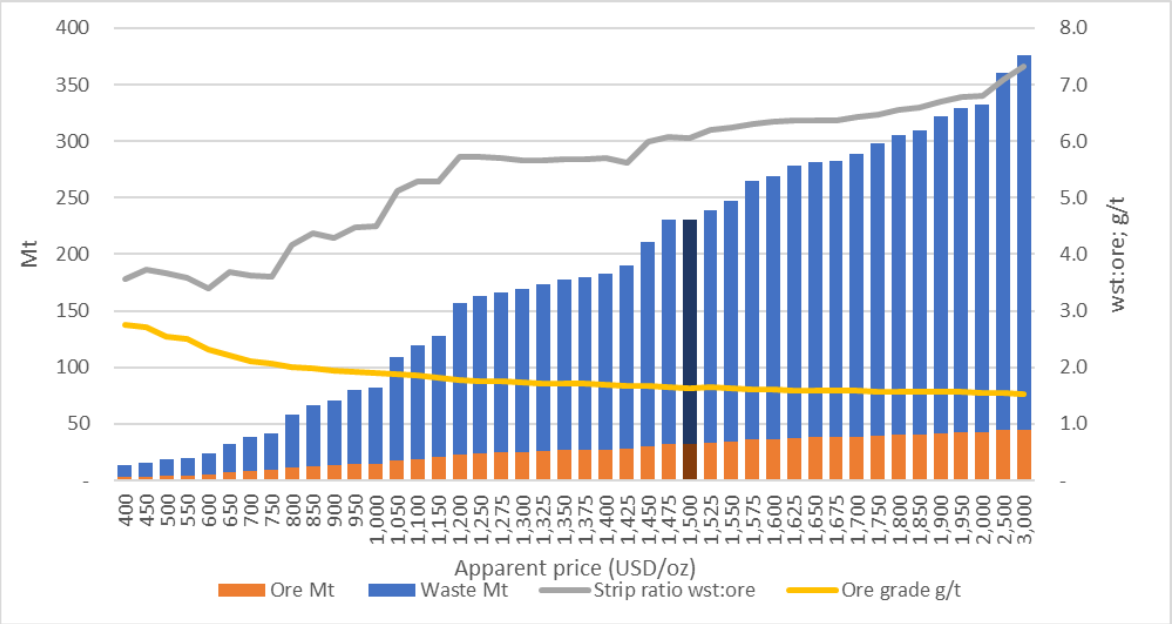


Figure 15.3.2 Kalanako Pit Optimisation Physicals

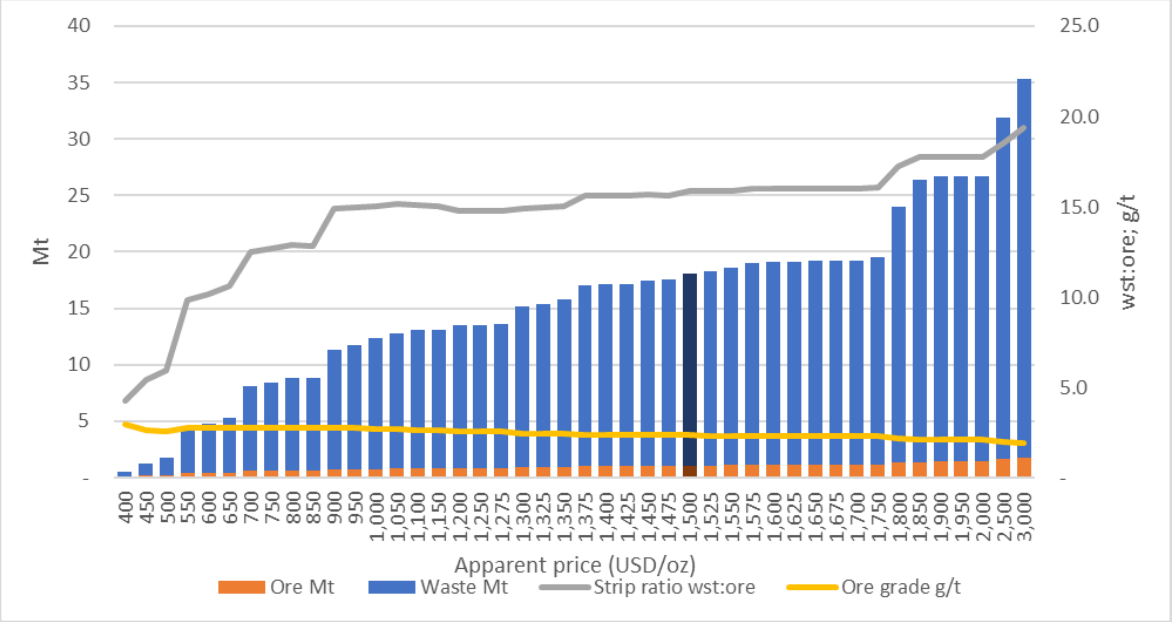


Figure 15.3.3 and Figure 15.3.4 summarise the pit optimisation economic values by the input price for Kalana and Kalanako respectively. Costs and revenues rise as the ore and waste quantities increase with input price. The undiscounted cash flows and indicative present values (IPV) (i.e. discounted cash flow assuming consistent mining and processing rates) rise before plateauing and finally falling. The peak IPVs occur in these plateaus at US$1,300/oz for Kalana (~2% higher than US$1,500/oz) and US$1,500/oz for Kalanako. This indicates that a similar financial result could be achieved with smaller pit shells. This would not however maximise the available Resource.

Figure 15.3.3 Kalana Pit Optimisation Values

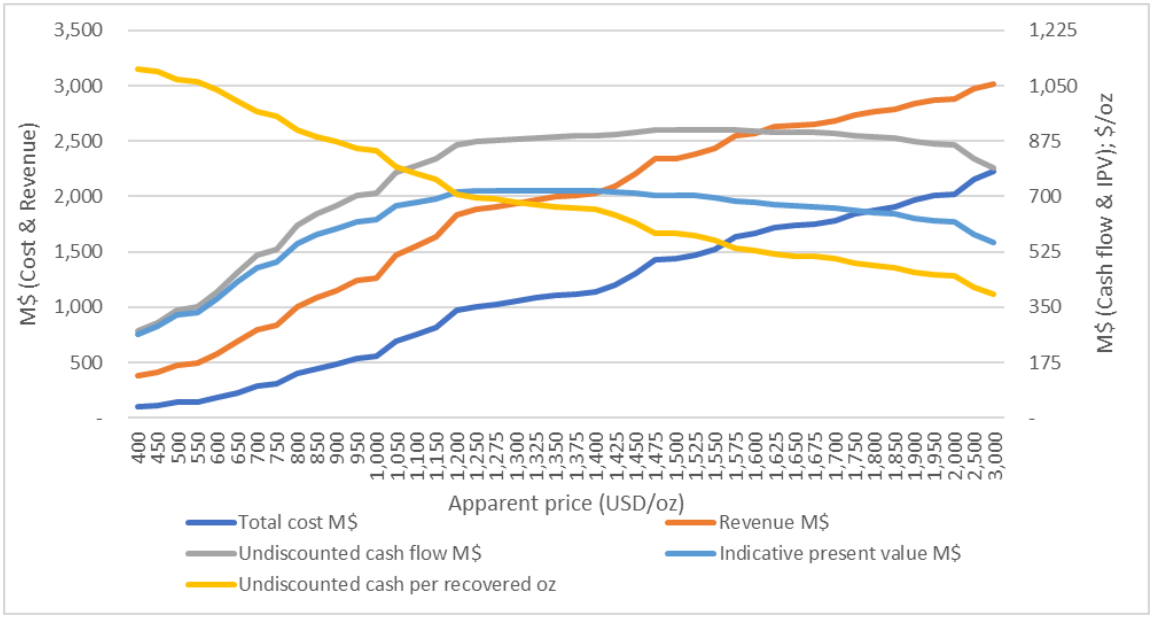


Figure 15.3.4 Kalanako Pit Optimisation Values

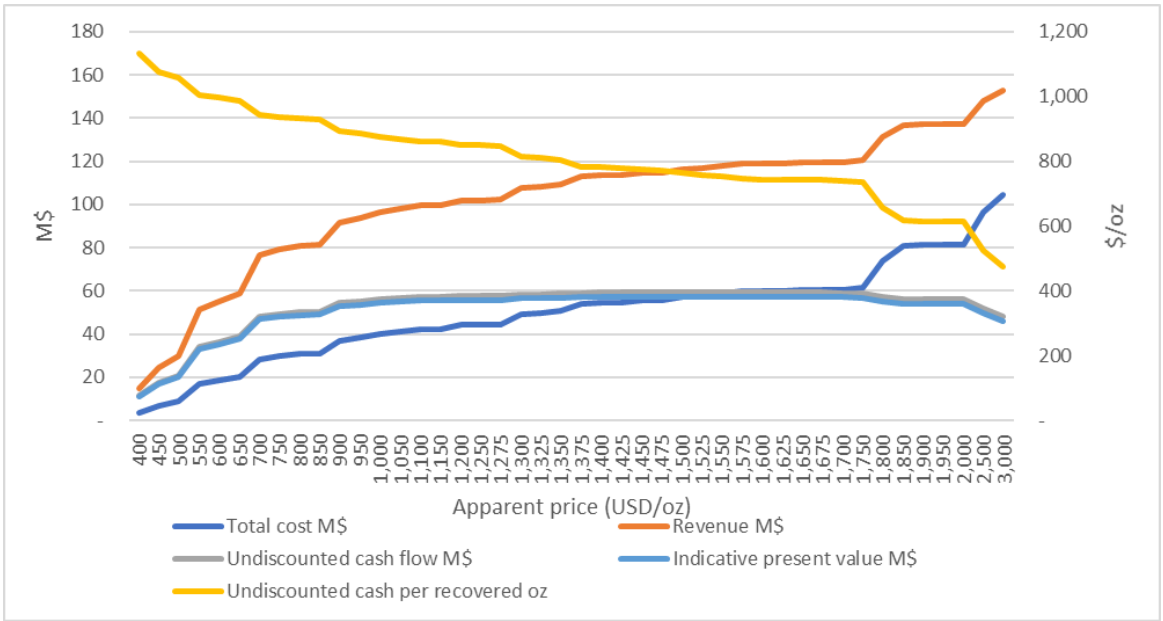


Table 15.3-1 and Table 15.3-2 summarise the pit optimisation results for Kalana and Kalanko, respectively.

Table 15.3-1 Kalana Pit Optimisation Summary

Pit Shell	Units	3	8	13	18	28	44
Revenue factor		0.3	0.5	0.67	0.83	1.00	2.00
Apparent gold price	US$/oz	500	750	1,000	1,250	1,500	3,000
Pit bottom	mRL	285	250	205	130	55	0
Physicals							
Ore	Mt	4.1	9.1	15.0	24.2	32.7	45.2
Waste	Mt	15.1	32.7	67.5	138.7	198.2	330.7
Total	Mt	19.3	41.8	82.5	162.9	230.9	375.9
Strip ratio	waste:ore	3.7	3.6	4.5	5.7	6.1	7.3
Diluted gold grade	g/t	2.55	2.06	1.90	1.77	1.64	1.53
In-situ gold	Moz	0.34	0.60	0.92	1.37	1.73	2.22
Recovered gold	Moz	0.32	0.56	0.84	1.25	1.56	2.00
Metallurgical recovery	%	94.1	93.1	92.0	91.1	90.6	90.2
Economics							
Mining cost	US$ M	39	89	186	396	600	1,057
Mining cost	US$/t	2.04	2.14	2.26	2.43	2.60	2.81
Ore cost	US$ M	79	183	318	538	744	1,051
Ore cost	US$/t ore	19.10	20.15	21.22	22.20	22.75	23.25
Selling costs	US$ M	18	32	49	73	91	117
Selling costs	US$/recovered oz	58	58	58	58	58	58
Total cost	US$/in-situ oz	404	507	604	731	831	998
Revenue at US$1,500/oz	US$ M	477	839	1,265	1,881	2,345	3,014
Undiscounted cash flow	US$ M	341	535	712	874	911	789
Indicative present value	US$ M	324	491	627	718	703	556

Table 15.3-2 Kalanako Pit Optimisation Summary

Pit Shell	Units	3	8	13	18	28	44
Revenue factor		0.3	0.5	0.67	0.83	1.00	2.00
Apparent gold price	US$/oz	500	750	1,000	1,250	1,500	3,000
Pit bottom	mRL	335	295	295	290	290	230
Physicals							
Ore	Mt	0.2	0.6	0.8	0.9	1.1	1.7
Waste	Mt	1.5	7.8	11.6	12.7	17.0	33.6
Total	Mt	1.7	8.5	12.3	13.5	18.1	35.3
Strip ratio	waste:ore	6.0	12.7	15.0	14.8	15.9	19.4
Diluted gold grade	g/t	2.59	2.77	2.71	2.57	2.36	1.93
In-situ gold	Moz	0.02	0.06	0.07	0.07	0.08	0.11
Recovered gold	Moz	0.02	0.05	0.06	0.07	0.08	0.10
Metallurgical recovery	%	96.0	95.7	95.6	95.6	95.5	94.7
Economics							
Mining cost	US$ M	3	15	22	24	32	65
Mining cost	US$/t	1.72	1.76	1.77	1.77	1.78	1.83
Ore cost	US$ M	5	12	15	16	20	34
Ore cost	US$/t ore	18.78	18.90	18.92	18.94	18.98	19.62
Selling costs	US$ M	1	3	4	4	4	6
Selling costs	US$/recovered oz	58	58	58	58	58	58
Total cost	US$/in-situ oz	425	539	597	622	703	973
Revenue at US$1,500/oz	US$ M	30	79	96	102	116	153
Undiscounted cash flow	US$ M	21	49	56	58	59	48
Indicative present value	US$ M	20	48	54	56	57	46

Sensitivities were also applied to input parameters, varying them by ±20%, to assess the impact on the pits. When considering the impact on cash flow, flexing the gold price (which is also equivalent to metallurgical recovery) resulted in about a 50% change in value whereas variation to other inputs resulted in changes of less than 20% impact on value (ore costs ~17%, dilution ~15%, mining costs ~14%, slope angles ~11% and selling costs ~2%).

15.4 Mine Design

15.4.1 Design Parameters

The design criteria listed in Table 15.4-1 was used for the pit designs. The minimum width for all stages was 40 m with a minimum mining width of 20 m.

Table 15.4-1 Pit Design Parameters by Weathering

Parameter	Laterite/Saprolite		Saprock		Fresh	
	Footwall	Hangingwall	Footwall	Hangingwall	Footwall	Hangingwall
Batter angle (°)	45	45	65	75	70	80
Batter height (vertical m)	10	10	10	10	10	10
Batter berm interval (vertical m)	10	10	10	10	10	10
Berm width (m)	7	7	5	5	5	5
Inter-ramp slope (toe to toe, no ramp) (°)	30.5	30.5	46.0	52.5	49.2	55.9
Geotechnical berm interval (vertical m)	Base of saprolite	Base of saprolite	100	100	100	100
Geotechnical berm width (m)	10	10	10	10	10	10
Dual ramp width (m)	22	22	22	22	22	22
Single ramp width (m)	15	15	15	15	15	15

Table 15.4-2 shows the dump and LTSP (long term stockpile) design parameters used. The as-dumped parameters reflect the offsets required to meet the rehabilitated parameters when considering the dumped batter angle (natural angle of repose).

Table 15.4-2 Dump and LTSP design parameters

Parameter	As-dumped	Rehabilitated
Batter angle (°)	35	18
Lift height (m)	5	5
Berm interval (vertical m)	15	15
Berm width (m)	30.9	6.1
Overall slope (toe to toe, no ramp) (°)	16	16

15.4.2 Ultimate Pit

Figure 15.4.1 and Figure 15.4.2 show the ultimate pit designs for Kalana and Kalanako, respectively. The Kalana pit is predominately in fresh rock but also has a significant proportion in the saprolite rock. Whilst the Kalanako pit is almost entirely within the saprolite rock.

Figure 15.4.1 Kalana Ultimate Pit Design

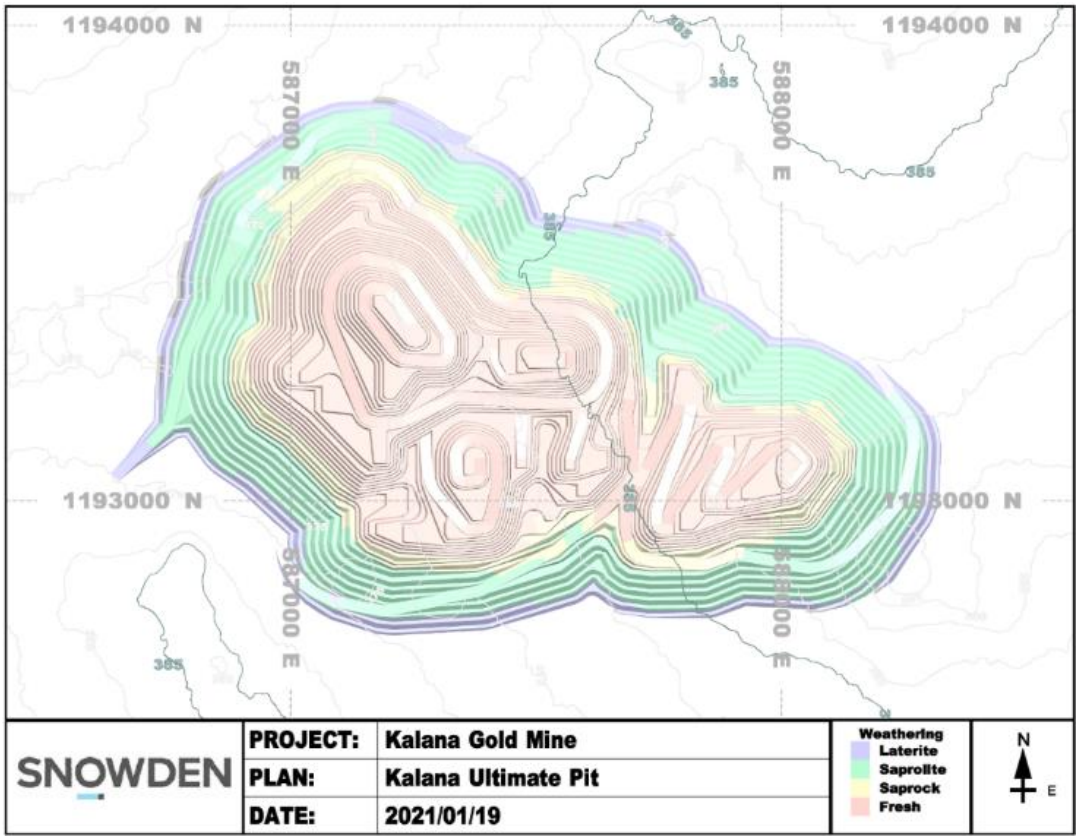
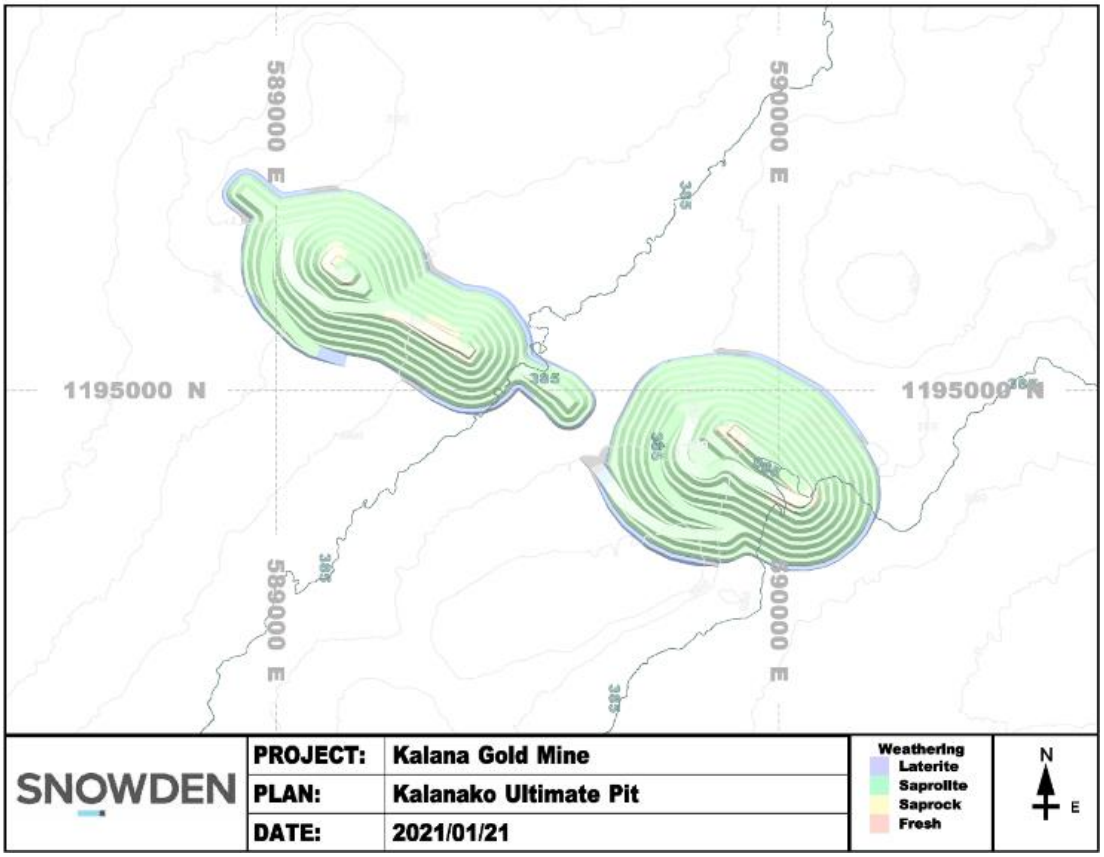


Figure 15.4.2 Kalanako Ultimate Pit Design



15.4.3 Stage Pit Designs

Figure 15.4.3 and Figure 15.4.4 show the stage designs for Kalana and Kalanako. Kalana comprises six stages with the earlier stages focused on accessing higher value ore. Kalanako consists of two independent stages that mine to the final pit limits.

Figure 15.4.3 Kalana Stage Pit Designs

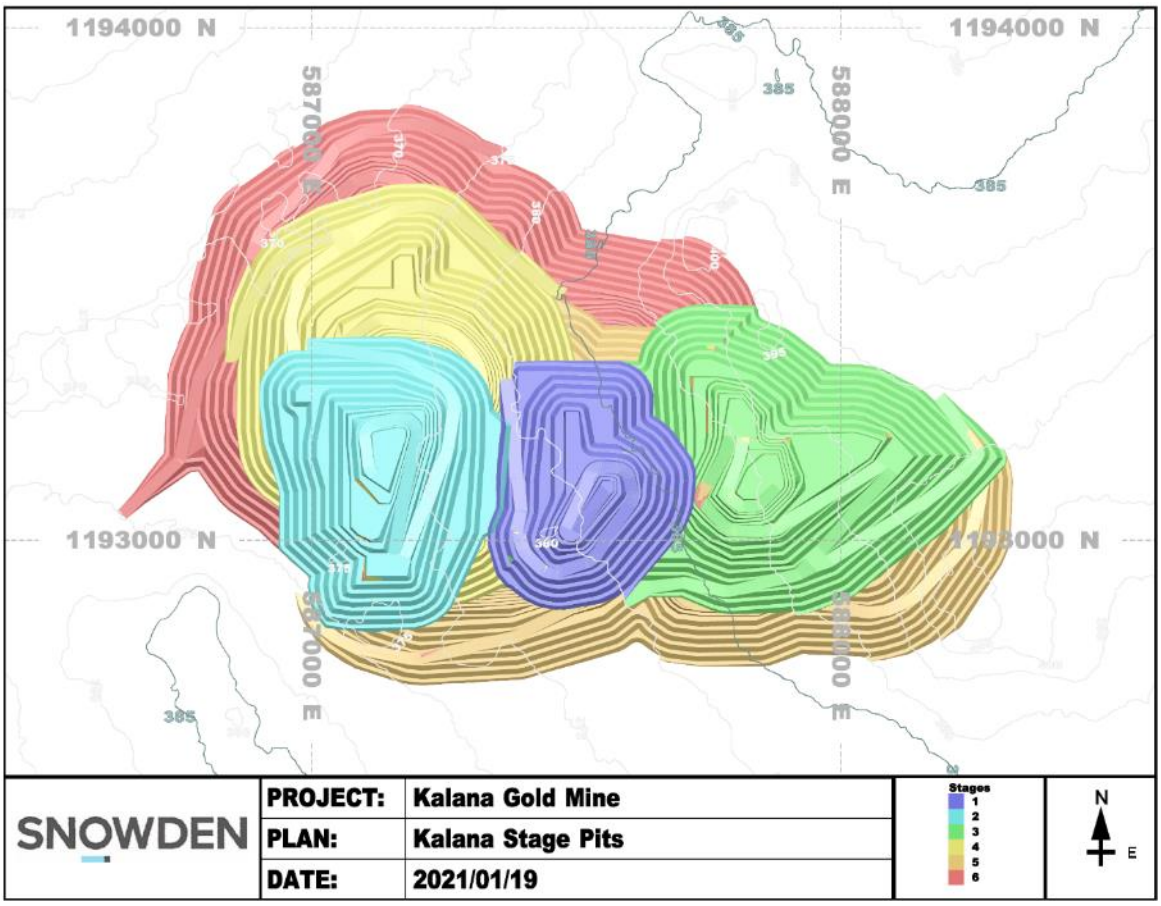
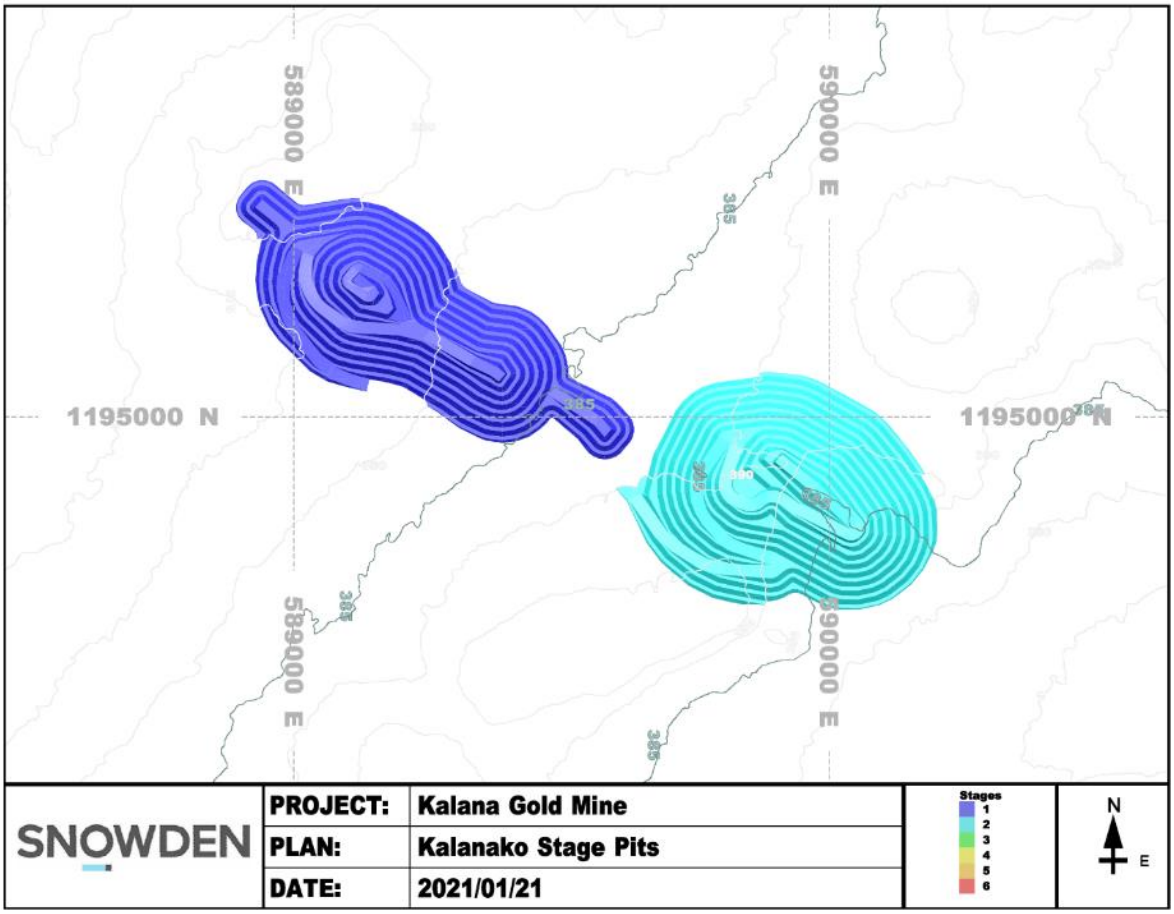


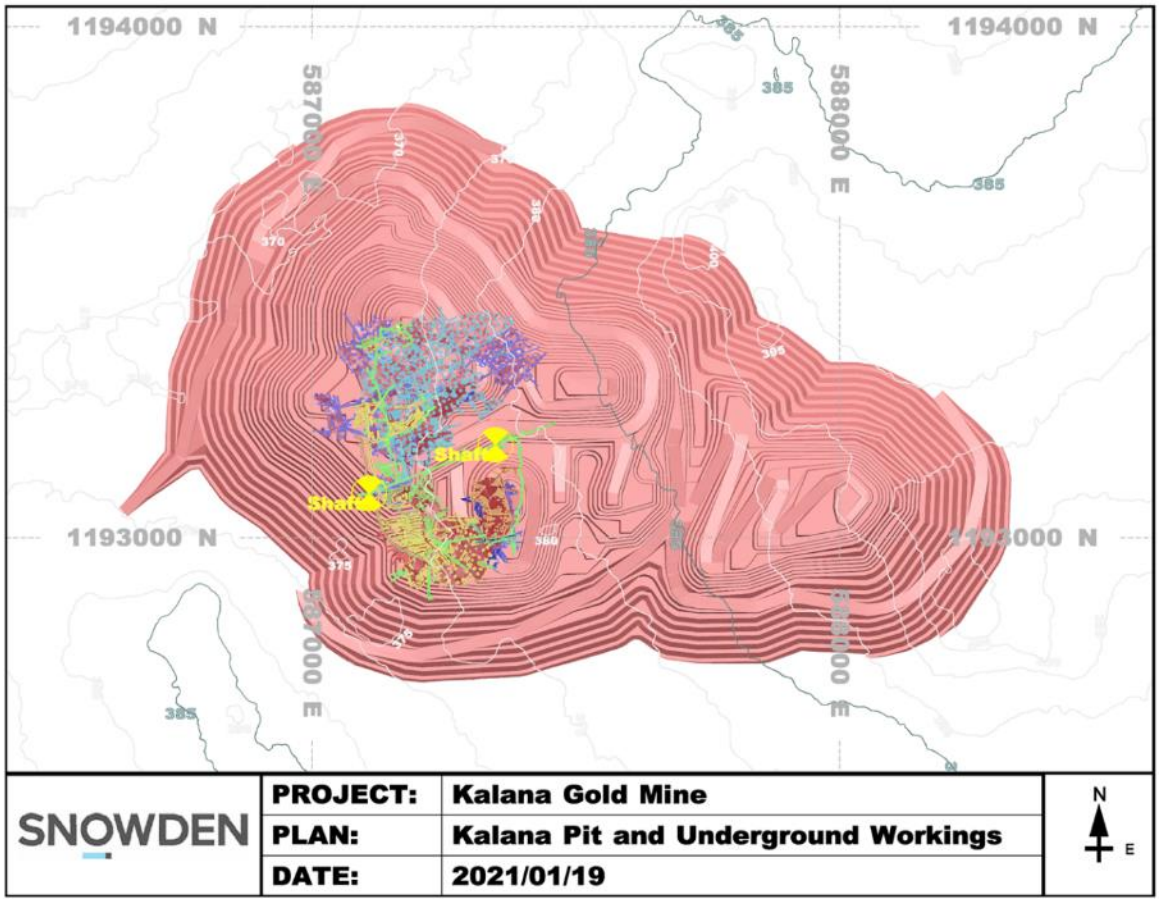
Figure 15.4.4 Kalanako Stage Pit Designs



15.4.4 Underground Workings

Kalana has previously been mined underground and the open pit will intersect these workings. Figure 15.4.5 shows the underground workings inside the ultimate pit. The workings are in the fresh rock except for the barrels of the two shafts. Both shaft barrels are mined by Stage 2.

Figure 15.4.5 Kalana Underground Workings

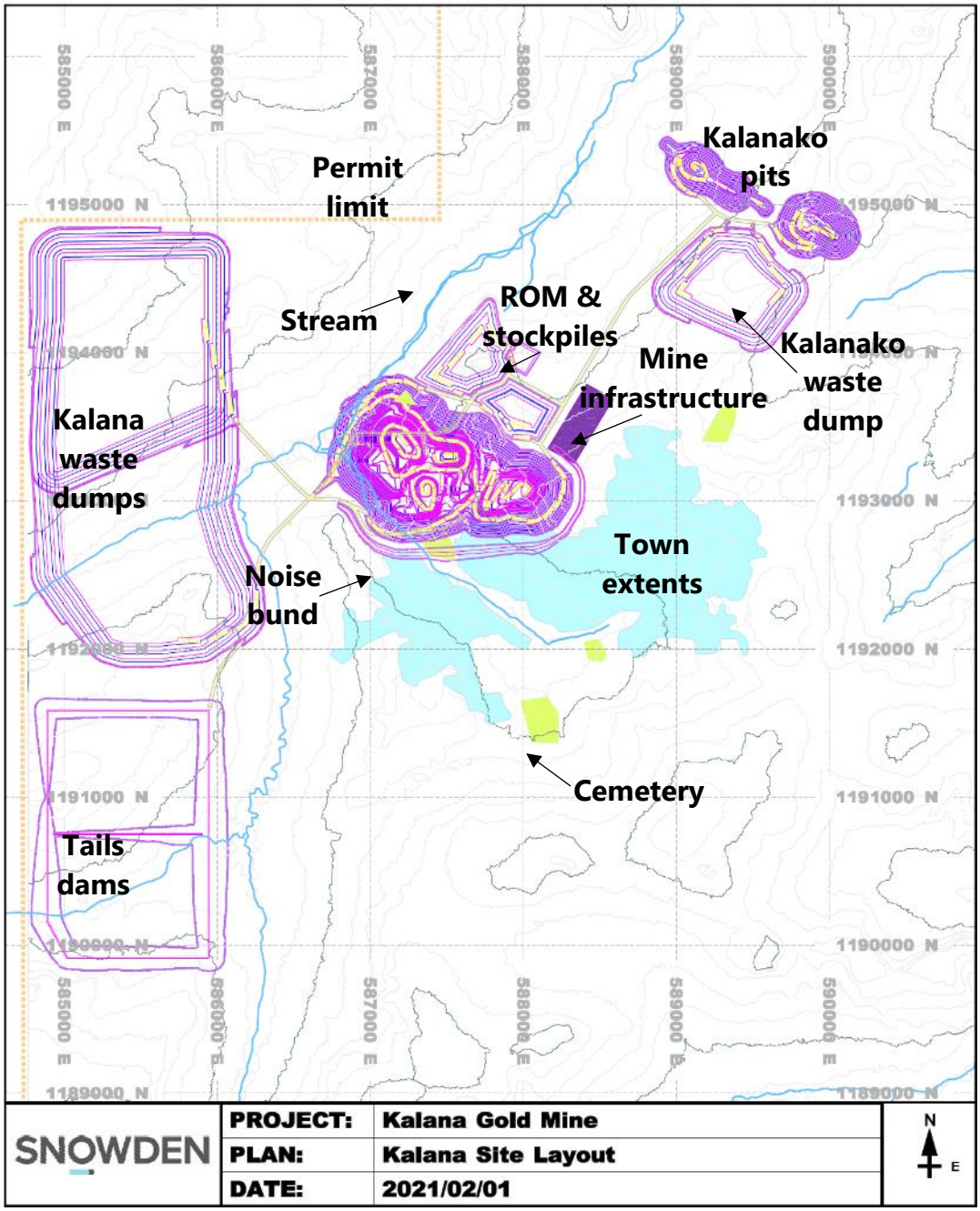


15.4.5 Site Layout

Figure 15.4.6 shows the overall Kalana project site layout. The main features are:

- Kalana waste dump and tails dams are to the west of the open pit.
- Kalanako waste dump is to the west of the open pits.
- Kalana open pit encroaches on the town and cemeteries. A bund constructed out of waste is likely to be required to reduce impacts on the town.
- Kalana open pit, waste dump and tails dams intersect nearby streams that will require diversion drains.
- ROM and long term ore stockpiles are located between the two deposits.
- Mine infrastructure is located close to the open pit and processing plant.

Figure 15.4.6 Site Layout



15.5 Mining Quantities and Mineral Reserve Estimate

The Kalana and Kalanako open pits and historical tails dams contain 35.6 Mt at 1.60 g/t, inclusive of dilution and mining loss, derived from an Indicated Mineral Resource which is classified as a Probable Mineral Reserve as summarised in Table 15.5-1.

Table 15.5-1 Kalana and Kalanako Mineral Reserve Estimate (December 2020)[1]

Deposit	Item	Proved	Probable	Total Mineral Reserve
Kalana	Ore (Mt)	-	33.6	33.6
	Gold (g/t)	-	1.58	1.58
	Contained gold (Moz)	-	1.71	1.71
Kalanako	Ore (Mt)	-	1.2	1.2
	Gold (g/t)	-	2.21	2.21
	Contained gold (Moz)	-	0.09	0.09
Tailings	Ore (Mt)	-	0.82	0.82
	Gold (g/t)	-	1.67	1.67
	Contained gold (Moz)	-	0.04	0.04
Total	**Ore (Mt)**	**-**	35.6	35.6
	Gold (g/t)	**-**	1.60	1.60
	Contained gold (Moz)	**-**	1.83	1.83

1 Some numbers may not sum correctly due to rounding

15.5.1 Risks and Opportunities

The Mineral Reserve statement for the Kalana Gold Deposit dated December 2020 are stated in accordance with the CIM guidelines. There are risks associated with achieving the stated mining outcomes should the underlying assumptions change. The main risks and opportunities include:

- Changes in gold price (or metal recovery) have a high impact on undiscounted cash flow which would result in smaller pits. A 20% change in price would change the Reserve ounces by a similar percentage.
- Changes in other parameters (such as mining costs, ore costs, slope angles and dilution) will impact the undiscounted cash flow and pit size. A 20% change in any of these would change the Reserve ounces by around 15%.
- The proximity of the Kalana township may impact negatively on the operation. Factors that could influence this include blasting impacts, noise levels, land access and water access and quality. Existing proposals appear sufficient but will need to be continually reassessed to ensure the social licence is maintained.
- There is an Inferred Mineral Resource, above the marginal cut-off grade and inside the final Kalana pit design could if realised, produce an additional 14,000 oz in the mine schedule.

16.0 MINING METHODS

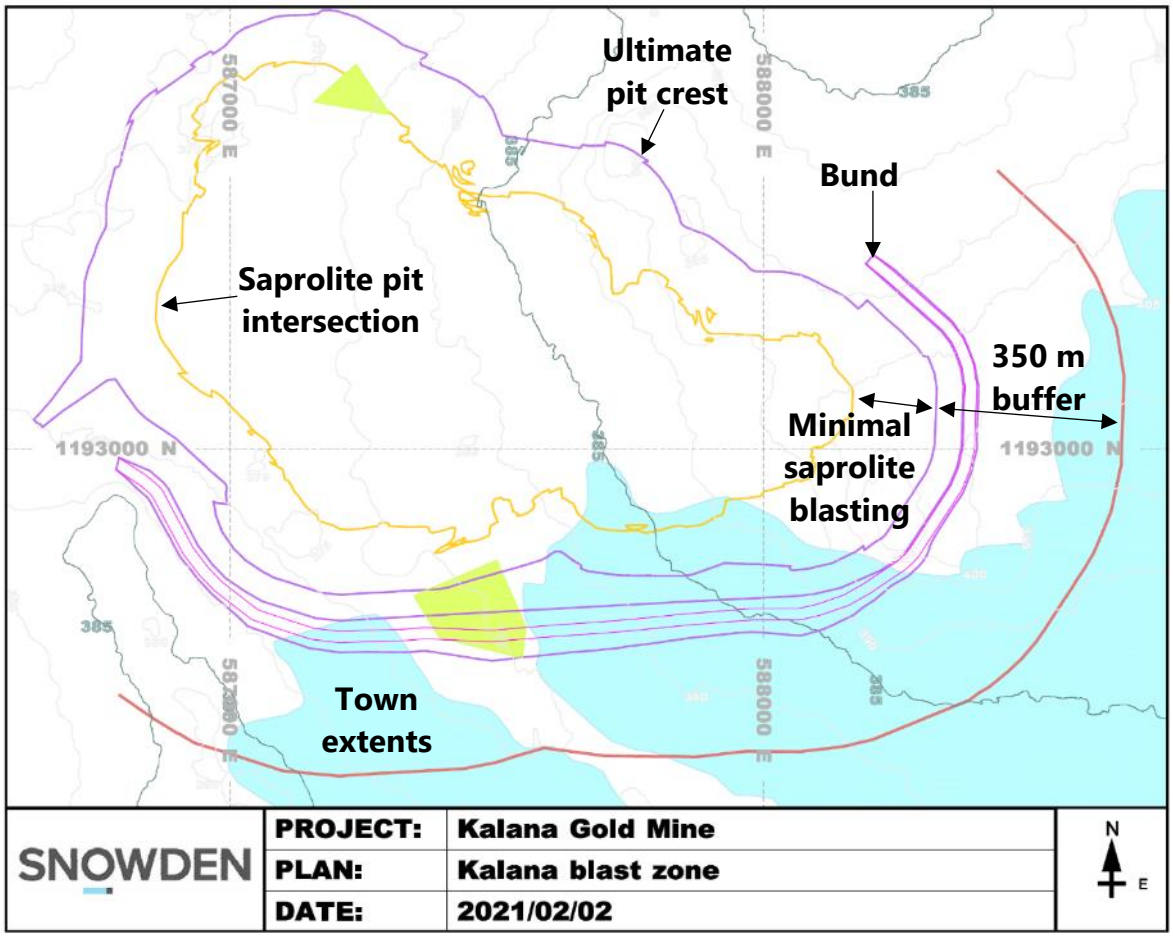
16.1 Mining Methods

The most recent mining at Kalana was underground, however to achieve the scale of mining required by the proposed processing plant the mining method going forward for the Kalana Gold Project is intended to be conventional open pit mining using a drill, blast, load, haul and tip mining cycle. Endeavour plans to contract out the mining to a suitable mining contractor whilst maintaining operation oversight.

The saprolite is anticipated to be primarily free-dig, potentially requiring ripping or only some very light drill and blast. Production drilling will be undertaken by top hammer drills drilling 127 mm diameter holes. As rock strengths increase, blasting will be utilised more regularly in the saprock with powder factors estimated at 0.38 kg/bcm. All the fresh rock will be blasted with powder factors estimated at 0.86 kg/bcm.

A blast zone buffer between the town and the open pit is proposed that will require sections of the town to be relocated. Figure 16.1.1 shows the proposed 350 m blasting buffer. The proposed bund and minimal blasting in saprolite may also act as an additional 70 m to 150 m buffer (dependent on ground conditions). Kalanako is over 1.3 km from the township and 800 m from the proposed bypass road and should not require a blast zone buffer.

Figure 16.1.1 Kalana Blasting Buffer



The underground voids are present and were surveyed, however despite the voids being typically narrow (<3 m) and flat dipping, drilling should be used to confirm their locations. As a minimum the grade control drilling could be utilised, although dedicated probe drilling would be more suitable. Safety procedures will be adopted to protect equipment and personnel when working on floors with minimal thickness to underground workings.

Loading will be undertaken by hydraulic excavators (100 t and 200 t operating weights) to provide a balance between mining selectively and productivity.

Hauling will be undertaken by rigid body dump trucks (90 t capacity).

- Ore will be tipped on:
  - Strategic long-term (LT) stockpiles to increase the plant feed grade and for rehandle during later stage pre-strip.
  - Run-of-mine (ROM) stockpiles for short-term rehandle to the crusher.

- Waste will be tipped on:
  - External waste dumps.
  - Bund and road construction.
  - Tails dam wall construction.

The primary mining equipment will be supported by suitably sized ancillary and support equipment such as, but not limited to, dozers, graders, water carts and wheel loaders. This equipment will be used for activities such as:

- Clearing and stripping of topsoil.
- Construction of haul roads and ramps (temporary and long-term).
- Pit, stockpile and dump floor maintenance.
- Ripping of free-dig material, if required.
- Dump face reshaping for rehabilitation, topsoil spreading, ripping, and seeding.
- Drill pattern preparation.
- Clean-up of spillage around pit, stockpile and dump working areas and haul roads and ramp.
- Stockpile rehandle.
- Dust suppression on roads, loading and tipping areas.
- Fire fighting.

16.2 Hydrogeology

The hydrogeology of the Kalana deposit is described in the PFS report prepared by Future Flow (2015). One main aquifer occurs in the area, associated with the fractured, weathered and fresh rock material. This aquifer is overlain by an aquitard that comprises the completely weathered, low permeability saprolite material. This reduces the hydraulic connection between surface water bodies and the underlying fractured rock aquifer. The laterite may also form a minor, local and seasonal aquifer.

Groundwater inflows to the underground mine from the lower aquifer are associated with the secondary fracturing in the competent rock and as such will be along discrete pathways associated with these fractures. Fault zones, fractures in the competent rock and contact zones bounding the diorite intrusives can be a significant source of groundwater depending on whether the fractures have been filled with secondary mineralisation. This information indicates that groundwater flows to the open pit will be structurally controlled.

Results from the drilling program that formed part of this PFS show regular fracturing and significant groundwater yields extend to a depth of 150 mbgl (the deepest that test bores were drilled). This indicates that the active aquifer extends to at least 150 m depth below surface.

The underground mine pumps water from No. 1 Shaft (100 m depth), No. 2 Shaft (180 m depth). Workings extending to 230 m depth are dewatered by secondary pumps. Groundwater inflows pumped from the mine reportedly total approximately 2 x 106 m^3/y (5,500 m^3/day); most inflows are associated with mine-scale faults. The inflows to the underground mine indicate that at least within the deposit, the aquifer extends substantially deeper than 150 m.

Groundwater conditions observed by Kalana personnel during the geological investigation drilling programs were reported as follows:

- Groundwater level in the residual / saprolite units is approximately 10 m below surface.
- "Normal" groundwater level in the fresh rock aquifer is close to the saprolite / fresh rock transition, at 50 m to 70 m below surface.
- Significant drawdown has occurred within the fresh rock aquifer due to mining activities.

Future Flow (2015) report the following pertinent information in relation to pit slope stability assessments for the proposed mine:

- Proposed dewatering of the pit by pumping from out-of-pit boreholes will substantially reduce pit inflows and lead to an extensive drawdown cone extending well outside the pit footprint, partially de-pressurising the deposit and pit walls.
- Transmissivity values from borehole pump testwork:
  - 'Weathered' aquitard material (saprolite / saprock): 0.1 m^2/day (estimated); assuming an average thickness of this unit of 80 m indicates permeability of approximately 0.001 m/day (1 x 10^{-8} m/s).
  - Fractured rock aquifer: 14 to 94 m^2/day (values vary depending on test location and analysis method).

- The underground mine workings will fill with water once the pumps are turned off, and groundwater conditions within the deposit will partially re-establish themselves prior to commencement of open pit mining.
- Annual drawdown profiles have been developed based on the proposed pit mining sequence.

16.3 Geotechnical Evaluation

16.3.1 Kalana

The geotechnical assessment work program for Kalana has included:

- Advising on geotechnical investigations, including logging, sampling and testing programs.
- Geotechnical investigations undertaken by the Kalana geological department, including:
  - geotechnical and structural core logging and sampling of nine diamond-core exploration holes
  - geotechnical mapping of underground workings
  - geomechanical laboratory test programs.
- Collating and processing the geotechnical and structure logging, mapping and testing data.
- Developing geotechnical and rock mass structure models.
- Assessing hydro-geological conditions pertaining to the pit slopes.
- Undertaking pit slope, trafficability, excavatability and blastability design studies.
- Assessing key geotechnical risks to the project and advising on mitigation strategies.

Within the deposit, mine-scale structural features affecting slope stability are limited to northwest trending strike-slip faults, with either a low, easterly dip or aligned sub-parallel to bedding. Bedding is the dominant rock mass structure striking roughly north-northwest to south-southeast and dipping between 60° and 90° towards the east. The rock mass structure data has been spatially allocated to the three principal structural domains identified by earlier geological investigations:

- Kalana 1 North (Kalana West) – south east dipping vein system
- Kalana 1 South (Kalana Central) – east dipping vein system
- Kalana 2 (Kalana East) – flat to south to south-east dipping vein system.

Weathering has occurred across the deposit to depths ranging from 30 m to 130 m below surface, forming the following zones that provide the basis for the principal geotechnical material domains:

- Laterite: Typically, 3 m to 5 m thickness, locally up to 10 m thick.
- Saprolite: Completely weathered material typically 40 m to 80 m thick, with relic structure observed in some sections (bedding, veins).
- Saprock: Highly to slightly weathered rock ranging from 10 m to 40 m thickness; subdivided for this assessment into Saprock 1 (highly weathered) and Saprock 2 (moderately to slightly weathered) domains.
- Fresh Rock: Unweathered meta-sediments and diorite.

Slope stability studies have utilised empirical, kinematic, limit equilibrium and numerical modelling analysis techniques to assess the stability of mine slopes at batter, inter-ramp and overall slope scales. These analyses incorporated probabilistic techniques to assess slope stability for the statistical range of key strength input parameters. In addition, groundwater drawdown models used for the analyses were developed using finite element modelling techniques.

Saprolite

Inter-ramp slope design recommendations for the Saprolite material are summarised in Table 16.3-1.

Table 16.3-1 Pit slope design recommendations – Saprolite

Slope Height	Maximum Inter-Ramp Slope Angle
40 m	42.5°
50 m	36.0°
60 m	32.5°
70 m	30.0°
80 m	27.5°

Batter design recommendations for the Saprolite are:

- Maximum batter slope angles:
  - 60° for 5 m height
  - 45° for 10 m height.

- Minimum berm widths:
  - 3.0 m for 5 m batters
  - 5.0 m for 10 m batters.
  - increase berm widths as required to meet inter-ramp angle recommendations.
- A 10 m wide geotechnical berm is recommended for the nominal base of the Saprolite horizon at a depth of 60 m below surface, then at 100 m vertical intervals in the absence of a ramp.

Saprock and Fresh Rock

Recommendations for inter-ramp and batter slope design parameters within the Saprock and Fresh Rock domains are summarised in Table 16.3-2.

Table 16.3-2 Inter-Ramp and Batter Slope Design Recommendations – Saprock and Fresh Rock

Material	Maximum slope height	Inter-ramp Slope Angles		Batter Face Angles	
		Walls facing eastward (west walls)	Walls facing all other orientations	Walls facing eastward (west walls)	Walls facing all other orientations
Saprock	30 m	46°	49°	65°	70°
Fresh Rock	100 m	54°	58°	75°	80°

Pit designs developed using the slope design recommendations have been reviewed using limit equilibrium analyses and empirical design charts to ensure overall slopes of the final pits will meet industry-standard design criteria and have adequate life-of-mine stability.

16.3.2 Kalanako

The geotechnical model for Kalanako can be summarised as representing the interaction or coupling of the four models:

- The upper slopes are anticipated to be excavated in the weak shear strength Saprolite
- Mass unit.
- The depth to the base of the Saprolite Mass Unit is variable across the deposit, however it is such that the majority of design slopes are anticipated to be excavated in this material.

- Seventeen major faults are identified and modelled, with three of these likely to influence slope stability at the stack slope scale. Local weakening of the rock masses in proximity to the faults due to fracturing, drag-folding and increased likelihood of groundwater storage could not be determined for this geotechnical design specification due to low quality of observational data.
- Structural fabric (bedding planes and joints) is anticipated to control slope stability in the Fresh Rock slopes. The incorporation of analyses for this could not be undertaken for this geotechnical design specification due to the nature of the orientation bias of the historical diamond drilling campaign.
- A pre-mining CE groundwater level of 15 mbgl has been adopted.
- An indicative rock mass shear strength has been developed for anticipated fresh rock lithologies.

16.4 Production Schedule

The schedule was completed in Snowden's Evaluator scheduling software. This is a Mixed Integer Linear programming-based tool with the objective to maximisation net present value, within a given tolerance, for the defined constraints (e.g. physical quantity, grade).

16.4.1 Parameters and Constraints

Material Types

The mining block model was coded with material types to allow ore and waste scheduling selectivity (e.g. stockpiling low grade). Table 16.4-1 shows the material types coded and their criteria. The grade ranges were based on splitting the ore into three roughly equal masses.

Table 16.4-1 Scheduling material types

Description	Material Type Code	Reserve Status	Gold Grade (g/t)
Waste	WASTE	Non-Reserve	Any
Low Grade	LG	Reserve	≥Cut-off and <0.9
Medium Grade	MG	Reserve	≥0.9 and <1.3
High Grade	HG	Reserve	≥1.3

Time scale

The schedule was completed in annual increments over the life of the project except for Year 1 which is a pre-production year and is equivalent to only three months.

Resolution

Based on the selected time scale, quantities were aggregated to 5 m high by stage and various groupings (e.g. material type, Resource class, weathering).

Precedencies

All benches within a stage were dependent on the bench above being mined out. In addition, benches from subsequent stages were prevented from mining below the current stage. This includes the old tailings dam which sits on top of Kalana stage 2.

Active mining areas

The number of active stages that can be mined in any period was not constrained.

Bench turnover

Bench turnover is used to assist in ensuring sufficient working space is available. It was restricted to 60 vertical metres per year for waste mining and 40 vertical metres per year for ore mining (except for benches in the lower size quartile which were weighted proportionally). This resulted in higher turnover rates at the top of the deposit (where there is more working space) and lower at the bottom (where working space is reduced). The average bench turnover was about 55 vertical metres per year.

Mining

The total mining was constrained based on the productivities in Table 16.4-2 and the number of available units in Table 16.4-3. The mining capacity was ramped up over three years (48%, 93%, 100% of capacity) to account for operator familiarisation with the equipment and site.

Table 16.4-2 Mining Productivities (t/h)

Rock Type	200 t Excavator		100 t Excavator	
	Ore	Waste	Ore	Waste
Laterite	1,109	1,706	771	1,187
Saprolite	1,109	1,706	771	1,187
Saprock	1,220	1,877	848	1,305
Fresh	1,018	1,567	586	902
Tailings	1,109	NA	771	NA

Table 16.4-3 Loading Units Available

	Years 1 to 10	Year 11 onwards
200 t excavator (#)	2	1
100 t excavator (#)	2	1
Capacity range based on productivity (Mt/a)	18.3 to 36.2	9.1 to 18.1

Processing

Table 16.4-4 summarises the annual processing rates which were varied based on the weathering. The process capacity was ramped up over two years starting in Year 2 (79%, 100% of capacity).

Table 16.4-4 Processing Throughputs

Rock Type	Throughput (Mt/a)
Laterite	4.0
Saprolite	4.0
Saprock	3.6
Fresh	3.0
Tailings	4.5

Product

The minimum amount of gold produced in the final schedule was constrained to produce a more consistent production profile. Between Years 2 and 8 this was 150 koz/a and 100 koz/a between Years 9 and 11.

16.4.2 Mining Schedule

Figure 16.4.1 shows the annual movements from the pits by stage. Mining peaks at around 30 Mdt/a with the variation largely due to the different productivities by weathering type (Figure 16.4.2). Mining typically occurs in multiple stages and is largely constrained by movement and bench turnover (i.e. maintaining sufficient working space).

Figure 16.4.1 Total Ex-Pit Movement by Stage

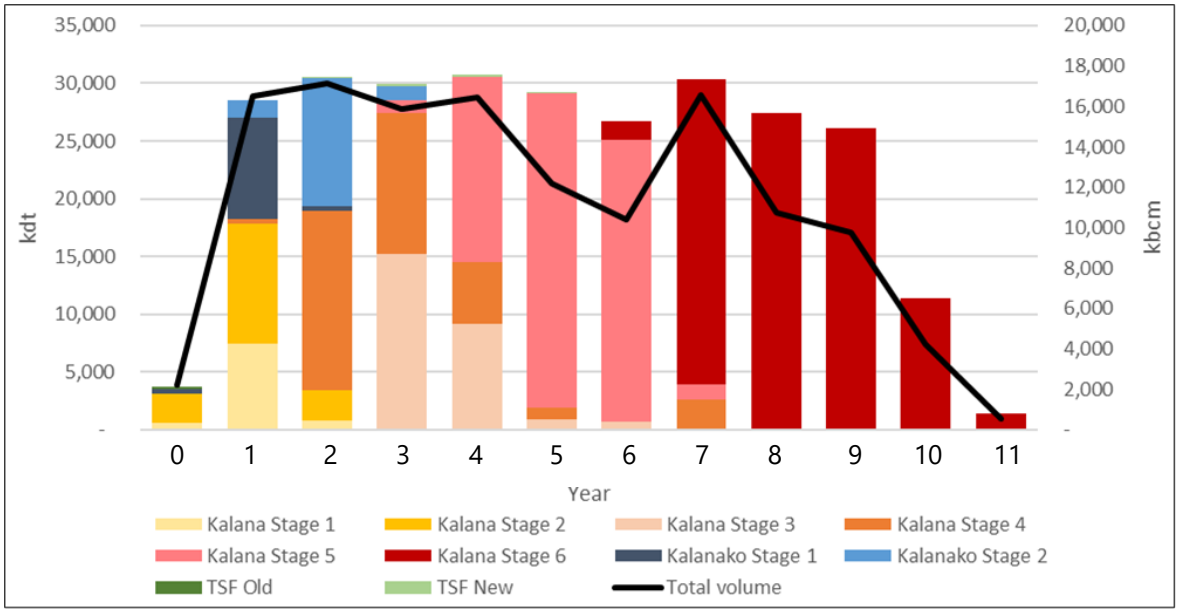


Figure 16.4.2 Total Ex-Pit Movement by Weathering

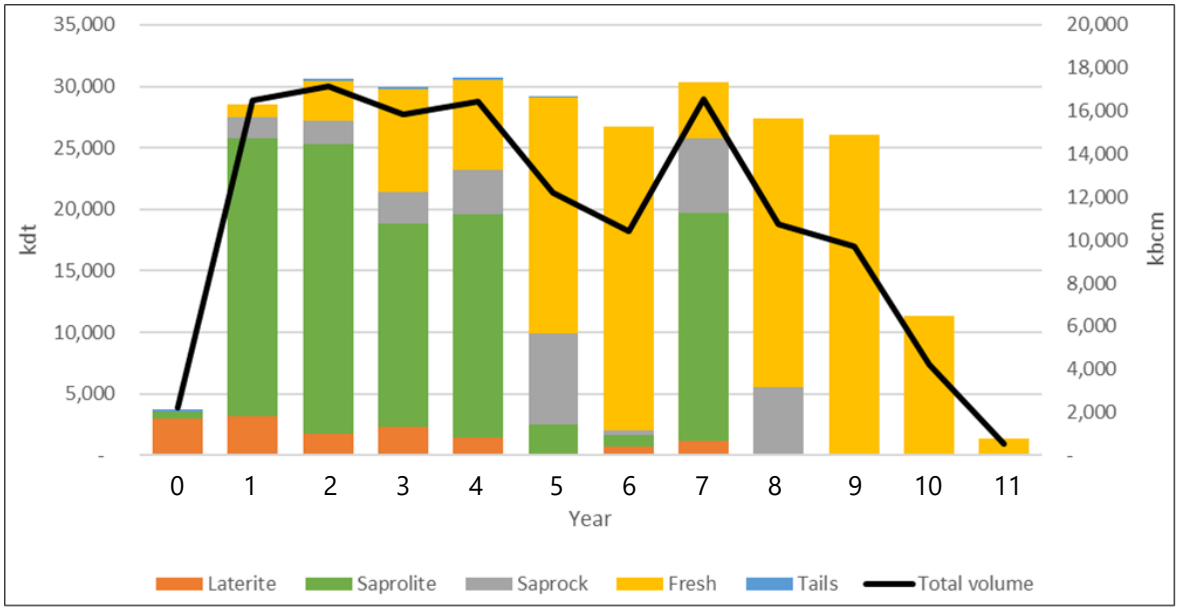


Figure 16.4.3 shows the annual movements from the pits by material type. Ore is mined consistently over the life of mine in quantities above the required processing rate. The lower grade ore is placed on long-term stockpiles (Figure 16.4.4) allowing the higher grades to be fed to the processing plant thereby increasing gold production and net present value. It also presents an opportunity if processing rates can be increased to feed additional ore or a risk mitigation should mining rates be lower than planned there is ore, albeit at a lower grade, that can be processed.

Figure 16.4.3 Total Ex-Pit Movement by Material Type

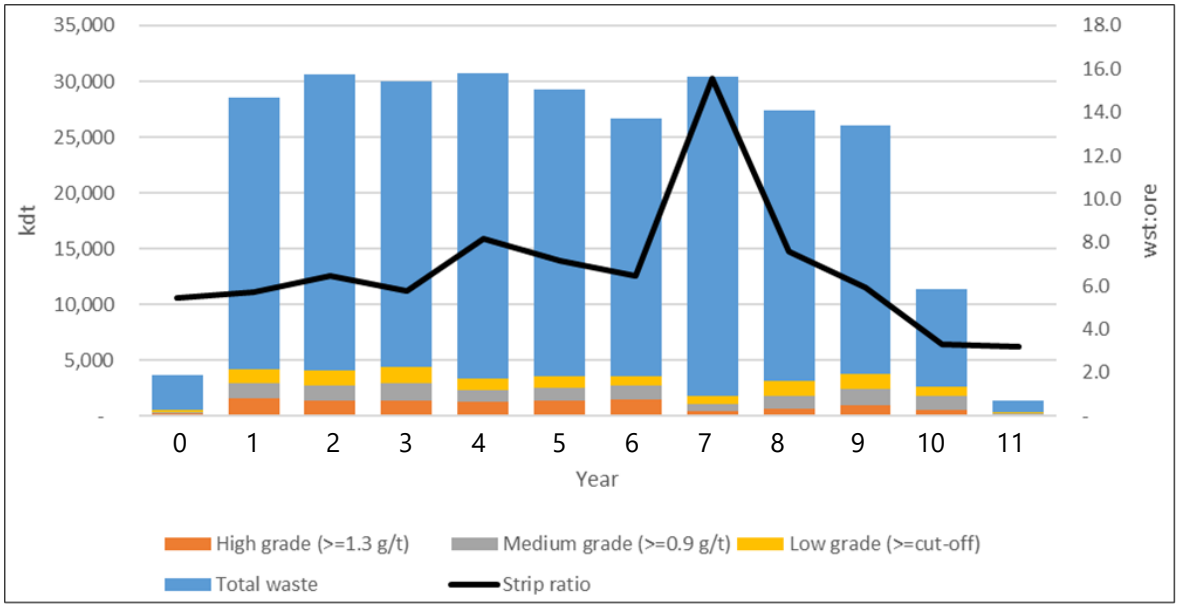
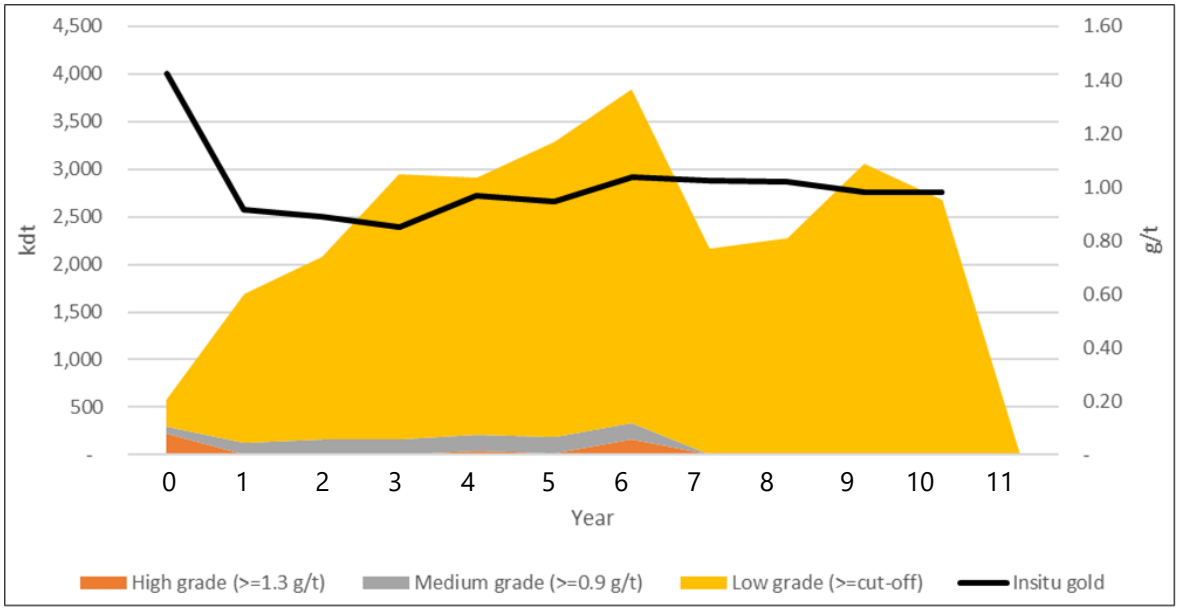


Figure 16.4.4 shows the amount and grade of the ore on the long-term stockpile. Low grade is the main ore type stockpiled as the higher grades are sent to the ROM and fed to the plant. The long-term stockpile is primarily rehandled twice over the project life; during waste-strip of Stage 6 (Year 8) and at the end of mining (Year 12).

Figure 16.4.4 Long-Term Stockpile by Material Type



16.4.3 Processing Schedule

Figure 16.4.5 shows the ore fed to the processing plant by weathering. Ore feed is above the nominal 3 Mt/a (fresh) for most of the processing life due to the presence of more highly weathered ore (which have higher throughputs). Plant usage is at the planned capacity until the final year of operation.

Figure 16.4.5 Ore Feed by Weathering

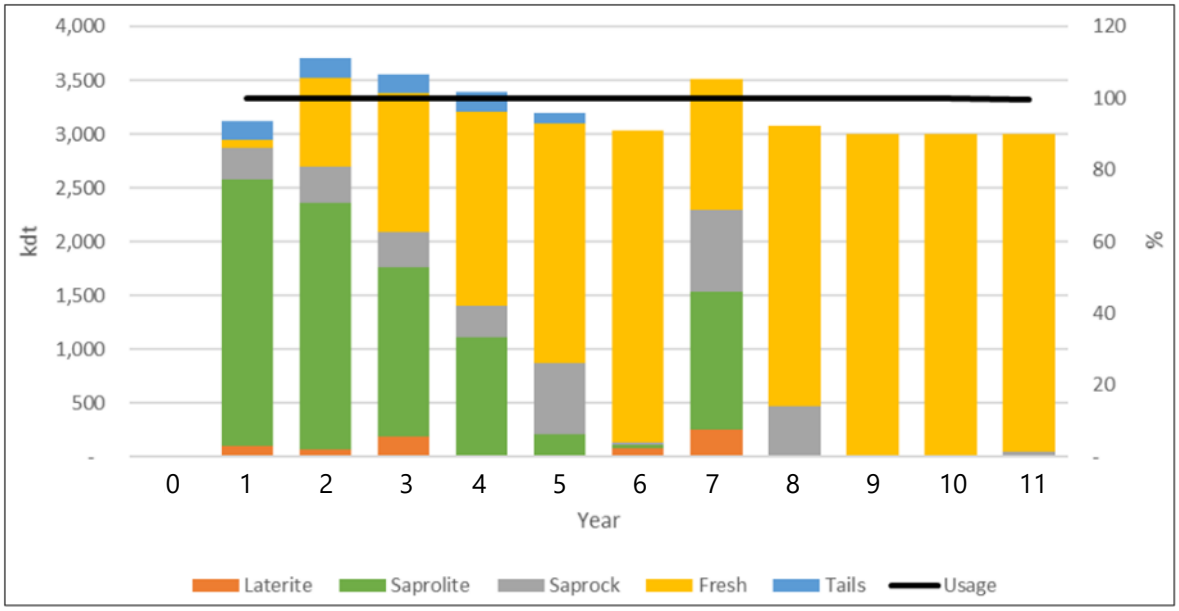


Figure 16.4.6 shows the ore feed to the processing plant by type and the feed grade. After an initial high grade peak, the feed grade is predominately around 1.6 g/t except when feeding from the long-term stockpiles.

Figure 16.4.6 Ore Feed by Material Type

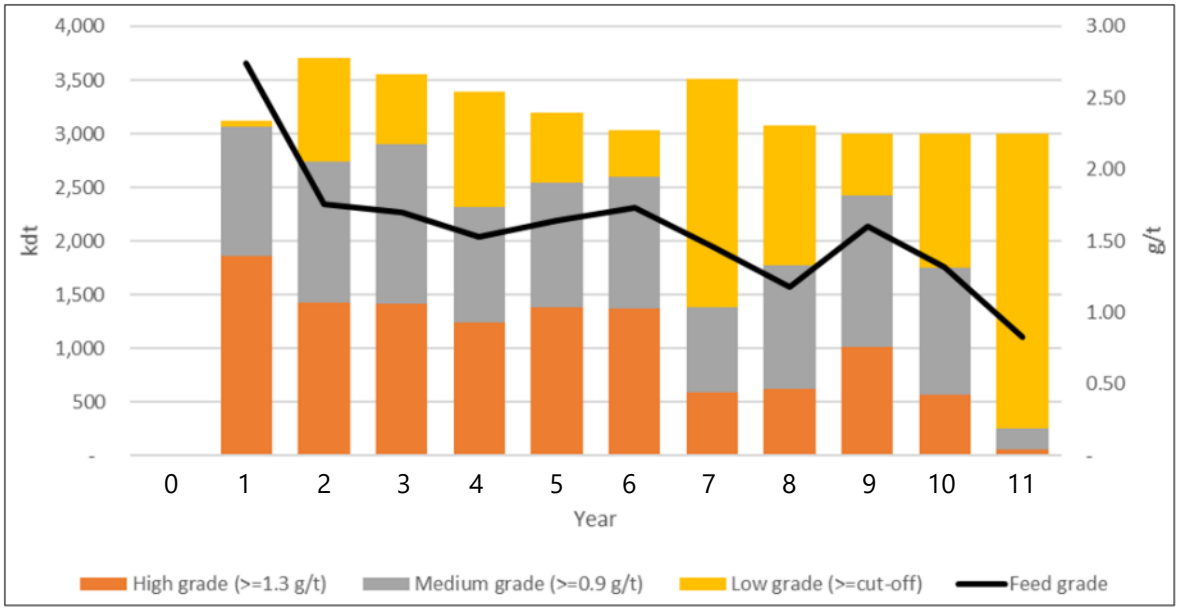
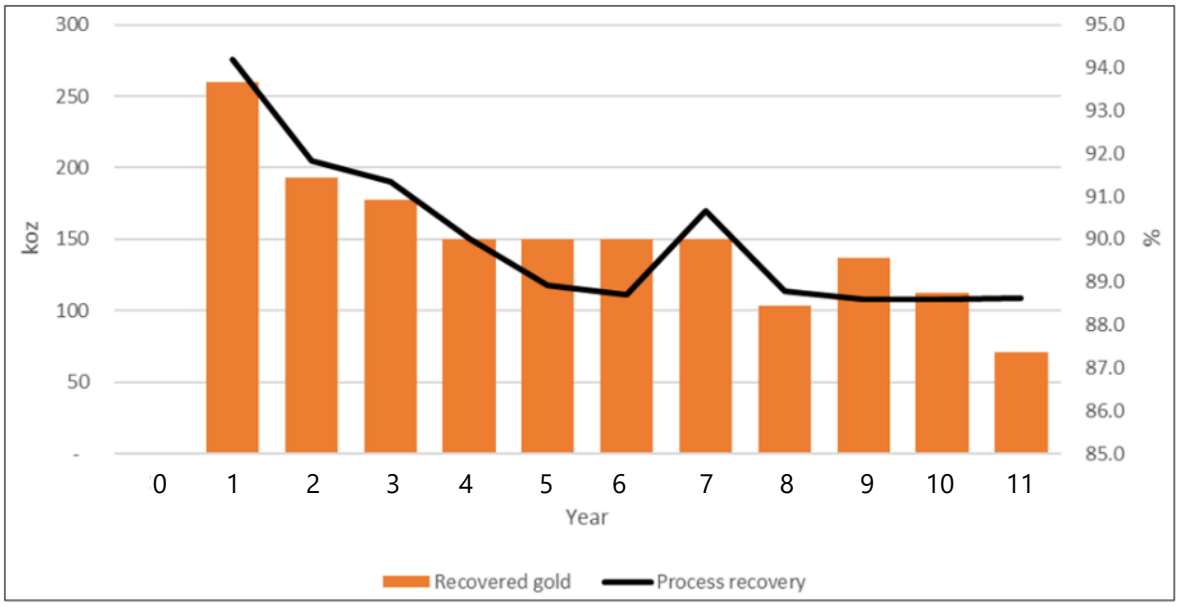


Figure 16.4.7 shows the metallurgical recovery and gold produced. The first three years of production are above 150 koz/a due to higher throughputs, above average grades, and high metallurgical recoveries. This is followed by four years at 150 koz/a, with the remaining years above 100 koz/a (except for the final year).

Figure 16.4.7 Gold Production and Metallurgical Recovery



16.5 Fleet Size and Personnel Numbers

A suitably experienced mining contractor will be engaged to operate the grade control, drill and blast, load and haul and other ancillary mining activities. The mining fleet and personnel number presented in this section are indicative only and will be refined once the mining contractor has been appointed. Table 16.5-1 summarises the indicative primary mining fleet. The equipment quantities vary over the mine life due to variations in the required movements and haul distances.

Table 16.5-1 Primary Mining Fleet

Activity	Type
Drilling	~20 t operating weight top-hammer drill
Loading	200 t operating weight excavator
Loading	100 t operating weight excavator
Hauling	90 t capacity rigid body truck
Ancillary	50 t operating weight track dozer
Ancillary	50 t operating weight wheel dozer
Ancillary	28 t operating weight motor grader
Ancillary	100 t operating weight wheel loader

Table 16.5-2 summarises the mining personnel required to operate the mine. The number of full time employees was estimated based on a 3 shift per day roster.

Table 16.5-2 Mining Personnel

Role	Peak Full Time Employees
Management	3
Production supervision	11
Maintenance supervision	21
Technical services	76
Training and safety	5
Production operators	320
Maintenance	124
Total	**560**

16.6 Technical Risks and Opportunities

Suitable infill drilling would allow sections of the Resource to be classified as Measured. As well as increasing the Resource confidence, this would also allow for a more accurate dilution and ore loss assessment.

The mine schedule is based on high level inputs that are appropriate to develop cost estimates for the life of mine cash flow models which are used to demonstrate economic viability of the Mineral Reserve. Inputs for detailed short-term planning will be required to reduce risk and realise the modelled cash flows.

Maintaining best practise ore control procedures in the open pit is critical to delivering scheduled tonnes and grade to the processing plant. Ore control personnel must be suitably trained and qualified. Suitably qualified and experienced open pit blasting personnel will also be required to ensure the high standards of blasting required are adhered to, not only to ensure continuing good community relations, but for dilution and ore control purposes.

The ground control management plan should be prepared prior to commencing mining operations, with provision and installation of basic slope stability monitoring instrumentation networks such as survey control stations and prisms, and piezometers to monitor groundwater pressures. The plan should include regular visual inspections of pit and dump slopes. A slope stability radar may be required, should major instability risks develop.

The operation requires a large number of personnel that will need to be sourced and if necessary, suitably trained. This will take time and if it is slower than the planned mining ramp-up will result in lower grade and potentially less tonnes fed to the processing plant manifesting as lower gold production and net present value.

At the peak, the number of haul trucks per loading unit is high (~12) which may lead to ramp congestion and reduced productivity. This may necessitate replacing feed with lower grades or larger trucks resulting in the need for wider ramps.

Geotechnical design reviews should be conducted after the mine has been operating for approximately two years, using revised, updated geotechnical and geological models and the results from the pit slope monitoring systems.

The mine will also require a void management plan to control the hazards posed by mining through old workings.

17.0 PROCESS PLANT

17.1 Process Design

17.1.1 Design Philosophy

The key criteria for equipment selection are the suitability for duty and the projected mine life of the operation without unnecessarily compromising reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a compact footprint to minimise construction costs.

The Kalana gold plant will process a range of material types from the Kalana pit (oxide, transition and primary ore) of variable material characteristics and head grades, as well as oxide ore from the Kalanako pit. A minor contribution to the overall LOM processing tonnage will also be made from retreatment of the existing gravity tailings material in the initial years of the plant operation. Primary ore is more competent than transition material, which in turn is significantly more competent than oxide ore. All ore types contain significant quantities of gravity recoverable ('free') gold.

The key project design criteria that the plant design must meet are as follows:

- 3,000,000 t/a of primary ore.
- Crushing plant mechanical availability of 80% (7,008 h/y).
- Mechanical availability for the remainder of the plant of 91.3% (8,000 h/y) supported by crushed ore storage and standby equipment in critical areas.
- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

A process design criteria (2166-FPDC-001) document has been prepared incorporating the engineering and key metallurgical design criteria derived from the results of PFS metallurgical testwork, as discussed in Section 5, and comminution circuit.

17.1.2 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Primary crushing with a jaw crusher to produce a coarse crushed product.
- A crushed ore surge bin and emergency stockpile.
- A SABC milling circuit comprising a SAG mill in closed circuit with a pebble crusher and a ball mill in closed circuit with hydrocyclones to produce an 80% passing (P_{80}) 90 µm grind size.

- Gravity concentration and removal of coarse gold from the milling circuit and treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to doré.
- Pre-leach thickening of the milled slurry to increase the slurry density feeding the carbon in leach (CIL) circuit.
- A CIL circuit to leach and adsorb gold and silver values from the milled mineralised material onto activated carbon in six CIL tanks.
- A split AARL elution circuit, electro-winning and gold smelting to recover gold from the loaded carbon to produce doré.
- Tailings treatment incorporating cyanide destruction using the INCO SO_2 / air process (sodium metabisulphite / oxygen), followed by arsenic precipitation and stabilisation.
- Tailings pumping to the tailings storage facility (TSF).

17.1.3 Plant Design Basis

Process Plant

The plant design has been based on a nominal capacity of 3.0 Mtpa of primary mineralised material. Over the life of mine the plant will be fed with approximately 27% oxide, 9% transition, 62% primary, and 2% tailings material with the majority of the oxide and tailings processed in the first four years of operation. When processing predominantly oxide ore in the early years of the mine life, it is anticipated that the plant throughput will be significantly greater than the nominal 3.0 Mtpa.

ROM Pad

The ROM stockpile will allow blending of feed stocks and ensure a consistent feed rate and grade to the plant, while the ROM bin will be designed to accommodate both direct tipping from mine trucks and blended feed addition by FEL.

A fixed rock breaker located adjacent to the ROM bin will be used to break any oversize rocks on the static grizzly.

Comminution Circuit Selection

A primary crush SABC circuit has been selected for the PFS plant design. The circuit will achieve a target P_{80} grind size of 90 µm treating primary mineralised material with the design based on the 85th percentile results of the PFS comminution testwork for the primary material. Details of the comminution testwork are included in Section 5. The comminution circuit is detailed in the comminution design report compiled by OMC.

The conclusions from the report are as follows:

- OMC recommended a primary jaw crusher / SABC (SAG mill, ball mill and recycle crusher) circuit.
- Use of identical SAG and ball mill sizes to those recently installed at Endeavour's Houndé and Ity gold operations would provide a good match to the proposed Kalana throughput rate.
- Treatment of the less competent oxide material in the initial years of operation will lead to over-grinding with a predicted P_{80} grind size significantly less than the 90 µm target.

The comminution equipment selection is summarised in Table 17.1-1. The comminution average power and consumables requirements for primary and oxide mineralised materials as modelled by OMC have been used in the estimation of the process plant operating cost.

Table 17.1-1 Summary of Selected Comminution Circuit

Description	Units	Parameter	
Comminution Circuit		Primary Crush SABC	
Design Blend		100% Primary	
Primary Crushing			
Type		Jaw Crusher	
Model		Metso C160 or equivalent	
Open Side Setting	mm	130	
Machine Load	%	60	
Milling			
Type		**SAG Mill**	**Ball Mill**
Mill Diameter x Length (EGL)	m x m	8.53 x 4.35	6.10 x 9.05
Discharge Configuration	-	Grate	Overflow
Speed	% Nc	75	75
Drive Speed	type	Variable	Fixed
Top Ball Size	mm	125	65
Milling Density	% Solids	75	75
Ball Charge	% vol	11.4	29.9
Total Load	% vol	25.0	N/A
Pinion Power	kW	4,773	5,117
Installed Power	kW	6,000	6,000
Pebble Crushing			
Type		Cone Crusher	
Model		Metso HP200 or equivalent	
Closed Side Setting	mm	13	
Machine Load	%	83	

Crushing and Crushed Material Storage

The crushing circuit has been sized on the basis of achieving 80% utilisation at a feed rate 14% above that of the nominal milling throughput.

Crushed ore will feed a surge bin and excess crushed ore will be stockpiled. During periods of crusher maintenance stockpiled ore will be reclaimed using a FEL and delivered to the plant via the surge bin.

Milling and Classification

The SAG mill will be equipped with a variable speed drive and will be capable of operating between 60% and 80% of critical speed to accommodate the variability in ore hardness. The ball mill will have a fixed speed drive.

A cluster of cyclones with standby units will facilitate maintenance on individual units without compromising operability. Cyclone overflow slurry will report to duty / standby trash screens ahead of a pre-leach thickener, while cyclone underflow will be returned to the ball mill for further grinding.

Selection of a pre-leach thickener to increase leach feed density allows operation of the milling and classification circuit to be optimised by operating the cyclones at feed densities that increase classification efficiency, thereby reducing circulating load and overall circuit power consumption.

Gravity Concentration

Gravity testwork has indicated that up to 70% of the gold in the plant feed may be recovered by gravity gold methods. A gravity recovery circuit will be fed directly from the mill discharge hopper, with intensive cyanidation of the concentrates. A dedicated electrowinning circuit will be provided for the intensive cyanidation solution to assist with metallurgical accounting and eliminate any potential impact on the operation of the carbon elution and electrowinning circuit.

Pre-leach Thickening

A pre-leach thickener will provide a higher and more consistent leach feed density to reduce the overall tankage volume required for the target residence time, reduce reagent costs and recover process water for immediate re-use in the milling circuit.

Thickener sizing and flocculant dose rates have been based on laboratory testwork. Target thickener underflow densities of 45% w/w and 50% w/w solids have been established when treating predominantly oxide and primary ore, respectively.

Leach and Adsorption Circuit

The historical and DFS metallurgical testwork indicated that:

- 'preg-robbing' is unlikely to be an issue at Kalana
- initial leach kinetics for all lithologies are rapid, with between 95% and 100% of total gold extraction being achieved in the first twelve hours of leaching
- silver was present in most samples with a silver:gold ratio between 0.3 and 0.6 while overall silver extraction averaged about 50%
- cyanide, lime, and oxygen consumptions vary depending on ore type but are generally low.

On this basis, a circuit configuration comprising six stages of CIL tanks has been selected in order to achieve acceptable stage efficiency and target soluble solution losses, whilst maintaining a moderate carbon in circuit inventory. All CIL tanks will be identical in size with the circuit designed for a total residence time of 36 hours when treating exclusively primary ore (375 t/h, 50% w/w solids).

Elution and Goldroom

On the basis of the design feed grade and maximum CIL gold and silver extraction (assuming no gravity gold recovery), a ten tonne capacity split AARL elution circuit has been selected requiring seven strips per week. The goldroom will require two electrowinning cells to recover the expected precious metals load.

The presence of mercury within the Kalana ores has not been identified, and thus no specific mercury mitigation measures have been included within the proposed elution circuit and goldroom designs.

Plant Tailings Treatment

The air / SO_2 cyanide destruction method will be used to reduce cyanide in the plant tailings to below a CN_{WAD} level of 50 ppm. Since oxygen will be generated on site for use in CIL, oxygen will also be used in the destruction circuit rather than air, to improve reaction efficiency.

Some of the lithologies also contain significant arsenic concentrations and consequently a ferric sulphate arsenic precipitation and stabilisation method (scorodite process) will be used to remove soluble arsenic from the cyanide destruction discharge slurry as a benign ferric salt precipitate suitable for disposal in the proposed lined tailings storage facility (TSF). Design parameters and reagent consumption rates for the cyanide destruction and arsenic precipitation circuits have been determined from a programme of metallurgical testwork.

Water Supplies

Raw water for the Project will be sourced from the mine pit and will be used as raw water in the plant, mine and camp and as the feed stream for the potable water treatment plants.

The pre-leach thickener overflowing to the mill water tank will allow direct recycle of the majority of process water required for the milling circuit with make-up water from the raw water supply as required. The TSF decant return water will be pumped to the mill water tank and excess water will overflow to the adjacent process water pond.

17.1.4 Key Process Design Criteria

The key process design criteria summarised in Table 17.1-2 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1-2 Summary of Key Process Design Criteria

Description	Units	Oxide	Primary	Source
Plant Throughput	t/y	3,800,000	3,000,000	EDV
Design Gold Head Grade	g Au/t	2.5	1.6	Assumed
Design Silver Head Grade	g Ag/t	1.0	1.0	Assumed
Gravity Gold Recovery	%	50	50	Testwork
Gravity Silver Recovery	%	30	30	Testwork
Design Overall Gold Recovery	%	90	90	Testwork
Design Overall Silver Recovery	%	50	50	Testwork
Crushing Plant Utilisation	%	80	80	Lyco / OMC
Milling / CIL Plant Utilisation	%	91.3	91.3	Lyco / OMC
ROM Ore Top Size	mm	500	800	OMC
Ore SG	-	2.55	2.78	Testwork / OMC
Comminution Circuit	type	Primary Crush / SABC		EDV / OMC
Crush Size, P_{80}	mm	29	143	OMC
Target Grind Size, P_{80}	µm	45	90	Testwork
Cyclone O/F Density	%w/w solids	35	35	Lyco / OMC
Pre-leach Thickener Solids Loading	$t/m^2.h$	1.0	1.0	Testwork
Leach Feed Slurry Density	%w/w solids	45	50	Lyco
CIL Residence Time	hours	24	36	Testwork
Number of CIL Tanks	#	6	6	Lyco
Average Cyanide Consumption[6]	kg/t	0.28	0.30	Testwork
Average Quicklime Consumption[7]	kg/t	0.85	0.67	Testwork
Elution Circuit Type	-	Split AARL	Split AARL	Lyco
Elution Circuit Capacity	t	10	10	Lyco
Frequency of Elution	strips / week	7	7	Lyco
Cyanide Destruction Method	-	INCO Air/SO_2	INCO Air/SO_2	Lyco
Arsenic Precipitation Method	-	Scorodite Process	Scorodite Process	Lyco

Notes:

1. 'Testwork' refers to comminution and metallurgical testwork conducted.
2. 'EDV' refers to advice / agreement from Endeavour Mining
3. 'Snowden' refers to advice / agreement from Snowden.
4. 'Lyco' refers to Lycopodium experience or generally accepted practice.
5. 'OMC' refers to advice from Orway Mineral Consultants.
6. Cyanide consumption makes allowance for 150 ppm residual cyanide in the CIL tail solution.
7. Lime consumption based on 90% CaO.

17.1.5 Tailings Disposal

Plant tailings and other miscellaneous waste streams from the process plant will be combined in the tails hopper and pumped to the TSF for deposition as described in Section 7. The supernatant water (decant return) will be pumped to the plant mill water and process water circuits for re-use.

Any plant spillage that may contain cyanide will report to plant bunding which will overflow to the event drainage channel and will gravitate to the event pond for reclaim.

17.1.6 Reagents

Reagents Storage

Reagents will be received on site either in bulk (grinding media) or in shipping containers, with a minimum of 60 days capacity stored on site.

Quicklime

Quicklime will be delivered to site in bulk tankers and pneumatically transferred to the lime silo for metering directly onto the mill feed conveyor for circuit pH control. The storage silo will be equipped with a reverse pulse bag filter to minimise dust emissions. An additional 'emergency' facility will be provided to enable bulk bags of quicklime to be loaded into a hopper and from there metered via a variable speed feeder onto the mill feed conveyor.

Cyanide

Sodium cyanide will be delivered as dry briquettes in bulk bags. The cyanide will be added to a mixing tank via a bag breaker and dissolved in process water to achieve the required solution concentration (20% w/v NaCN). The cyanide solution will be transferred to the storage tank from which it will be reticulated to the CIL circuit via a ring main and dosed to the CIL tanks as required. A dedicated pump will provide cyanide solution for the elution circuit and intensive cyanidation reactor as required.

Caustic

Caustic soda (sodium hydroxide) will be delivered to site as dry 'pearl' pellets in bulk bags. The caustic will be added to a mixing tank via a bag breaker on the receiving hopper. The pellets will be discharged into the mixing tank via a screw feeder and dissolved in raw water to the required solution concentration (20% w/v NaOH). The caustic solution will be pumped to the elution circuit as required. A separate ringmain will deliver caustic to the intensive leach reactor and cyanide destruction and arsenic precipitation facilities.

Hydrochloric Acid

Concentrated hydrochloric acid will be delivered to site in intermediate bulk containers (IBCs) and pumped into the correct quantity of filtered water in the acid mixing / storage tank to achieve the target concentration (3% w/v HCl) for acid washing. The dilute acid will be pumped to the elution circuit as required.

Activated Carbon

Activated carbon will be delivered in bulk bags. Carbon will be added via the carbon quench tank as required for carbon make-up to the CIL inventory. This addition point will allow any fine carbon particles to be removed on the carbon sizing screen prior to entering the CIL circuit.

Grinding Media

Grinding balls will be delivered to site in drums and will be charged to the SAG and ball mill, as required, to achieve the target power draws.

Flocculant

Flocculant powder will be delivered to site in bulk bags and subsequently added to the flocculant plant storage hopper. The vendor supplied package flocculant mixing plant will automatically mix batches of flocculant with raw water and transfer the mixed flocculant to a separate storage tank after each mixing cycle is complete.

Flocculant solution will be metered to the pre-leach thickener and to the intensive leach reactor.

Sodium Metabisulphite

Sodium metabisulphite (SMBS) will be delivered to site in bulk bags as a dry powder. SMBS will be added to the agitated mixing tank using a hoist and bag breaker. The powder will be dissolved in raw water to the target solution concentration (20% w/v Na2S2O5) and then transferred to the storage tank and metered to the cyanide destruction circuit. Both the mixing and storage tanks will be vented to atmosphere via an extraction fan to prevent accumulation of SO2 in the SMBS make-up and storage area.

Copper Sulphate

Copper sulphate will be delivered to site in bulk bags as a dry powder and will be added to the mixing tank via a bag breaker on the receiving hopper. The powder will be dissolved in raw water to the target solution concentration (20% w/v $CuSO_4$). The buffer tank will be used as the dosing tank during copper sulphate mixing, with the solution being dosed into the cyanide destruction circuit.

Ferric Sulphate

Ferric sulphate will be delivered to site in bulk bags as a dry powder and will be added to the mixing tank via a bag breaker on the receiving hopper. The powder will be dissolved in raw water to the target solution concentration (20% w/v $Fe_2(SO_4)_3$), transferred to a storage tank and metered into the arsenic precipitation circuit.

Fluxes

Sodium borate (borax), silica flour, sodium nitrate (nitre), and sodium carbonate (soda ash) will be used as fluxes for gold smelting. The fluxes will be delivered in 25 kg bags and mixed in small quantities with the dried gold and silver sludge prior to smelting.

Diesel

Diesel will be pumped from the site storage tank to the plant day tank for use in the elution heater and carbon regeneration kiln and for refuelling plant mobile equipment.

17.1.7 Services

Raw Water

Raw water from the pit will be distributed to the plant, mine, and camp areas, and to the gravity concentrators.

Fire Water

Fire water for the process plant will be drawn from the reserve in the lower section of the raw water tank and reticulated using a standard pump skid with an electric jockey pump, electric and diesel delivery pumps.

Fire hydrants and hose reels will be located throughout the process plant, fuel storage and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

Mill Water

Overflow from the pre-leach thickener will gravitate to the mill water tank. Decant water from the TSF will be pumped to the mill water tank, with supplementary addition of raw water provided from the raw water tank overflow. Excess mill water will overflow to the process water pond.

Process Water

Overflow from the raw water and mill water tanks will gravitate to the process water pond, which will also receive reclaimed water from the event pond and backwash from the water treatment plant.

Antiscalant will be added as required to condition the water and reduce fouling of pipelines, spray nozzles and screen decks.

Filtered Water

Filtered water for the elution circuit and some reagent mixing uses will be produced by treating raw water in the filtered water treatment plant.

Gland Water

Water from the filtered water storage tank will be distributed as gland service water using duty / standby gland water pumps.

Potable Water

Potable water will be generated by treatment of raw water in the camp potable water treatment plant. using micro filtration, ultra-violet sterilisation and chlorination. Potable water will be pumped from the camp potable water storage tank to separate potable water tanks located at the process plant and mine services areas. Potable water from the plant potable water tank and will be reticulated to the site ablutions, safety showers and other potable water outlets. Additional ultra-violet sterilisation units will be installed on outgoing potable water distribution headers.

Cooling Water

Cooling water for the SAG and ball mills will be generated using an evaporative water cooling tower and heat exchanger. Filtered water will be added to the cooling tower sump to supplement the water loss from evaporation and blowdown. Antiscalant and biocide will be dosed into the cooling water system to maintain water quality.

Plant and Instrument Air Supply

Plant and instrument air will be supplied from a bank of three rotary screw air compressors. The air will be filtered and dried before distribution to separate area specific air receivers which will supply air to the plant at a nominal 700 kPag.

Oxygen

Dual (duty / standby) pressure swing adsorption (PSA) oxygen plants will be installed which will generate an oxygen rich (~90% O_2) gas stream. Gaseous oxygen will be reticulated under pressure from the oxygen receiver to the intensive leach reactor, CIL, cyanide destruction, and arsenic precipitation circuits.

17.2 Plant Layout and Design Considerations

The plant has been located to suit the available usable area within a restrictive topography, while providing appropriate access for all supporting infrastructure.

Where practical, general project facilities will be located around the process plant site to maintain a single centre of operations. Other facilities provided are divided into the following categories:

- Mine services area (refer Section 8).
- Administrative and general support services (refer Section 8).
- TSF (refer Section 7).
- The accommodation camp is located more remotely from the process plant (refer Section 8).

17.3 Electrical Design

The Project, including all mining and processing facilities, accommodation and infrastructure is estimated to have a maximum demand of 17.5 MW, and an average load of 16.4 MW. The largest single loads will be the SAG mill and ball mill, each with an installed motor power of 6.0 MW. The SAG mill will be provided with a VVVF drive which will minimise the impact on the power system during starting, while the ball mill will have a liquid resistance starter. Refer to Table 17.3-1 for a breakdown of power demand by load centre.

Table 17.3-1 Power Demand by Load Centre

Equipment Number			Description	Connected Load	Maximum Demand		Av Power
Area No	Equip Type	Seq No		kW	kW	kVA	MWh/year
340	SB	001	Plant Main 11 kV Switchboard	13,992	10,571	11,756	89,827
120	MC	001	Crushing MCC	790	520	620	3,870
130	MC	002	Milling MCC	2,223	1,074	1,238	8,427
160	MC	003	CIL MCC	2,850	1,983	2,340	15,326
210	MC	004	Reagents MCC	819	540	668	3,643
220	MC	005	Services MCC	3,582	1,430	1,744	12,005
350	MC	006	TSF MCC	341	272	325	2,315
230	DB	100	Sewage Treatment Plant Distribution Board	19	9	11	82
380	DB	101	Camp Distribution Board	1,000	500	588	3,066
450	DB	102	MSA Distribution Board	518	240	293	1,892
			Plant Buildings Distribution Board	490	392	478	3,768
Total				**26,623**	**17,532**	**20,062**	**144,222**

17.4 Control System

17.4.1 General

The main control room will house two PC based operator interface terminals (OIT), with two additional servers to act as the control system supervisory control and data acquisition (SCADA) servers in a redundant configuration. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

The PCS will be a programmable logic controller (PLC) and SCADA based system to control the process interlocks and PID control loops for non-packaged equipment.

Vendor supplied packages will use vendor standard control systems throughout the project with limited interfaces with the PCS. General equipment fault alarms from each vendor package will be monitored by the PCS and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

17.4.2 Field Input / Output

All instrumentation and field controls will be captured by the PCS via remote I/O modules, generally located at each process module of the plant, primarily to minimise extended cable run lengths back to switchroom. All remote I/O nodes will be linked to the PLC via a PCS Ethernet network, supported over the site fibre optic cabling between switchrooms, remote I/O modules and key infrastructure buildings.

17.4.3 Drive Control

In general, the plant process drives will report their ready, run and start pushbutton status to the PCS and will be displayed on the OIT. Local control stations will be in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) pushbuttons which will be hard-wired to the drive starter. Plant drives will predominantly be started by the control system in automatic operation.

17.4.4 Control Loops

There will be two modes for loop controlled setpoints available in the OIT. These are 'Loop Auto Mode' and 'Loop Manual Mode'. In Loop Auto Mode (analogous to cascade control), the setpoint will be predominantly controlled by the applicable 'master' PID loop while in Loop Manual Mode, a setpoint may be entered manually from the loop setpoint pop up in the OIT.

17.5 Communications

17.5.1 Network Topology

The onsite communications network is designed around a site-wide fibre optic backbone which will be shared by all services. This will minimise cabling and related communications equipment and be installed in underground conduits, cable tray and via the overhead powerline. The services that will use the common fibre optic backbone include:

- Corporate LAN including telephony (Voice over Internet Protocol – VoIP).
- Plant Control System.
- CCTV / Security Services.

17.5.2 External Connectivity

A microwave internet link will provide external network connectivity to the site. Camp entertainment services will be provided via satellite TV with dishes located on each senior accommodation building.

17.5.3 Private Mobile Radio (PMR)

Cost allowances have been made to provide a site-wide PRM system typically supported above ground by a 30 m high communications tower. Heavy and light vehicles will be equipped with radios.

17.5.4 Server / Computer Infrastructure

Corporate servers, network switches and a firewall will be installed onsite to support the users locally with the expectation the proponent will implement a VPN link to any remote central office as required. An allowance for staff workstations / laptops has been made along with required software and office equipment such as docking stations, monitors, photo-copiers and cabling.

17.5.5 CCTV / Security

CCTV cost allowances have been made to incorporate general plant coverage, 'no-man's land' perimeter monitoring, access gates coverage and detailed coverage of the concentrator and gold room. Security cost allowances include automated access boom gates to the site and swipe card access control to the gold room, switch rooms, control room and server room.

17.6 Metallurgical Accounting

Weightometers located on conveyors will totalise primary crushed ore, mill feed, and pebble crusher circulation, with stockpile feed tonnes calculated by difference.

Automated stream samplers on the leach feed and adsorption tails streams will provide reliable composite shift samples for leach head grade and tails solution and residue grades.

Density and flow meters on the leach feed and plant tailings lines will allow the dry tonnage of solids pumped to the leach circuit and TSF to be determined. In conjunction with the leach feed and plant tails samples, the mass flow measurements will allow the gold recovered in CIL to be calculated.

A dedicated electrowinning cell will be provided for recovery of the gold leached from the gravity concentrate and the recovered gravity gold sludge can be smelted separately. The plant head grade can be back-calculated from the gravity and leach head grade.

Regular gold in circuit (GIC) surveys will allow reconciliation of precious metals in feed compared to doré production.

Reconciliation of the amount of reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates to unit operations will be measured (L/h) and accumulated (m^3) using flow meters.

18.0 PROJECT INFRASTRUCTURE PLANT

18.1 Site Development

The existing Kalana gold plant was designed and constructed using Soviet engineers, technology and equipment between 1982 and 1985. The plant was originally designed for a throughput of 3,600 tpm with excellent plant offices and infrastructure. The plant will be demolished and the site made safe prior to the construction starting for the new plant, therefore the new plant will be a greenfield constructed plant.

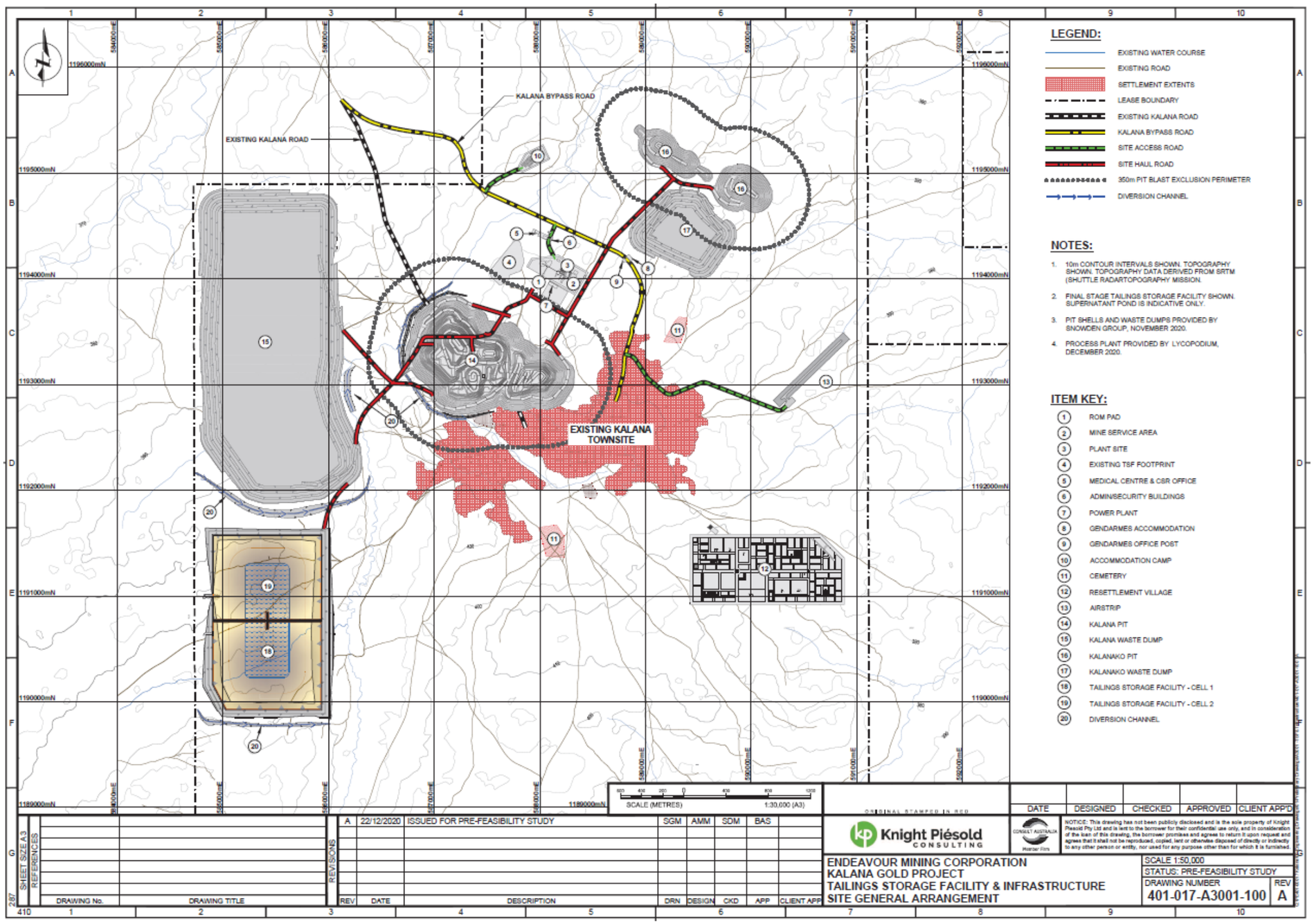
The overall site development plan is shown in Figure 18.1.1.The figure shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, mine services area and mine open pit.

The process plant and tailings storage facility will be located on the north eastern side of the open pit, outside of the 350 m blast zone. The ROM pad location has been selected for its centralised location close to the pits as well as proximity to the mine services area (MSA). The accommodation camp will be located north of the process plant.

The 4.9 km main access road will approach the site from the south east and connect to RN8, part of the regional road network between Yanfolila and Bougouni. The layout provides easy access for personnel and material movements.

The process plant will be fenced to prevent animal access and deter access by unauthorised persons. Monitored high security fencing will surround the process plant. Road access into the fenced area will be through a manned checkpoint. Security fencing will surround the accommodation camp and general site infrastructure.

Figure 18.1.1 Kalana Project General Arrangement



18.2 Site Access Roads

Knight Piésold have provided design parameters for the site access road and major on-site roads. The design basis for the PFS costs is presented in Table 1.3-1. The proposed site access road alignments from Kalana are shown on Figure 18.2.1 with typical road sections and details in Figure 18.2.1. The total length of the site access roads is 13.8 km.

The road vertical alignments were designed to balance cut to fill as much as practicable over the entire road length. Local balancing of cut and fill volumes will be completed during the detailed design phase. The vertical alignment design included allowance for fill build-up around stream crossings to ensure correct operation of crossings.

The site access road and major on-site roads will consist of two 3.5 m width running lanes with a 1 m shoulder each side of the road, for a total formation width of 9 m. The road crossfall will vary along the road alignment to suit drainage requirements. A 200 mm laterite wearing course will be placed over the subgrade / general fill. Drainage ditches, turnouts and level spreaders will be incorporated into the site access road designs during the detailed design phase. A drainage ditch will run either side of the road formation as required, to convey runoff to level spreaders and culvert structures. The drainage ditch will be grader cut, with the excavated material pushed into the road formation to be used as general fill or sub-base.

Culvert crossings were designed at significant stream crossing locations along the site access road alignments to convey all runoff resulting from a 20-year average recurrence interval storm event (duration equal to time of concentration). The culverts will comprise either corrugated metal pipe or precast concrete box culverts, depending on the peak flow rates.

Figure 18.2.1 Site Access Roads Sections and Details

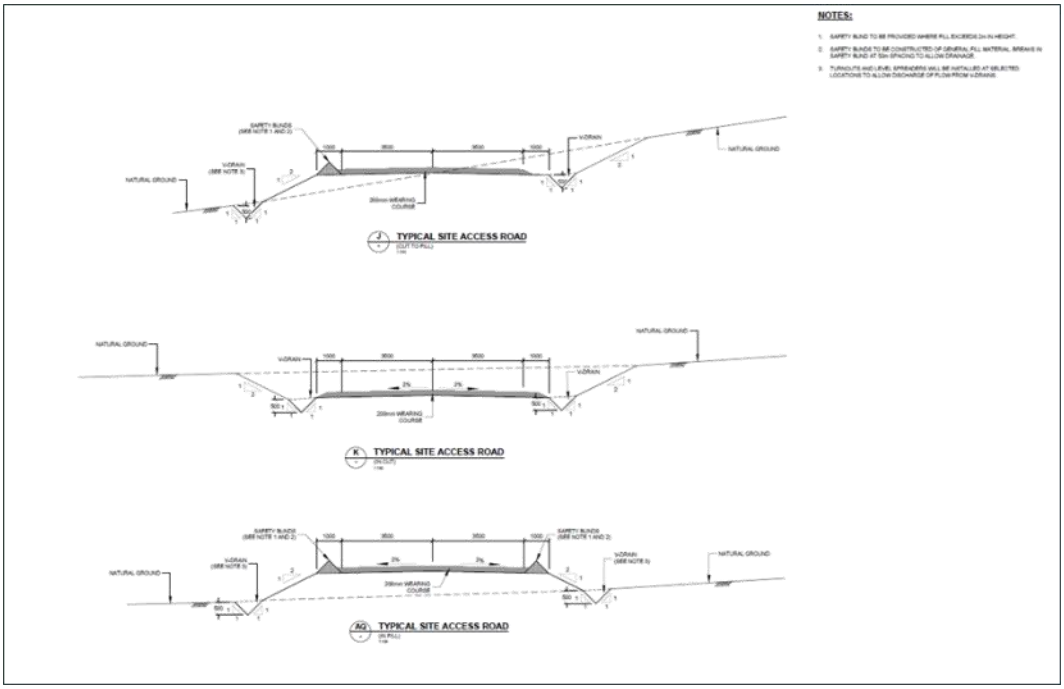


Table 18.2-1 Site Access Road Design Parameters

Design	
Road Cross Section	Formation Width – 9 m
	Lane Width – 3.5 m with 1 m shoulder each side of road
	Safety Bunds – 0.5 m height where fill height > 2 m
	Crossfall – 2%
Minimum Horizontal Curve Radius	30 m
Minimum Vertical Curve Length	25 m
Maximum Vertical Grade	8%
Minimum Culvert Diameter	600 mm
Culvert Design Criteria	20 year ARI (Average Recurrence Interval)
Pavement	200 mm laterite gravel wearing course

18.3 Plant Roads

Allowance has been made for minor roads and tracks around the process plant to facilitate operations and maintenance access. These include:

- The tails pipeline is 4,000 m long from the process plant to the farthest discharge point. A track has been included from the plant to the north west corner of the embankment. From here access will be over the embankment wall.
- Access roads (unsealed) have been allowed to the main accommodation camp (450 m), and the security camp (800 m).

18.4 Mine Haul Roads

Mine haul roads connect the open pits, waste dumps and mine services area. The total length of the haul roads is 1.3 km. The haul roads will be constructed progressively in line with the staged pit development as described in Section 4.

Knight Piésold have provided design parameters for the haul roads in Table 18.4-1. The typical road sections and details on Figure 18.4.1.

The road vertical alignments have been designed to balance cut to fill as much as practicable over the entire road length. Local balancing of cut and fill volumes will be completed during the next design phase. The vertical alignment design included allowance for fill build-up around stream crossings to ensure correct drainage at crossings.

The haul roads will consist of two 12 m width running lanes with a 1.5 m high safety bund on each side of the road, for a total formation width of 30 m. The road crossfall will vary along the road alignment to suit drainage requirements. A 200 mm laterite wearing course will be placed over the subgrade / general fill, with a view to being upgraded during operation by the mining fleet when competent rock is available). Drainage ditches, turnouts and level spreaders will be incorporated into the site haul road designs during the detailed design phase. A drainage ditch will run either side of the road formation as required, to convey runoff to level spreaders and culvert structures. The drainage ditch will be grader cut, with the excavated material pushed into the road formation to be used as general fill or sub-base.

Culvert crossings have been designed at significant stream crossing locations along the haul road alignments to convey all runoff resulting from a 20-year average recurrence interval storm event (duration equal to time of concentration). The culverts will comprise either corrugated metal pipe or precast concrete box culverts, depending on the peak flow rates.

Figure 18.4.1 Mine Haul Roads Sections and Details

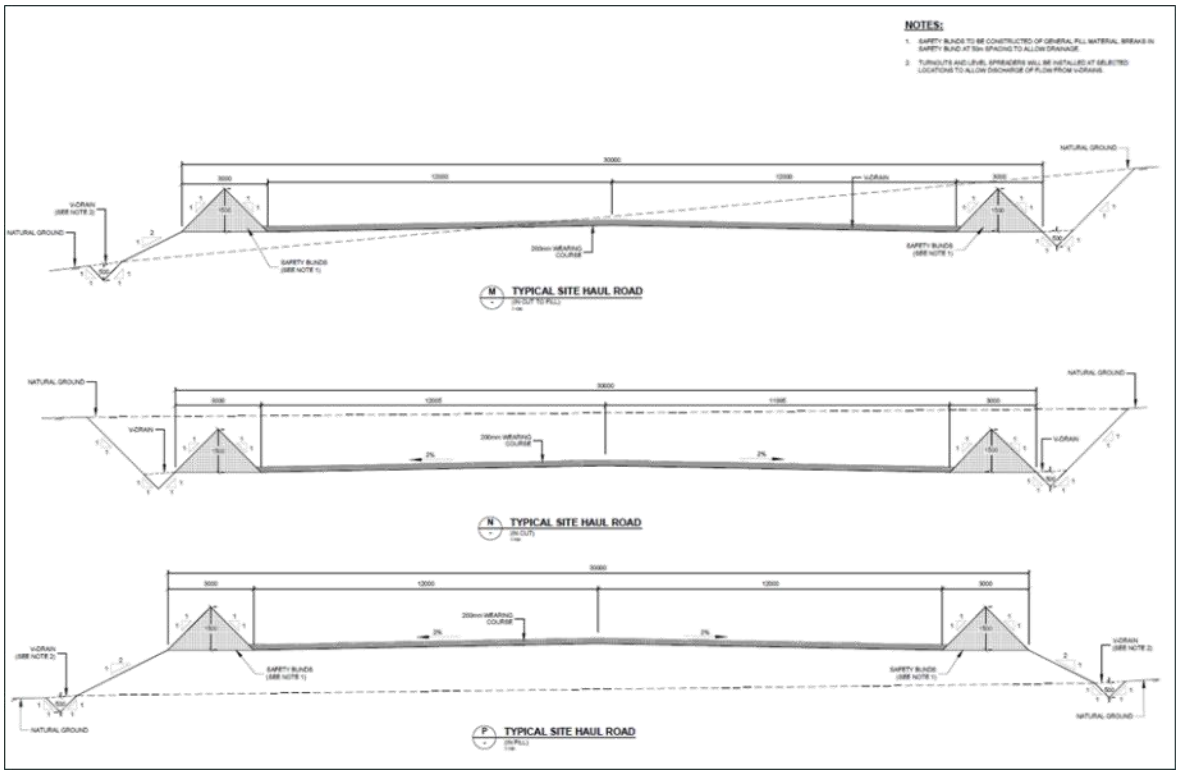


Table 18.4-1 Haul Road Design Parameters

Design	
Road Cross Section	Running Width – 24 m
	Lane Width – 12 m
	Safety Bunds – 1.5 m height both sides of road
	Crossfall – 2%
Minimum Horizontal Curve Radius	100 m
Minimum Vertical Curve Length	25 m
Maximum Vertical Grade	8%
Minimum Culvert Diameter	600 mm
Culvert Design Criteria	20-year ARI
Pavement	200 mm laterite gravel wearing course (to be upgraded by the mining fleet when competent rock available)

18.5 Airstrip

A preliminary non-instrument design has been allowed for the Project airstrip, based on the nominated airstrip location 4 km north of the process plant. No site investigation has been completed at this stage. A fully automated weather station is included together with refuelling facilities.

18.5.1 Airstrip Design Specification

Knight Piésold completed a PFS level design for the site airstrip, in order to provide information for conceptual costing for the PFS study capital cost estimate. The non-instrument design for the airstrip is summarised below:

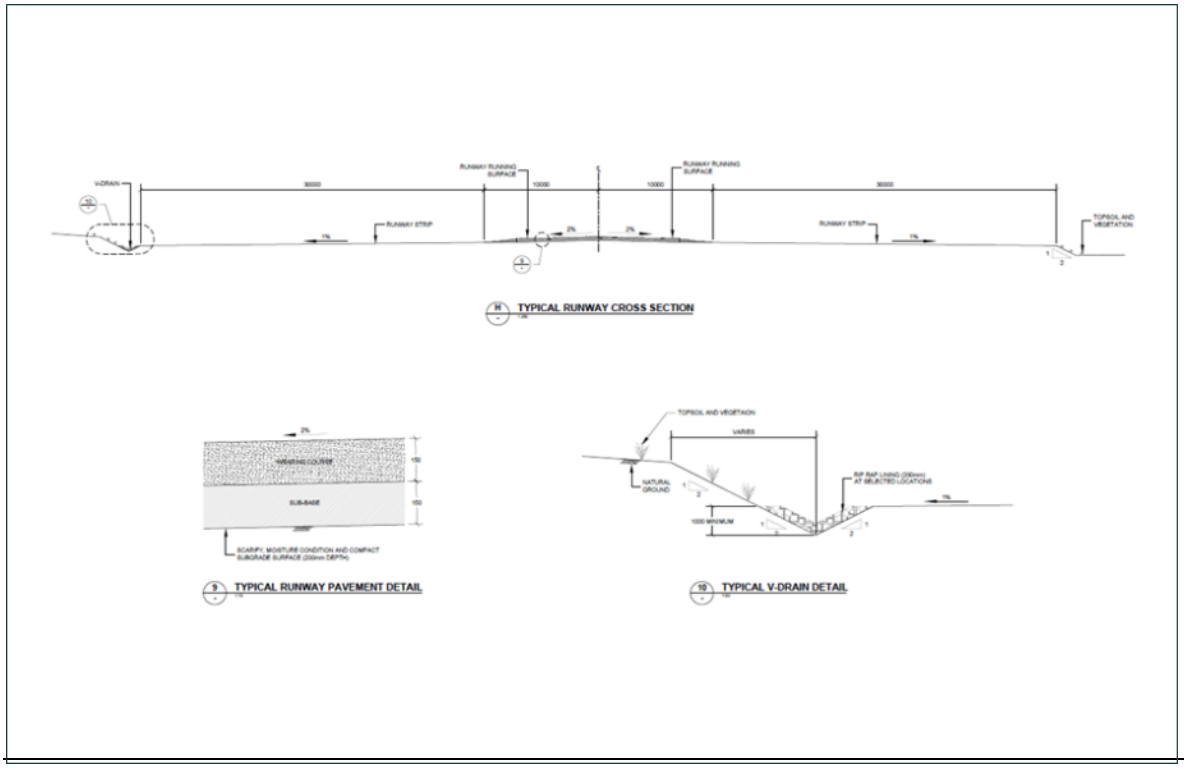
- The airstrip location is shown on Figure 18.2.1.
- The airstrip design parameters are provided in Table 18.5-1.
- The airstrip design was completed using a Pilatus PC-12 as the design aircraft, based on previous Endeavour projects.
- The prevailing wind direction is north-north-east for the Bouake station. The proposed airstrip alignment runs parallel to the prevailing wind direction.
- The runway running surface is 800 m long and 18 m wide.
- The surrounding runway strip is 80 m wide.

- The longitudinal profile of the current natural topography exceeds the minimum longitudinal slope of 2% in multiple locations along the runway alignment. As a result, cut and fill operations will be required to achieve a design compliant with regulatory guidelines.
- Typical cross sections for the site airstrip are shown on Figure 18.5.1.
- The current runway width of 20 m does not accommodate the turning radius of the Pilatus PC-12 aircraft at its design speed. It is recommended a turnaround area 24 m wide (critical turning radius for the Pilatus PC-12) is included in the final 50 m of the runway at each end to facilitate the aircraft.
- A windsock should be constructed in close proximity to the airstrip.
- The apron is 100 m by 50 m and will comfortably park two design aircrafts.
- The aircraft trafficked pavement design (runway, taxiway and apron) comprises two pavement layers constructed over a prepared subgrade. In order of foundation to surface, these can be summarised as follows:
  - Subgrade – scarified and moisture conditioned in-situ material to 200 mm depth, compacted to achieve a California Bearing Ratio (CBR) of at least 15%.
  - Sub-base course – 150 mm well-graded sandy gravel material compacted to achieve a CBR of at least 30%.
  - Wearing course – 150 mm well-graded sandy gravel material compacted to achieve a CBR of at least 60%.
- It is recommended that geotechnical site investigation be conducted along the airstrip alignment and in potential borrow areas to confirm that suitable material is available on site for the airstrip pavement construction.
- External to the trafficked areas (i.e. the runway strip, RESAs), the in-situ material will be graded to line and level, and proof rolled.
- Surface water management around the airstrip should be considered in detail during the next design phase.

Table 18.5-1 Airstrip Design Parameters

Design Criteria	
Design Aircraft	Pilatus PC-12
Aircraft Classification	1B
Runway Length	≥ 800 m
Runway Width	≥ 18 m
Runway Longitudinal Slope	≤ 2.0%
Longitudinal Slope Change Between Segments	≤ 2.0%
Longitudinal Slope Change	≤ 0.4% per 30 m
Longitudinal Slope Radius of Curvature	≥ 7,500 m
Runway Traverse Slope	1.5 - 2.5%
Runway Strip Width*	≥ 80 m
Runway Strip Traverse Slope	≤ 3.0%
Runway Shoulders	Not required
Taxiway Width	≥ 10.5 m
Taxiway Strip Width	≥ 43 m
Taxiway Longitudinal Slope	≤ 3.0%
Wheel Distance to Runway Edge (taxiing)	≥ 2.25 m
Turning Radius (≤ 20 km/h)	≥ 24.0 m

Figure 18.5.1 Airstrip Sections and Details



18.6 Power Supply and Distribution

18.6.1 Installed Load and Maximum Demand

The installed load and maximum demand for the site is shown in Table 18.6-1. The maximum demand is calculated for a ½ hour window and represents the minimum supply capacity required for the site.

Table 18.6-1 Plant Power Demand

Area	Plant Installed Load	Plant Maximum Demand	Plant Average Continuous Load
Process Plant	24,615 kW	16,400 kW	14,924 kW
Infrastructure	4,000 kW	3,200 kW	2,560 kW
Totals	**28,615 kW**	**19,200 kW**	**17,484 kW**

18.6.2 Power Supply

The existing mine is supplied by a 66 kV overhead transmission line from Yanfolila substation located approximately 51 km away from the site. A new 66 kV bay and transformer will be installed at the Yanfolila substation and a new 66 kV transmission line run to a new 66 / 11 kV substation installed adjacent to the process plant. The 66 kV supply voltage will be stepped down to 11 kV at this substation via a 25 / 33 MVA step down transformer.

18.6.3 Electrical Distribution

The electrical system for the Project is based on 11 kV distribution and 415 V working voltage. System frequency is designed at 50 Hz. The largest drives within the process plant are the ball and SAG mill motors (both 6,000 kW).

From the 66 / 11 kV switchyard, the 11 kV supply will be reticulated from a tariff metered plant feeder to the process plant 11 kV switchboard. Remote loads such as the camp loads will be fed via 11 kV overhead transmission lines.

The main process plant will have four containerised switchrooms.

Emergency power generators will be supplied at the main plant (2,000 kVA).

18.6.4 Electrical Buildings

All site electrical switchrooms are designed to house the LV Motor Control Centres (MCC) MCC, MV switchboards, Variable Speed Drives (VSD) and Process Control System (PCS) hardware. The switchrooms will be sealed against dust ingress and be supplied with air conditioning, Uninterruptible Power Supplies (UPS) and fire detection system.

The switchrooms will be mounted on 2 m high steel pedestals to facilitate cable installation below the switchroom and bottom entry connection to the internal equipment through gland plates. Entry to the rooms will be via stairs and access platforms constructed at each end.

18.6.5 Transformers and Compounds

All 11 kV / 415 V distribution transformers will be of ONAN (non-fan forced) cooling configuration and vector group Dyn11.

Fire rated concrete walls will be constructed around the pad mounted transformers.

Distribution transformers have been rationalised in the electrical design to minimise spares holding requirements. The standard sizes are:

- 11 / 0.415 kV 500 kVA – 3 off total
- 11 / 0.415 kV 1,000 kVA – 2 off total
- 11 / 0.415 kV 2,500 kVA – 3 off total

18.6.6 11kV Switchboards

One 11 kV switchboard has been allowed for in the 66 / 11 kV switchyard and one will be supplied for the plant.

The indoor 11 kV switchboards will be a withdraw-able design. All 11 kV switchboards will be supplied with protection, metering and earthing facilities.

The design fault level and circuit breaker ratings adopted are:

- 11 kV switchboard busbar 2,000 A, 25 kA at 1 seconds
- 11 kV incomer circuit breakers 2,000 A
- 11 kV feeder circuit breakers 630 A

Protection will be provided by microprocessor-based protection relays. Protection relays on the RMU (Ring Main Unit) will be the self-powered type.

18.6.7 415 V Motor Control Centres

The LV MCCs will be single-sided and housed in the LV switchroom. Construction of all MCCs will have Form 4b segregation, Type 2 coordination. Starters in MCCs will be of demountable design and main incoming circuit breakers will be of withdraw-able design complete with protection. Motor starters up to 90 kW will be equipped thermal overload protection and electronic protection used for all larger drives. The LV MCCs will supply power to the low voltage motors, low voltage variable speed drives and low voltage distribution boards.

18.6.8 Electronic Variable Speed Drives and Soft Starters

Low voltage variable speed drive (VSD) units will be supplied from the LV MCCs. These units will be installed along the internal wall of the relevant LV electrical switchrooms. LV VSDs rated 90 kW or greater will have active front-end converters to improve the power factor and power quality to the power system.

18.6.9 Fire Protection

All switchrooms will be provided with local fire detection systems consisting of Very Early Smoke Detection Apparatus (VESDA) sampling for the switchboard. Signals from the fire detection system will be wired to the respective Fire Indication Panel (FIP) in the switchrooms and all signals will be monitored by a master fire detection panel (MFIP) in the Administration Building. Each FIP will also be wired to a local siren with beacon to warn staff of the fire detection.

18.6.10 Earthing system and Lightning Protection

The earthing system within the plant will be designed in accordance with relevant Australian Standards and equivalent IEC Standards. The following method of system earthing will be implemented at various voltage levels:

- 11 kV Earthed via earthing transformer or NER at 66 / 11 kV Substation
- 415 V Solidly earthed system / Multiple Earthed Neutral (MEN) / T-N-C-S

Earth stakes and grading rings will be provided around the switchrooms to mitigate against step and touch potential risks.

Lightning protection will be provided for buildings and structural steel as appropriate. Lightning protection systems will have their own independent earthing electrodes and will be interconnected with the power earthing system.

18.6.11 Electrical Field Installation

Cables up to 25 mm^2 will be PVC insulated and larger cables will be XLPE insulated.

VSD cables will be three phase and three earth cables symmetrically laid out within an overall shielded cable.

In general, cables within the plant area will be installed above ground on cable ladders and follow the pipe racks wherever possible. Cables to equipment in open areas such as process water pumps and concentration pumps will be partially installed underground in conduits for ease of access and to minimise clashes with pipework.

Cable ladders will generally be laid horizontally, with vertical ladders only used in areas where regular spillage may occur. Hot dip galvanised cable ladder will be used throughout the plant with stainless steel cable ladder being used in the acid regen circuit. Ladder routes will in general follow the mechanical pipe racks.

Cables of different voltage groups will generally be installed on separate ladders. Where they need to be installed together, segregation in the form of barrier strips will be provided.

Sun cover will be provided over the top level of all cable ladder to provide protection against UV damage and plant spillage.

Plant lighting will be designed in a fit for purpose manner to suit the operational requirements of the plant. LED luminaires will be used to maximise light spread and energy efficiency. Enclosed areas and staircases will be fitted with traditional swivel lighting poles. Vibration resistant fittings and auxiliaries will be used where required. Flood lights and high-bay luminaires will be provided for perimeter, general area and workshops lighting.

UPS maintained emergency light fittings will be installed as required throughout the plant to ensure that personnel can safely negotiate obstacles in substations, control rooms, stairways, access ways and safety shower locations. Emergency lighting will comply with AS 2293.

18.7 Potable Water

The potable water demand for the Project has been calculated on a per capita usage basis and is summarised in Table 18.7-1 below.

Table 18.7-1 Potable Water Demand

Area	No. of Personnel	Usage (L/person/day)	Demand (m^3/day)
Accommodation Camp	100	300	30
Plant	171	70	12
Other	187	70	13
Total			**85**

Raw water will be sourced from bores around the site and pumped to a raw water storage tank located adjacent to the potable water treatment facility at the process plant.

A vendor package modular potable water treatment plant including filtration, ultra-violet sterilisation and chlorination will be installed. Potable water will be stored in the plant potable water tank and will be reticulated to the plant buildings, site ablutions, safety showers and other potable water outlets.

Potable water for the accommodation camp will be pumped from the water treatment plant at the plant site to a water storage tank at the camp. The tank will provide two-day reserve storage in the event of supply line disruptions. Water will be delivered into the camp reticulation system using a constant pressure variable flow pump system. The pump skid will include a UV disinfection unit to provide additional security against contamination.

18.8 Sewage

Effluent from all water fixtures in the process plant, mine services area and accommodation camp will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a vendor package sewage treatment plant system located at the process plant.

Treated effluent will be discharged into leach drains or the plant tails hopper. Treatment plant sludge will be suitable for direct landfill burial in unlined pits.

18.9 Fuel and Lubricant Supply

A vendor supplied fuel storage and pumping system will be supplied as part of the fuel supply contract. This will include:

- two 12,000 m^3 double skinned self-bunded fuel storage tanks and pump skids
- a transfer pumping system to fuel the heavy vehicles in the mine services fuel bay
- a transfer pumping system to fuel the light vehicle bowsers
- a lubricants storage facility.

The fuel farm will include an integrated fire control system.

18.10 Solid and Hydrocarbon Wastes

Wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

Waste lubricating oils will be returned to the supplier for recycling.

General solid wastes will be deposited into a landfill at the toe of the mine waste dump and promptly covered to deter vermin and scavengers.

Materials such as cyanide packaging will be decontaminated and buried, under supervision, on site beneath mine waste to prevent unauthorised use, or burned in an appropriate incinerator.

18.11 Communication System Infrastructure

Internal communications and IT services will be via a site wide fibre optic network.

One of the local mobile phone providers will be contracted to install facilities on site and provide a link into the local, national and international telecommunication network.

A radio network will be established with dedicated operational, security and emergency channels.

A local ground station will be installed to provide a data connection via microwave to the local ISP.

18.12 Explosive Storage and Handling

An emulsion plant for mixing and storage of ANFO (ammonium nitrate / fuel oil) will be established by contract with a reputable supplier. The cost of this facility has been included in the explosive supply cost in the operating cost estimate. The facility will be located south of the mine services area and accessed by an unsealed road. The facility will be fully fenced and will include security.

A high explosive magazine will be built and located separately. This will store high explosives and detonators and will include a blast berm and will be fully fenced.

18.13 Security and Fencing

18.13.1 Perimeter Fencing

Security fencing will be installed around the perimeter of the pit to protect the local residents from entering within the blast zone and potentially interrupting the mining operations.

18.13.2 Accommodation

The accommodation camp will be perimeter fenced and will include a security gatehouse and security guards on 24-hour duty.

18.13.3 Process Plant Security

The process plant area will have two levels of security and fencing:

- A perimeter fence and security gatehouse will regulate entry of personnel to the low security buildings including the main administration building and warehouse.

- A double fence separated by 10 m will surround the high security process plant. A separate gatehouse with turnstile and search facility will regulate personnel entry to the process plant itself.

18.13.4 Goldroom Security

The goldroom area will be fenced off separately. Goldroom security will include magnetic locks with a password entry together with motion sensors, CCTV and proprietary security.

18.14 Site Buildings

Layouts for site buildings were developed for scope definition purposes. These drawings were used to develop cost estimates for the study. During project implementation, these drawings will be provided to potential vendors for the purposes of preparing their bids. Their bids will, however, be based on the supply of their standard buildings that meet the buildings' functional requirement but may differ in detail.

18.14.1 Mine Services Area

The mine services area (MSA) has provision for offices, workshops and other facilities to support the mining operation. All the buildings in the MSA will be supplied, constructed and maintained by the Mining contractor.

Typical facilities within the mine services area will include:

- **Mine Office:** The office will be of pre-fabricated modular construction. The building will include office, data room, ablution and meals area.
- **Mine Warehouse:** Mine warehouse of clad steel frame construction with external hardstand and an internal sea container office. The facility will include internal racking and fit out.
- **Mining Vehicle Wash down Facility:** The vehicle wash down facility will be used for the cleaning of mining and mobile equipment. It will include a drive-in sump, water management system and will be fitted with an oil / water separator, which will also capture grease for subsequent disposal. A standpipe for water truck fill is typically situated at this location and adjacent to this facility will be a light vehicle wash down area draining to the same sump.
- **Mine Shift Change:** This facility will be based around converted 40-foot containers and will include male / female showers, lockers and ablutions.

- **Heavy Vehicle Workshop:** A three bay workshop of clad steel frame construction with additional welding and tyre change bays on either end of the building. A 20-t overhead crane will service the three maintenance bays and welding bay. The facility will have a concrete floor slab with an approach apron. One bay will have embedded rails for the maintenance of tracked vehicles. A compressor and air receiver will be provided. Converted shipping containers will be used as external tools / materials stores and offices. Adjacent to this facility will be a smaller light vehicle workshop suitable for servicing light vehicles, including buses, road trucks and forklifts.

18.14.2 General and Administration Facilities

The administration area north of the processing plant will include the following:

- **Main Administration Building:** The main administration office will be of pre-fabricated modular construction. The building will incorporate administration office facilities with reception, conference room, training room, offices and server room, ablution and meals area.

- **Clinic:** The clinic will include a reception, waiting area, treatment room, paramedics' office, stores, ablution, change room, rest room and a covered ambulance bay.

- **Laboratory:** The laboratory will be of pre-fabricated modular construction with sample preparation, fire assay, wet and dry facilities to process grade control and plant process control samples. Incorporated in this building will be offices and a laboratory for the site environmental monitoring team. All laboratory and safety equipment will be supplied by the company selected to manage the laboratory.

18.14.3 Process Plant Area Buildings

- **Plant area:** The plant area will have a main access road for reagent supply. Within the process plant area will be the following support facilities:

- **Plant Meeting and Training Room:** The plant meeting and training room will be of pre-fabricated modular construction installed on a concrete slab.

- **Plant Office:** The plant office will be of pre-fabricated modular construction.

- **Warehouse :** Plant workshop of clad steel frame construction with external hardstand and pre-fabricated offices.

- **Reagent Storage Shed:** Two steel framed reagent storage sheds each with concrete floors, sumps and appropriate layout and facilities to isolate, separate and handle all plant reagents.

- **Plant Workshop:** Plant workshop of clad steel frame construction with external hardstand and pre-fabricated offices. Additional storage will be provided in shipping containers as required.

- **Plant Ablutions:** The ablutions will be a pre-fabricated 40-foot container installed on a concrete slab.

- **Plant Crib Room:** This facility will be based around converted 40-foot containers and will include kitchen facilities, tables and chairs.

- **Core Shed – Grade Control Facility:** A core shed / grade control facility will be provided by means of a double row of 40-foot shipping containers modified to suit with a covered arched polyethylene roof and placed on a raised earth gravel floor.

18.15 Workforce Accommodation

18.15.1 Permanent Accommodation Camp

The 100-bed accommodation camp will be located approximately 1 km north of the process plant and will provide accommodation for salaried and security staff not originating from the local area.

The camp will be constructed from blockwork and will consist of the following:

- Eight management units – two blocks of four units.

- 72 accommodation units – Nine blocks of eight units.

- 20 man fully furnished containerised units – relocated from construction camp.

- Dry mess with food storage and preparation, kitchen and dining facilities and camp administration office.

- Wet mess with a bar, TV area and store room.

- Combined laundry building / gymnasium incorporating ablution facilities for camp staff.

Costing has been based on the already built camp facilities at Endeavour's Karma site.

18.15.2 Temporary Construction Accommodation

The existing operations camp supplemented with containerised accommodation units will be used for pioneer accommodation pending early completion of the new permanent camp. These containerised units will be moved to the permanent camp once is it operational.

The permanent camp will then be used for Endeavour and EPCM contractor staff with any surplus capacity made available to senior contractor personnel.

Contractors will be required to provide their own accommodation for their construction workforce. An area will be set aside for temporary contractor camps adjacent to the construction site. Alternatively, some contractors may choose to source temporary accommodation in nearby towns and villages for Malian workers.

18.16 Tailings and Water Management

18.17 Tailings Storage Facility

18.17.1 Design Objectives

The design objectives for the Tailings Storage Facility (TSF) are as follows:

- Permanent and secure containment of all solid waste materials (tailings) generated by the process plant.
- Maximisation of tailings densities using subaerial deposition.
- Removal and reuse of free water as much as practicable.
- Reduction and control of seepage.
- Excess storage capacity to retain ANCOLD-prescribed design storms and annual rainfall sequence, including containment of runoff from upstream catchments.
- Rapid and effective rehabilitation.
- Ease of operation.
- Monitoring network comprising embankment piezometers, survey pins and groundwater bores.

18.17.2 Design Criteria

The TSF has been designed to ANCOLD guidelines. Design criteria for the TSF are summarised in Table 18.17-1.

An ANCOLD consequence category of 'High A' was determined on the basis of a potential PAR (Public At Risk) in the range of '≥100 to <1,000' and a Severity Level of 'MAJOR'.

Table 18.17-1 Tailings Storage Facility Design Criteria and Specifications

Design Standards	
TSF Consequence Category	High A
Dam Spill Consequence Category	Significant
TSF Stormwater Storage Capacity	Average supernatant pond superimposed with greater of: • 100-year ARI (Average Recurrence Interval) wet year peak pond volume • 100-year ARI, 72 hr flood and • 10-year ARI wet season runoff (assuming 100% runoff and no evaporation)
TSF Emergency Spillway: • Spillway capacity • Erosion protection	 PMF (Probable Maximum Flood) / critical duration 100-year ARI / critical duration
TSF Closure Spillway: • Spillway capacity • Erosion protection	 PMF / critical duration 100-year ARI / critical duration
Contingency Freeboard • Wave Run-up • Additional Freeboard	 10-year ARI Wind 0.3 m
Earthquake Loading • Operating • Final	 Operating Basis Earthquake (OBE) Safety Evaluation Earthquake (SEE)
Operations	
Capacity • Final • Starter	 35.6 Mt of dry tails 4.8 Mt of dry tails (18 months initial capacity)
Production Rate	3.2 Mtpa.
Slurry Characteristics • Beach Slope[*1] • Density – Stage 1 – Final	 150H:1V 1.12 t/m^3 (from density modelling) 1.36 t/m^3 (from density modelling)

18.17.3 Design Summary

The TSF will comprise a two-cell paddock storage facility formed by multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 77 ha for the Stage 1 TSF, increasing to 239 ha for the final TSF. For Stage 1, only Cell 1 will be constructed. Cell 2 will be commissioned in Year 3 of operation.

The TSF is designed to accommodate a total of 35.6 Mt of tailings. The Stage 1 TSF is designed for 18 months storage capacity. Subsequently, the TSF will be constructed in annual raises to suit storage requirements, however this may be adjusted to biennial raises to suit mine scheduling during the operation.

Downstream raise construction methods will be utilised for all TSF embankment raises of the perimeter embankments.

Staged embankment crest elevations and heights are presented in Table 18.17-2.

Table 18.17-2 Staged Embankment Construction

Stage	Tailings Storage (Cumulative) (Mt)	TSF Embankment Elevation (m RL)		Maximum TSF Embankment Height (m)
		Cell 1	Cell 2 [*2]	
1[*1]	4.8	390.5	-	15
2	8.0	394.5	-	19
3	11.2	396.2	396.2	20
4	14.4	397.7	397.7	22
5	17.6	399.5	399.5	24
6	20.8	401.1	401.1	25
7	24.0	402.6	402.6	27
8	27.2	404.0	404.0	28
9	30.7	405.4	405.4	29
10	35.6	407.5	407.5	31

[*1] ***Stage 1 embankment designed for 18-month storage capacity.***
[*2] ***Cell 2 will be commissioned in Stage 3.***

The embankments will have an upstream slope of 3H:1V (to facilitate HDPE (High-density polyethylene) geomembrane liner installation), an operating downstream slope of 3H:1V and a minimum crest width of 8 m. The final downstream embankment profile will consist of an overall slope of approximately 3.5H:1V, comprising a 3H:1V slope with 5 m benches at 10 m vertical intervals. The perimeter embankment upstream face will be lined with textured HDPE geomembrane liner, to allow safe egress from the TSF. A downstream seepage collection system will be installed within and downstream of the TSF perimeter embankments, to capture and return seepage from the TSF.

The TSF perimeter embankment comprises an upstream low permeability zone (Zone A), and downstream structural fill zone (Zone C1). A chimney drain (Zone F1) is included within the Stage 1 embankment cross section, as the embankment will be water retaining for the initial stages of operation. Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability zone of the embankments.
- Zone C1 material shall be delivered to the embankment by the mining operation, levelled with a dozer and traffic compacted by loaded haul trucks on an ongoing basis during the operation. When material from the open pits is not available, Zone C material shall be won from borrow.
- Zone F1 shall be clean sand / gravel drainage material supplied to a stockpile adjacent to the works.

A cut off trench will be located beneath the upstream low permeability zone (Zone A) of the embankments. The cut-off trench will be excavated to extend through to competent low permeability foundation material and backfilled with low permeability fill (Zone A).

Once Cell 2 is commissioned (in Stage 3), the Cell 1 northern embankment will become the divider embankment between the two cells. HDPE geomembrane liner will be installed over the Stage 2 embankment crest to connect the Cell 1 and Cell 2 basin liners. The divider embankment will then be raised using centreline construction methods for each embankment raise.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 200 mm thick compacted soil liner will be constructed in the TSF basin area, comprising either reworked in-situ material or imported low permeability material. The area within the TSF basin will be lined with 1.5 mm HDPE geomembrane liner, overlying the compacted soil liner. The basin HDPE geomembrane liner will be extended over the divider embankment at the Stage 2 crest elevation, to ensure full lining of the TSF basin.

The TSF design incorporates an underdrainage system to reduce pressure head acting on the basin liner system, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The underdrainage system comprises a network of collector and finger drains. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin.

A leakage collection and recovery system (LCRS) will be installed beneath the basin composite liner, and reduce water pressure build-up on the HDPE liner. Solution recovered from the underdrainage system and LCRS will be released to the top of the tailings mass via submersible pump, reporting to the supernatant pond.

Supernatant water will be removed from the TSF via submersible pumps located within a decant tower in each cell. A starter decant tower will be constructed in each cell, to allow early water recovery until the supernatant pond is located at the permanent decant tower location. The supernatant pond will be maintained in the centre of the TSF, adjacent to centre of the divider embankment in each cell. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

An operational emergency spillway will be available at all times for each cell during TSF operation, constructed in the eastern embankment of the TSF in order to protect the integrity of the constructed embankments in the event of emergency overflow. Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the TSF embankment.

A pipeline containment trench will be constructed during Stage 1 to contain both the tailings delivery pipeline and decant return pipeline between the TSF and Plant Site, to reduce environmental impact if the pipeline bursts.

18.17.4 Monitoring

A total of four groundwater monitoring stations will be installed downstream of the TSF to facilitate early detection of changes in groundwater level and/or quality, both during the operating life and following decommissioning.

Standpipe piezometers will be installed in the TSF embankment to monitor pore water pressures at several locations within the embankments to inform stability assessments.

Survey pins will be installed at regular intervals along the TSF embankment crest in order to monitor embankment movements and assess effects of any such movement on the embankment.

18.17.5 Tailings Testing

Physical and geochemical testing of combined tailings samples derived from the different ore bodies was conducted during the study.

The rate of supernatant release for all samples sample was quick and reached typical dry densities, with a good increase due to drying and consolidation. Assuming that the facility is efficiently operated, it is estimated that the average settled density for the tailings mass will be approximately 1.35 t/m^3.

The results of the geochemistry testing is summarised below:

- The test results indicate that the tailings solids are Non-Acid Forming (NAF).
- The tailings samples were found to have a low number of element enrichments, with arsenic the only metal found to be highly enriched across all samples. Antimony was found to be slightly to significantly enriched in all samples, with four samples slightly to significantly enriched in sulphur and bismuth
- Comparison of the multi-element results to soil quality screening guidelines indicated that the tailings samples did not meet the guidelines for human health, ecology or site contamination due to elevated concentrations of metals and metalloids.
- The supernatant samples did not meet water quality standards for release of water from mining operations, with exceedances recorded in the pre-detox tailings sample for fluoride and sulphate.
- The four arsenic precipitate tailings samples recorded a greater number of exceedances, with total dissolved solids (TDS), arsenic, mercury and sulphate elevated in all samples. Total cyanide was found to elevated above the guidelines in two samples.

On the basis of the tailings solids multi-element results and the elevated arsenic and post detox cyanide concentrations, the TSF has been designed to prevent the loss of tailings solids. The TSF design incorporates a compacted soil liner overlain by HDPE geomembrane and drainage collection system above and beneath the geomembrane basin liner. The TSF design also includes freeboard capacity to provide dilution of the supernatant prior to any spillway flow.

18.17.6 Closure Summary

At the end of the TSF operation, the downstream faces of the embankments will have a slope of 3H:1V, with 5 m wide benches located at 10 m height intervals, for an overall slope profile of approximately 3.5H:1V. The profile will be inherently stable under both normal and seismic loading conditions, will provide a stable surface water drainage system and will allow for revegetation.

Rehabilitation of the tailings surface will commence upon termination of deposition into the TSF. The closure spillway will be constructed in such a manner as to allow rainfall runoff from the surface of the rehabilitated TSF to discharge via the closure spillway.

The TSF closure spillway will be excavated after the remaining supernatant water is proven to be suitable for release and during rehabilitation of the tailings surface subsequent to decommissioning. The spillway will be constructed by excavating a channel from the lowest point of the final tailings beach (adjacent to the eastern perimeter embankment in each cell), through the eastern embankment to discharge into the adjacent downstream water course. The TSF will be a fully water-shedding structure on closure. The closure spillway will allow conveyance of PMP (Probable Maximum Precipitation) storm events without any attenuation in the TSF.

The final soil cover for the tailings surface subsequent to decommissioning will be confirmed during operation based on ongoing operational tailings geochemistry testing results. The following cover design for the tailings beach has been adopted at this stage but is subject to ongoing testing and review:

- Mine waste capillary break (500 mm).
- Low permeability fill layer (300 mm).
- Topsoil growth medium layer (200 mm).
- The finished surface will be shallow ripped and seeded with shrubs and grasses.

18.18 Water Management

The management of water is a critical aspect of the design for Kalana. In order to understand (and control) the flow of water around the site, a water balance model was developed.

The primary objectives of the water balance modelling are summarised below:

- Determine supernatant pond volumes within the TSF under average climatic conditions throughout operation.
- Determine supernatant pond volumes within the TSF for design wet rainfall sequences and storm events, check TSF storm water storage capacity and confirm the suitability of the current TSF design philosophy.
- Determine staged embankment crest elevations, to ensure containment of tailings and design supernatant pond volumes.
- Determine the likely magnitude and frequency of recycle water shortfalls during average conditions and design dry rainfall sequences.
- Assess the impacts of pit dewatering on the site water management systems, and determine peak dewatering rates for the process, and peak discharge rates for pit dewatering.
- Assess risk factors for water balance modelling.

The water balance modelling included the TSF with a view to determining site water storage requirements. Design wet conditions were modelled to ensure that the TSF is designed with sufficient storage capacities to comply with design criteria.

A constant pit dewatering rate of 605 m^3/h from the Kalana Pit was used for the water balance modelling and will be available to provide make up and raw water to the process plant. It was assumed that any excess water generated from pit dewatering will be discharged directly off site.

The quality of the pit inflows was not considered in the water balance modelling (i.e. the modelling assumed that pit dewatering quality is suitable as process water make up without polishing).

Key findings from the water balance modelling are as follows:

- The TSF is designed to hold the tailings plus the design rainfall conditions, and thus has sufficient storm water storage capacity for all design storm events and rainfall sequences.
- The supernatant pond volume peaks in September each year (at the end of the wet season), before returning to the minimum operating pond volume during the subsequent dry season.
- Decant return / process water shortfall is expected to occur under average and design dry climatic conditions.
- The Kalana Pit dewatering supply (605 m^3/h) is sufficient to supply all raw and make-up water requirements for average and design dry climatic conditions.
- The Kalana Pit dewatering supply to the process plant should have a capacity to supply the total water in tailings slurry (plus additional supply) to the process plant, to cover the event in which tailings decant return is not available.
- The maximum annual discharge to the environment of excess water from the Kalana pit dewatering (after abstraction of raw water requirements) under design wet conditions is 3.6 Mm^3/year.

18.19 Hydrogeology

A revised hydrogeological conceptual and a more robust numerical groundwater model were constructed to refine predicted pit inflows and develop a conceptual design of the dewatering borefield for the Kalana Pit. The new numerical model was developed to include the historic underground workings and be calibrated to a measured water level rise once dewatering operations ceased in the previous workings. The model predicted a pit inflow rate of 14,518 m^3/day (605 m^3/h) under steady state conditions.

A preliminary groundwater desktop assessment was conducted for the Kalanako Pit. With limited hydrogeological data available for the Kalanako ore body area, conclusions were based on data from the Kalana area assuming similar hydrogeological conditions. On this basis the preliminary, estimated total Kalanako inflow for both pits ranges between 3,110 m^3/day (129 m^3/h) and 4,147 m^3/day (173 m^3/h).

18.20 Hydrology

Hydrologic and hydraulic modelling was completed for the Kalana Project site and its surrounds, with a view to sizing the following structures:

- TSF diversion channel.
- Waste dump diversion channel.
- Kalana Pit protection bund and diversion channel.

Design criteria for surface water management structures are summarised in Table 18.20-1.

Table 18.20-1 Surface Water Management System Design Criteria

Parameter	Detail
Channel Design	
Flow capacity • Diversion Channels	100-year ARI storm event (duration equal to time of concentration)
Diversion Bund Design	
Crest Elevation	100-year ARI storm event (duration equal to time of concentration), plus 0.5 m (minimum) freeboard.

All structures were incorporated in the hydraulic modelling, including all floodway and culvert crossings prevalent throughout the site.

The TSF and waste dump diversions consist of a diversion channel with a 10 m base width. The maximum diversion channel depth is approximately 15 m and 18 m respectively. It is anticipated that the channel excavation will be used as a source of embankment fill for the TSF.

The Kalana Pit protection bunds were designed at the 100-year ARI flood level plus 500 mm freeboard. The bunds will have a crest width of 3 m and side slopes of 2H:1V. The Kalana Pit diversion channel comprises a channel with a 7 m base. The maximum diversion channel depth is approximately 7 m, with side slopes of 2H:1V.

Several combined culvert and floodway crossings will be required to facilitate drainage under the site haul roads and the Kalana Bypass Road, which intersect these diversions.

18.21 Kalana Bypass Road

The design objectives for the Kalana Bypass Road are as follows:

- Provide suitable access to the project area and Kalana village from the existing highway, including allowance for design speeds.
- Full road operability during design storm events with consideration of reducing the earthworks volumes.

Kalana Bypass Road design parameters are provided in Table 18.21-1.

Table 18.21-1 Kalana Bypass Road Design Parameters

Design	
Road Cross Section	Formation Width – 7 m Lane Width – 3.5 m Shoulder width – 1.0 m Safety Bunds – 0.5 m height where fill height > 2 m Crossfall – 2%
Maximum Vertical Grade	8%
Culvert Sizing	2,400 mm by 2,400 mm precast concrete box culverts
Culvert Design Criteria: • Culvert Design Criteria	 100-year ARI
Pavement	150 mm laterite gravel base course Sealed running surface to comply with statutory requirements

The existing Kalana village access road will be decommissioned and realigned to the northeast around the Kalana Pit. The Kalana Bypass Road (4.7 km long) runs from the Kalana village to the existing highway. The road comprises two 3.5 m width running lanes with a 1 m shoulder each side, for a total formation width of 9 m. The Kalana Bypass Road running surface will be overlain by a road sealing designed to comply with statutory requirements. Linemarking, kerbs and other appurtenances will be incorporated into the Kalana Bypass Road design during the Feasibility Design.

The culvert crossing for the Kalana Bypass Road are designed to convey 100-year Average Recurrence Interval (ARI) storm event without the flow overtopping the road.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Markets

The gold market is highly liquid and benefits from terminal markets (London, New York, Tokyo, and Hong Kong) on almost a continuous basis. Gold prices were in general downward trend from 1980 to 2000 where gold traded down to approximately $250/oz. From 2000 the price increased annually until 2011 and 2012 where the price peaked at just under $1,800/oz. There was a sharp correction to the gold price in 2013 with the end of Quantitative Easing monetary policy by the US Reserve Bank. Since then gold has remained range bound between $1,350/oz and $1,050/oz.

Since 2019, gold price increased steadily from the $1,300/oz level to about $1,800/oz today, with a peak above $2,000/oz in August 2020.

Gold produced at the mine site will be shipped from site, under secured conditions, to a refining company. Under proposed contractual conditions, the refiner purchases the gold from the mine with the proceeds automatically credited to the mine's bank account. Gold production will be sold on the spot market, with no plan currently to hedge any sales.

19.2 Contracts

It is Endeavour Mining strategy to outsource mining activities to contractors and in all instances ensure that the mining operation can purchase the equipment at the end of the contract period at its depreciated price or, should the contractor default, at a pre-determined pricing mechanism. Prior to start-up, all major mining contractors are requested to tender and the most appropriate tender is accepted, thereby ensuring that the best competitive current pricing is achieved. Care is taken at the time of finalising contracts to ensure that the rise and fall formula is totally representative of the build-up of the quoted price per unit. At the time of award, prices quoted are compared to benchmark prices of other owner mine operations.

The contract mining costs are highly dependent on when tenders are issued as the price of major equipment varies dependent on demand as well as the cost of finance. Rise and fall can be negatively affected by currency fluctuations as well as price squeeze due to scarcity.

It is also expected that the future operation of the Kalana project will contract the following services to specific subcontractors:

- Drilling services.
- Laboratory services.
- Supply of diesel and lubricants.

- Supply of electrical power with national electrical power provider.
- Security services provider.
- Camp catering services provider.
- Gold refining services.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Law and Permitting

The Project comprises an exploitation permit (Permis d'Exploitation) registered to SOMIKA (the Permit). The Permit was transferred from Avnel on 23 July 2003 to SOMIKA.

The Permit is a unique 30-year exploitation permit that is derived from the legislation originally passed to enable SOGEMORK to develop the Kalana Mine. The National Cleaning, Pollution and Nuisance Control Department (DNACPN) monitors and ensure that environmental issues are considered through sectorial policies and development programmes, and that respective decided measures are implemented.

An Environmental and Social Impact Assessment (ESIA) was undertaken and submitted to the DNACPN on 29 January 2016 and a clarification meeting was held on 10 March 2016. This submission was accompanied by a Resettlement Action Plan (RAP) for the families to be resettled that are located within the 350 m buffer zone from the Kalana Open Pit. Subsequent to the clarification meeting, the Final ESIA was submitted on 21 March 2016 for approval by the Minister of Environment. The ESIA for the Kalana Project was approved and the environmental permit issued on 28 April 2016 by the 'Ministere de L'environnement, de L'assainissement et du Developpement Durable'.

According to the new design including the new Kalanako pit, the exclusion zone will be extended. The extension of the pit size inside Kalana will result in a change in the perimeter of the part of the village to be relocated and will increase the numbers of people that will be affected by the resettlement. There has also been construction on existing properties in the former impacted areas that have been approved by the Mediation Committee. In addition, the national road that was sealed by the government in 2019 will require a sealed deviation. These changes and additions will affect the current ESIA and RAP, which will need to be addressed in an ESIA and RAP amendment to be submitted to the authorities.

A permit is required for the discharge of water from the pit to ensure a safe working environment and to ensure compliance with the relevant standards. This is included in the ESIA report submitted and has been approved. Due to the changes to the Project since this approval was granted, this will also be captured in the amendment.

Over and above the local Malian laws and guidelines, additional best practice guidelines considered for the Project are described below:

- The Equator Principles (EPs) have been developed in conjunction with the International Finance Corporation (IFC) in an attempt to establish an international standard with which companies must comply with in order to apply for approved funding. The EPs apply to all new project financings globally across all sectors. These EPs are an attempt to 'encourage the development of socially responsible projects, which subscribe to appropriately responsible environmental management practices with a minimum negative impact on project-affected ecosystems and community-based upliftment and empowering interactions'.

- The IFC's sustainability framework includes the Policy and Performance Standards on Social and Environmental Sustainability and provides the private sector clients with a clear and comprehensive view of requirements early in their engagement with IFC. The IFC have provided a range of technical reference documents with general and industry-specific examples of Good International Industry Practice (GIIP).

- The IFC's Guidance Notes: Performance Standards on Environmental and Social Sustainability are technical reference documents with general and industry-specific examples of GIIP. These Guidance Notes are designed to be used together with the relevant industry sector EHS guidelines (such as for mining). The Guidance Notes for environment consist of (inter alia):

  - air emissions
  - ambient air quality
  - energy conservation
  - wastewater and ambient water quality
  - water conservation
  - hazardous materials management
  - waste management
  - noise and contaminated land.

- The International Cyanide Management Code (ICMC) is a voluntary initiative for the gold mining industry, the producers and transporters of the cyanide used in gold mining. It is intended to complement an operation's existing regulatory requirements. Measures to prevent access by wildlife and livestock are required for all open waters (examples tailings impoundments and leach ponds) where WAD cyanide is in excess of 50 mg/L.

20.2 Baseline Environment

20.2.1 Climate

The Project area is in the pre-Guinean climate zone, where the average rainfall is greater than 1,000 mm and may attain 1,200 mm, with a maximum of 85 days of rain and a rainy season lasting six months, from May to October, which is characterised by torrential rains. There is a large variability in average annual rainfall. The temperatures in the region vary depending upon both latitude and season, with a mean annual temperature of approximately 28°C.

20.2.2 Topography

Mali is divided into three natural zones: the southern cultivated Sudanese zone, central semiarid Sahelian zone, and northern arid Saharan zone. The Project area falls within the cultivated Sudanese zone. The terrain is primarily savannah with generally a flat topography consisting of plains and plateaus ranging up to 420 m.

20.2.3 Geology

Mesothermal gold mineralisation hosted mainly in quartz veins follows shallow dipping and vertical dykes adjacent to small intrusions emplaced in a sub-volcanic environment. The Kalana deposit is hosted by volcano-sedimentary rocks of the lower part of the Upper Birimian Group and is discussed further in Section 3.

20.2.4 Geochemistry

Fifty-one samples from the mineralised zones and six samples from the mineralised zones were tested to determine the potential for acid mine drainage (AMD). Less than 10% of the samples tested were classified as potentially acid forming. The majority of the bulk waste samples have relatively high acid neutralising capacity. Thus, the potential for co-disposal to mitigate the risk of acid formation is high.

A series of cyanide detoxification tests conducted on a range of metallurgical sample composites was able to demonstrate in all cases that the weak acid dissociable cyanide (CN_{WAD}) concentration could be reduced to <2 mg/L from an initial concentration of ~200 mg/L. This final concentration is well below the ICMC compliance limit of 50 mg/L CN_{WAD}. In addition, arsenic removal tests performed on the detoxified liquors demonstrated that soluble arsenic could be precipitated as a benign hydrated ferric arsenate complex using the scorodite process to consistently achieve <1 mg/L arsenic concentration in the final tailings solution.

20.2.5 Soils, Land Use and Land Capability

The baseline study undertaken for soils, land capability and land-use indicate that 20% of the Project site can be used for arable land, 70% of area is rated as marginal potential for arable crop production and is recommended for natural grazing and/or natural open woodlands, 9.9% of the area can be classified as wetland. The soil in the Kalana Mine region has a low soil fertility, with essential nutrients like phosphate (P), potassium (K), calcium (Ca) and magnesium (Mg) showing low values. All the micro-elements like iron (Fe), copper (Cu), manganese (Mn) and zinc (Zn) show low values. It is recommended that fertiliser and lime are added to select arable land portions for agronomic crop production.

20.2.6 Surface Water

The site is located within the upper Niger Basin which is drained by the non-perennial Kalanako stream and Bale River, which eventually feeds into the Sèlenguè Reservoir which discharges into the Niger River and finally the Atlantic Ocean at the Niger Delta. Local communities have an intimate reliance on the water resource for consumption (direct drinking), cooking and personal hygiene, as well as for agriculture maintenance and livestock watering. Communities also rely on fish consumption and various invertebrate species caught from the water resource, harvesting and consumption of aquatic plants and utilisation of clay soils collected from the edges of watercourses for the manufacturing of bricks.

Although some water quality elements were shown to be elevated within the watercourses of the area (copper, iron, aluminium), these elements are naturally occurring in the soils and geology of the area and therefore expected. Elevated levels of arsenic (As) and lead (Pb) in the immediate vicinity of the existing TSF were also detected and are considered associated with groundwater seeps from the TSF, with the zone of influence localised. Escherichia coli (E. coli) is present within water catchments at most of the survey sites. The significance of this is low because of the low concentrations present, but it does highlight a potential emerging problem.

20.2.7 Hydrogeology

One main aquifer occurs in the Project area and is associated with the fractured rock material. This aquifer is overlain by an aquitard consisting of saprolite that reduces the hydraulic connection between surface water bodies and the underlying fractured rock aquifer. Two groundwater boreholes / users were identified in the Project area, namely Kalana town and the Kalana Mine technician's camp. The groundwater in both boreholes is used for domestic purposes.

20.2.8 Ecological Assessment

Twenty-four plant species occurring in Mali are red-listed by the International Union for Conservation of Nature or IUCN (IUCN, 2012). Thirteen of these occur in southern Mali in the general vicinity of the Project area. Ten species have a moderate likelihood of occurring in the Project area because of the presence of suitable habitat. Three red-listed species were confirmed during the wet season fieldwork. Afzelia Africana is a widespread tree of woodland and dry forest in the savannah belt of West Africa, but has suffered a decline in numbers because of overexploitation of its timber for the commercial market. African mahogany, Khaya senegalensis, is a widespread species of high-rainfall savannah woodland north of the Congo Basin, occurring from Senegal to Uganda. Blyxa senegalensis, is classified as "data deficient", meaning that the species is poorly known, and insufficient data were available to assess the conservation status of this species. Limited data suggest that this species is endemic to West Africa, occurring in temporary pools on lateritic outcrops.

A baseline survey of large mammals within the Project area was conducted during wet season fieldwork in August 2014 and dry season fieldwork in March 2015. Twenty-four species of mammals were confirmed to occur in the area. Seven IUCN Red-listed species potentially occur within the vicinity of the Kalana study area in southern Mali. Two of these were confirmed to occur during wet season fieldwork:

- Hippopotamus (vulnerable).
- African Elephant (vulnerable).

One hundred and forty-seven bird species were recorded in the Project area during wet season fieldwork, while the dry season fieldwork produced 157 species, with a combined total of 207 species for the study area.

Thirteen reptile species were confirmed in the Project study area during the wet and dry season fieldwork, comprising six snakes, five lizards and two monitors. Most of these are relatively widespread species in West Africa, apart from Gambia Agama, which is a narrow West African endemic confined to south-western Mali, Senegal and Guinea. No threatened reptiles are likely to occur in the Kalana study area. A total of 12 species was confirmed for the Project area during the wet season survey and none were added during dry season fieldwork. While most of these are widespread species and not considered threatened, several species are endemic to the Sudanian savannahs in West Africa, such as the Spiny Reed Frog and Brown Running Frog.

20.2.9 Aquatic and Wetland Survey

The survey area incorporates a network of low-gradient streams and channelled and unchannelled valley-bottom wetland units. The vast majority of the watercourses had a narrow band of floodplain wetland interaction where grasses dominated the vegetation unit. The periphery of these floodplains sees the dominance of large trees, making for distinct and well-developed riparian zones. It was noted that only areas in close proximity to villages were suffering transformation and degradation (mostly through informal mining activities, agriculture and general resource usage from a largely subsistence-based lifestyle of the local people). However, in general, the riparian zones and physical condition of the watercourses are currently in a good ecological state.

20.2.10 Socio-economic

The socio-economic survey focused on the nine localities, namely Kalana (chief town of Commune), Faboula, Daolila, Kobada, Kalako, Koumbala, Solomanina, Diabala and Dalagouè.

Like most rural areas in Mali, the municipality of Gouandiaka is mainly agro-pastoral (practice of agriculture that includes both the growing of crops and the raising of livestock). Besides the agro-pastoral activities, gold panning proves to be a popular activity. It can be considered that access to basic social services is better in the Kalana study area, compared to national benchmarks, as a result of development activities undertaken by Kalana Mine.

Due to the practice of gold washing as well as the establishment of the Kalana Mine, the study area has become a place of ethnic and cultural diversity. Due to the cultural mix that has been created, 30 different languages were recorded as being used in the Kalana study area, six foreign languages (English, Russian, French, Arabic, Spanish and German) in comparison to the 24 African languages spoken. The most common spoken language is Bambara followed by French, Peulh, Sénoufo and Minianka. Islam and Christianity are practiced by the individuals in the Project and surrounding areas. Islam is the most practised religion (96%) and Christianity (3%).

In terms of basic infrastructure, the Project has access to a variety of social infrastructure points, which includes 18 educational facilities (public and private), 19 of 24 public fountains / waterpoints and 11 medical facilities. The surrounding Project area localities have access to 19 public and private schools and five madrassas (privately-funded, Islamic teaching institutions). The Kobada Hamlet does not have access to an education facility and children are required to walk 4 km to the nearest school.

Access to safe water remains a concern in the greater Kalana study area. The socio-economic survey identified 24 public water taps in the Project area of which 19 are located in Kalana town and five in Kalako.

The household survey identified that temporary loans and leasing of land is rare in the Kalana study area and that only two households paid rent for their land and three other households paid in kind. Leasing of land in the Kalana study area was indicated to be very isolated; however, the surrounding areas operated with free loaned land. While the land vests in the State, access and land use is managed by local authorities.

Employment in the Project area can be classified in two broad categories:

- Formal employment: Employment with a formal contract where employees are remunerated according to the contract terms. These individuals would typically be local employees of Endeavour / SOMIKA, teachers, government employees and NGOs.
- Informal employment: is generally self-employment; these are people who carry out their own business in the localities concerned. These individuals are farmers, growers, stockbreeders, tailors, hairdressers, mechanics, carpenters, tradesmen, blacksmiths, masons and welders.

Agriculture was identified as the main economic activity in the nine localities with corn, millet, sorghum, rice, groundnut, beans, cotton, yam and potatoes the main agricultural products.

20.2.11 Archaeology and Cultural Heritage

The cultural and archaeological assessment was undertaken in the nine towns / villages within the study concession. In total 65 sites were identified, comprising 13 archaeological sites, 29 places of worship sites and 23 places of remembrance.

The archaeology study will be updated according to the new Project design during the Definitive Feasibility Study.

20.3 Public Consultation

The public participation process was undertaken in accordance with decree N°2013/0256/MEA-MADAT-SG of 29 January 2013 which stipulates the methods for conducting public participation. A series of public consultation meetings were held for the Project within Kalana town and the surrounding communities. The meetings were attended by the relevant engineering departments on a local, regional and national level.

An issues register was developed to record issues that were raised during the public meetings. These grievances were responded to in the issues register and handed back to the Sub-prefect of Kalana, the Mayor of Kalana town and the village chiefs and their representatives for their approval and/or amendments.

As per previous section a new public consultation will be conducted according to the new Decree and record all additional issues that will be raised during the Definitive Feasibility Study.

20.4 Environmental and Social Impacts

20.4.1 Biological, Physical and Socio-economic Environment

The geochemical considerations for the Project relate to the mobility of arsenic, which is present at elevated concentrations in some of the Kalana study ore samples. Metallurgical testwork has demonstrated that soluble arsenic may be precipitated as a benign hydrated ferric arsenate (scorodite) complex and this process step has been included in the flowsheet for the proposed Kalana treatment plant. Implementation of this arsenic precipitation process, together with the additional safeguard of having a TSF with an HDPE liner to mitigate any leakage, will eliminate the risk of potential arsenic contamination of groundwater.

As part of the 2016 ESIA, Epoch Resources considered the primary socio-economic impacts associated with the Project to be positive, in that residents of the study area and the greater region will be offered employment opportunities during construction and operation phases. This will translate into an improved standard of living for those hired and their families, which will in turn improve local health and education through increased spending on both, as well as improve the status, skill and experience for those hired. During the construction and operational phases of the Project, payment of dividends, royalties, corporate tax, and income tax would also improve the financial capacity of the State to improve community infrastructure and service delivery in the district. The annual patent tax will increase significantly, and this is paid to the Local Government to support local development. Endeavour will also make significant investments in community infrastructure and services through its Community Development Plan (CDP) programme that will last beyond the life of the Project.

National, regional and local businesses and contractors will benefit both directly and indirectly from Project-related construction and operational activities due to purchase of goods and services. Increased incomes and profits of local businesses and major suppliers for purchase of goods and services manufactured and supplied in Mali will be realised.

This increased revenue to local businesses will in turn contribute positively to the local economy through expanded local employment, increased disposable income, and growth in the local consumer base.

Acquisition of land for the Project will cause economic displacement of local farmers and landowners. Disturbed land (including the WRDs, TSF, plant infrastructure and target resettlement area) will be reclaimed wherever possible back to its original land use (primarily agriculture) or an agreed alternative land use. Based on the 2016 ESIA, of the total surface area occupied by the Project infrastructures associated with the pit design 225 ha is occupied by agricultural and pastoral activities and approximately 38 ha by housing / dwellings. The total area to be occupied by the Project and the numbers of affected peoples will be reviewed according to the new design in an amendment to the ESIA and RAP.

The establishment of the various haul roads and the primary access road required the crossing of a number of smaller and larger drainage lines in the Project area. Protection of the planned Kalana and Kalanako Open Pit from flood events associated with the Kalana and Kalanako streams, required the development of a protection berm along the Kalanako stream as well as a small earth wall embankment and channel to divert the flow to the west of the planned Kalana Open Pit to and into the Kalanako stream. Endeavour does not expect these diversions to have a significant impact on the water resources and associated ecosystem services available in the Kalana or Kalanako streams in terms of flow velocities, evaporation or mean annual runoff once construction work is completed.

The maximum drawdown in the groundwater level compared to regional groundwater levels is calculated at between 260 m and 280 m in the lower aquifer. The zone of influence of the groundwater level drawdown cone will extend up to 1,800 m from the planned Kalana Open Pit area and will underlie the Kalanako stream and the non-perennial streams that flow into the Kalanako stream. The drawdown in groundwater level in the mining pit area will occur in the lower fractured rock aquifer, which is overlain by a 35 m to 80 m thick low yielding saprolite aquitard. The thick aquitard forms an effective groundwater flow barrier between the surface water bodies and the fractured rock aquifer. Knight Piésold considers the impact of the groundwater level drawdown cone on the flow volumes in the Kalanako River and tributary streams to be negligible. Six of the boreholes owned by Kalana Mine and the three groundwater supply boreholes to Kalana town are expected to be impacted by the groundwater level drawdown that will develop in the area due to mine dewatering. The maximum drawdowns in groundwater level in each of the boreholes are calculated using the numerical groundwater flow models. Within Kalana town the drawdown is forecast to be 30 m to 60 m.

An air quality emissions inventory for the Kalana Mine operations was established, forming the basis for assessing the impact of the Project on the environment. With appropriate mitigation measures in place, simulated annual average particulate matter (PM) 10 concentrations for the 1.2 Mtpa scenario comply of the World Health Organisation's (WHO) Interim Target-1 (IT-1) and IT-2 guidelines inside the 350 m buffer zone. The highest simulated PM10 concentrations are to the southwest of the pit area (because the prevailing wind direction is from the north east). Similar to the 1.2 Mtpa scenario, simulated annual average PM10 concentrations for the 1.5 Mtpa scenario follow the WHO IT-1 and IT-2 guidelines outside the 350 m buffer zone. All other constituents modelled, including sulphur dioxide (SO_2), nitrous oxides (NOx) and dust fall, are within the limits adopted outside the 350 m buffer zone guidelines. During the early years monitoring of air quality will provide real time information to assess the impact within the exclusion zone. During the Definitive Feasibility Study, simulations for the revised plant throughput of 3.0 Mtpa will be undertaken and included in the ESIA amendment.

Surface blasting of the diorite, if required, will be a minimum of 120 m from the Kalana Open Pit perimeter and blasting of saprock will be at an initial depth of 60 m and at least 80 m horizontally from the planned Kalana Open Pit perimeter. The blast design shows that an exclusion zone of 250 m horizontal from the Kalana Open Pit perimeter is very conservative and will enable safe blasting and minimum risk of damage due to blast vibration. The impact of blasting will be monitored to confirm the blasting design.

Simulations show that IFC criteria (55 decibels or dBA) during the day, 45 dBA during the night and maximum 3 dBA increase at nearest receptor, will most likely be exceeded within the Project buffer zones if an environmental berm is not established. As soon as the planned Kalana Open Pit depth is more than 5 m higher than the heights of the sources within the pit (approximately 10 m and deeper), the Kalana Open Pit walls will effectively reduce levels at Kalana residences outside the buffer zone to acceptable levels. As part of the 2016 ESIA, studies also showed that the optional berm to the south of the Kalana Open Pit will be sufficient to ensure noise levels below IFC criteria, especially at night, during those years when mining activities will mostly be on the surface. These simulations will be confirmed for the 3.0 Mtpa scenario during the Definitive Feasibility Study.

Encroachment into the wetland habitats by mining infrastructure, as well as a possible decrease in the availability of the water resource due to reduced discharge from the underground mining, will lead to the long-term transformation of wetland-dependent biota. Contamination of the surface water resources (through the potential discharge of poor quality effluents), may also lead to displacement of ecologically sensitive features that will affect the long-term conservation of the surface water resources. This will be avoided through the implementation of the pollution prevention measures and overall water management on site. The Project layout has been adapted to avoid, as far as possible, any wetland habitat features, except for the northern perimeter of the Kalana Open Pit where a flood protection berm will affect a limited portion of the wetland. The diversion bund has been designed to remain as close to the Kalana Open Pit as practicable, thereby reducing impact to the wetland. The Kalana Open Pit will continue to discharge through pit dewatering, into the division system which will assist in maintaining the available water resources for the duration of the LoM. Various mitigation measures will also be implemented to prevent the contamination of surface water resources.

The Project requires the partial resettlement of the Kalana community due to the placement of Project infrastructure and ensuring a safe buffer zone between the mining activities and community of Kalana. The development will also result in the economic displacement and compensation of various farmers, in the area due to the establishment of mining infrastructure and associated activities on farms carrying subsistence and cash crops. This is discussed in more detail in Section 20.5.

The habitat and biodiversity of the Kalana study area is already significantly affected and the planned Kalana Open Pit is therefore expected to have a limited impact on habitat fragmentation and loss of biodiversity due to land clearing and establishment of mining infrastructure.

The mining and associated activities, including the handling and storage of hazardous materials has the potential to impact on the surface water and ground water quality and may result in an increased sediment load on the receiving water bodies, if not managed well. Measures have been included in the design of the Project to ensure the handling, storage and use of the hazardous materials do not pose a direct impact to the environment.

The Kalana Open Pit will result in a pit as final landform to the local community, and the design of the final landform is important to ensure the safety of the local community and their livestock. The Project will however transform the land use permanently where the pit, WRD and TSF facility is established.

The reclamation of the old TSF facilities will result in a long term positive impact on the land use and water resources in the Kalana study area due to the removal of the tailings material and the deposition thereof within the new lined TSF facility after processing at the new plant.

Malaria is the most frequently occurring disease in the study area. Project-related water storage features, including the TSF, pit lake and diversion structures may lead to increased malaria in the Kalana study area, thereby increasing the risk of infection within local communities and Project accommodation camps. Through the implementation of the national programmes and ongoing Company initiatives, an effective malaria control programme will significantly reduce the malaria contraction rate in the Kalana study area.

The incidence of human immunodeficiency virus (HIV) infection and other sexually transmitted diseases (STDs) may increase as a result of in-migration of workers seeking employment. Outbreaks of tuberculosis (TB) and acute respiratory infections can be facilitated by crowded housing conditions and therefore pose a risk for both employees and community members.

Endeavour will transport material and equipment to and from the site via Senegal and Ghana. Routes from both ports experience relatively heavy traffic and congestion and pass through numerous small communities along narrow, unpaved roadways; posing a potential risk to pedestrians and local traffic. Spills or accidents that may occur while transporting chemicals and hydrocarbons from either port may at a later stage have an impact on human health and the environment, depending on the location and timing of the spill. Transport of cyanide will be undertaken as per the requirements of the international cyanide management code which will be implemented by Endeavour.

Traffic and congestion on local public roads will increase during construction as the Project is advanced. The placement of Project infrastructure will require some local residents to travel greater distances, but less than 2 km to avoid mine operations. In addition, local residents, particularly children, lack safety awareness particularly related to road / truck traffic. In summary, Endeavour considers safety risks to vehicles and pedestrians will be increased through Project advancement.

The formation of a pit lake after mine closure may pose a public safety risk, if not managed effectively, while the maintenance of malaria control post closure in the Kalana study area will be important to limit malaria infections.

The archaeological and cultural heritage study revealed the presence of archaeological and cultural resources, and the establishment of Project infrastructure will affect some of these archaeological and cultural resources. Two small cemeteries are located in close proximity to the Kalana Open Pit, namely the camp cemetery of the technicians and the cemetery located behind the office of the Sub-prefect. The implementation will require the relocation of these cemeteries. South of the Kalana Open Pit, there is a third cemetery that requires special measures to ensure it remains unaffected. However, the new waste dumps will cover others archaeological sites. These will be included in the ESIA amendment during the Definite Feasibility Study.

20.4.2 Local Food Production

There is a reliance by the community on the Kalanako stream and specific sections of wetland where certain fish species breed. While the Project development will impact to some extent on the Kalanako stream, the impact assessment found that the wetland will remain largely unaffected by the Project and will continue its ecological function.

Agriculture – Loss of Cultivated Fields and Related Assets

The establishment of mining infrastructure will affect some agricultural resources in the Kalana study area that may not be replaced should the owners select to settle for cash compensation rather than alternative land. This will be addressed through improved agricultural practices and skills transfer relating to agriculture, as well as compensating those parties opting to discontinue their agricultural activities.

Water Supply and Potable Water

The Project development is expected to affect some of the boreholes in the Project area and these will be replaced with boreholes supplying water of similar quality and quantity. The impact on the Kalana stream and greater catchment is expected to be negligible in terms of runoff and water quality and water quality is expected to improve in the catchment due to improved and modern pollution prevention measures associated with the Project.

Energy Resources

A reliable source of energy is required for daily activities, such as cooking and for household warmth. The most important source of energy in Mali is wood and firewood consumption is expanding more rapidly than its market supply. Material cleared during the construction phase will be made available to interested parties for collection and use. A large portion of Kalana town is electrified, thus reducing the demand on local energy resources.

20.5 Resettlement

The Project requires the partial resettlement of Kalana town due to the placement of Project infrastructure and ensuring a safe buffer zone between the mining activities and community of Kalana. The development will also result in the economic displacement and compensation of various farmers, in the area due to the establishment of mining infrastructure and associated activities on farms carrying subsistence and cash crops.

Resettlement and compensation will be undertaken in terms of the relevant Malian legislation, which includes the following key points:

- Determining eligibility for compensation and resettlement assistance, including livelihood initiatives.
- Approach to land access and management.

- Establishing rates of compensation.
- Establishing mechanisms to resolve grievances among affected persons related to compensation and eligibility.
- International best practice guidelines have also been taken into consideration and will be continued during the detailed planning and implementation phases, which include the IFC's Performance Standards and the World Bank Group's Operational Policies.

20.5.1 Resettlement Process

The resettlement process was aligned with the legislative requirements of Mali as well as international best practice guidelines, as described earlier. In summary, the process can be outlined as follows:

- Stakeholder engagement.
- Declaration of the moratorium.
- Rapid asset survey (RAS).
- Resettlement action plan (RAP).
- Negotiations and development of agreement with affected parties.
- Physical construction.
- The partial resettlement of Kalana individuals / households is limited to the houses and associated infrastructure located within a 350 m buffer from the edge of the final Kalana Open Pit.
- A zone of influence around the non-mining related infrastructure, such as the plant, WRD, TSF and associated infrastructure and it was assumed that no activities would be allowed within 100 m of these facilities. The property and resources affected by this buffer zone were included in the asset survey and compensation assessment.
- A RAP was completed in 2015 and submitted with the ESIA, and approved by the Ministerial Committee. The RAP included a cost estimate associated with the resettlement of the affected community members as well as the compensation required for agricultural assets affected by the mining infrastructure and other economics activities to be affected by the proposed development. A phased approach to the implementation of the RAP is proposed.

The RAP will be updated as part of the Definitive Feasibility Study to incorporate the new design (increased project footprint and processing plant throughput).

20.5.2 Zone of Influence

As part of the 2016 ESIA, the Project conducted site-specific impact assessment studies in order to determine whether 350 m is an appropriate buffer for the Project (assessment done at a nominal production rate of 1.2 Mtpa competent fresh ore and 1.5 Mtpa soft saprolite ore):

- Air quality impact assessment.
- Noise impact assessment.
- Blast overpressure, vibration and fly rock assessment.
- Surface water assessment.
- Hydrogeology assessment.

Air quality dispersion modelling was undertaken for TSP, PM10, NO_2 and SO_2 as part of the impact assessment process. The predicted values were compared with ambient air quality guidelines. The concentrations are predicted to comply with the Environmental Protection Agency (EPA) guideline at all sensitive receptors outside of the 350 m buffer zone. Therefore, based on this assessment, it appears that the 350 m buffer zone is adequate in terms of air quality, provided that the proposed mitigation measures are implemented and the air quality monitoring programme confirms the outcome of the assessment.

The noise impact assessment confirmed that no additional parties would need to be resettled as a result of the noise impacts associated with the proposed development. Mitigation measures are required and a noise monitoring programme will be implemented.

The blast overpressure, vibration and fly rock assessment indicates that the blast overpressure and vibration criteria will be achieved within 250 m of the pit edge. Therefore, the use of a 350 m buffer zone provides an additional factor of safety criteria in achieving these criteria. Specific blast management measures will be incorporated into the mine plan of the Kalana Operation.

The impact on the surface water resources is expected to be limited and will not affect adversely affect the downstream users. This applies to the life of the Project as well as post closure.

The pit dewatering model indicates that the dewatering impact will be limited to specific community boreholes and these community pumps and boreholes will be replaced by Endeavour.

All modelling and simulations will be updated during the Definitive Feasibility Study to determine impacts and required mitigating strategies for the increase Project throughput (3.0 Mtpa) and footprint.

20.6 Environmental and Social Management Plan

In order to achieve appropriate environmental management standards and ensure that the findings of the environmental studies are implemented through practical measures, the recommendations from the ESIA will be used to compile an Environmental and Social Management Plan (ESMP). The role of the ESMP is to assist Endeavour in reducing potential impacts and risks and achieving its environmental objectives as well as fulfilling its commitment to the environment. The ESMP will be used to ensure compliance with environmental specifications, monitoring and management measures. The ESMP will be implemented from site preparation through to decommissioning and closure.

With the appropriate mitigation measures implemented, impacts associated with the Project can be mitigated to an acceptable level. Endeavour notes that social impacts relating to procurement of local goods and services and access and mobility, are considered to be positive impacts. Endeavour considers that the Project, with suggested mitigation measures, will enhance the socio-economic environment of the Kalana study area. The removal of the historic TSF facility is considered a positive impact.

20.7 Monitoring

Environmental monitoring is intended to provide constant feedback on the effectiveness of mitigation measures identified in and implemented through the various ESMPs. The monitoring plans will define the actions required for continuous tracking of these ESMPs, and will allow for the identification of any problems encountered, whilst providing the opportunity to adjust and mitigate ESMPs and monitoring plans accordingly. The various monitoring plans will be based directly on the ESMPs.

Endeavour will develop detailed procedures for each of these monitoring plans and will ensure that monitoring reports are prepared and reported internally, on a monthly basis, and externally, on an annual basis.

20.8 Decommissioning and Closure Plan

Endeavour's objective for the rehabilitation and closure of the planned mine is to ensure that the site is left in a condition that is safe and stable where long-term environmental impacts are minimised and any future liability to the community and future land use restrictions are minimised. The final post-mining land use will be determined in consultation with the DNACPN, stakeholders and local communities and is likely to include the following:

- Areas for agriculture.
- Areas livestock grazing.
- Wildlife habitat.

For reasons of health and safety, and in order to protect certain rehabilitation works from damage, portions of the Permit area may be designated as exclusion zones. Natural soil covers and vegetation will as far as possible be re-established over these areas but access by humans and/or livestock will be discouraged.

The closure cost estimation process and policy framework applied is discussed in Section 10. Closure of the existing underground mine and infrastructure will be carried out by Endeavour and is not included in the capital estimate for the Kalana Main Project.

20.9 Opportunities, Risks and Recommendations

The pit inflows and overall water balance for the Project was based on the available hydrogeological and climate information. Actual pit inflows as well as the efficiency of the out of pit dewatering holes and the ability of these holes to produce the projected volumes of clean water will be confirmed once the boreholes have been established and a sustainable yield tests completed on the boreholes. This may require the establishment of additional boreholes and a review and update of the water balance once the testwork has been completed.

The expected water quality and ultimate salt balance is based on a series of geochemical tests undertaken on the available material subjected to the short and long term testwork completed. Additional testwork will be undertaken, and a programme will be implemented to review and update the model based on results as they become available, together with actual water quality monitoring. Endeavour has provided budgets for this ongoing testwork.

A separate ESIA submission for the road realignment of the Yanfolila-Kalana road was requested by the DNACPN, this will be amended according to the new design.

The Project's revised RAP and ESIA will be submitted to DNACPN and for approval by the authorities. The implementation of the RAP will be a complex process requiring negotiation with individuals and communities. A consultant with experience in implementing resettlements in Mali will be appointed.

The 2016 ESIA was completed based on an engineered US$1,000/oz Au pit design and associated infrastructure. A review of the impacts associated with the revised 2020 pit design found that the Project impact will be materially different due to an increase in the WRD footprint, additional cultivated fields affected by the mining infrastructure, as well as the need for the diversion of the Kalanako stream (and its associated impact on the aquatic environment). The ESIA will have to be amended together with the CAPEX and OPEX costs associated with the environmental and social mitigation measures required. The Definitive Feasibility Study budget has made provision for this additional capital expenditure. Although the 2016 ESIA for the Kalana Project was approved and the environmental permit issued on 28 April 2016, allowing Endeavour / SOMIKA to commence construction and mining, this permit is no longer valid. A new environmental permit will be required to commence construction and mining. The application for a new environmental permit will be initiated when the ESIA amendment in finalised.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

21.1.1 Summary

The Project capital cost estimate (CCE) was compiled by Lycopodium with input from Knight Piésold on the tailings storage facility, water infrastructure, site access roads and airstrip. Endeavour have provided project specific portions of mine establishment and facilities.

The CCE reflects the Project scope as previously described in this study report and is summarised in Table 21.1-1.

Table 21.1-1 Capital Cost Estimate Summary (US$, 3Q20, +20/-10%)

Main Area	Capital (US$M)
Treatment Plant	**68.3**
Reagents & Plant Services	**16.1**
Infrastructure	**43.1**
Mining	**16.9**
Construction Distributables	**18.6**
Subtotal	**163.0**
Management Costs	21.7
Owners Project Costs	67.5
Working Capital	5.7
Subtotal	**257.9**
*Contingency	35.8
Definitive Feasibility Study	3.3
Project Total	**297.0**

*Contingency for Pre-strip are included in the estimated subtotals and have not been included in the stated contingency value.

All costs are expressed in US$ dollars ($) unless otherwise stated and are based on 3Q20 pricing. The estimate is deemed to have an accuracy of +20/-10%.

Where costs used in the estimate were provided in other than US dollars the following exchange rates have been used:

- 1 AUD = 0.7100 USD
- 1 EUR = 1.1833 USD
- 1 GBP = 1.3097 USD
- 1 ZAR = 0.061 USD

21.1.2 General Estimating Methodology

General arrangement drawings and a 3D model were produced with sufficient detail to permit the assessment of the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems and infrastructure.

The process plant was broken down into unit operation areas with quantity take-offs based on similar facilities from previous projects to provide an acceptable level of confidence required for a Pre-Feasibility Study (PFS) estimate.

Unit rates were established for bulk commodities, materials and labour that were drawn from previous studies within the region.

Capital equipment pricing was based on project specific budget quotation requests (BQRs) for major equipment and database pricing for the balance, drawn from similar projects currently under construction or recently completed.

The rates used in the estimate have been reviewed and deemed to reflect the current market conditions.

21.1.3 Estimate Basis

The capital cost estimate was prepared in accordance with Lycopodium's standard estimating procedures and practices. The basis and methodology are summarised in Table 21.1-2 and Table 21.1-3.

Table 21.1-2 Capital Cost Estimate Basis

Description	Basis
Site	
Geographical Location	Client advice from Site Plan
Maps and Surveys	Topo provided
Geotechnical Data	Not Available. Assumed competent
Process Definition	
Process Selection	PFS testwork
Design Criteria	Preliminary
Plant Capacity	Client advised
P&IDs	Not Required
Mass Balances	Preliminary
Equipment List	PFS
Process Facilities Design	
Equipment Selection	PFS
General Arrangement Drawings	Preliminary
3D model	Preliminary
Electrical Drawings	Single Line Diagram
Infrastructure Definition	
Existing Services	None
Power Supply	Overhead power line and grid connection cost estimated by ECG based on preliminary study.
Water	KP quantities & estimate
Accommodation	Preliminary – 100 man camp advised by client plus 80 man security barracks
Site Access and on-site roads	KP Quantities & estimate
TSF	KP quantities & estimate
Mine Services	Endeavour estimate
Security / Fencing	Estimated from layout
Design Basis	Preliminary
Layout	Preliminary

Table 21.1-3 Capital Cost Estimate Methodology

Description	Basis
Bulk Earthworks	Preliminary from Topo
Detailed excavation	Allowance for detailed based on footprint
Concrete Installation	Quantities estimated from the layout and completed like projects
Structural Steel	Quantities estimated from the layout and completed like projects
Platework & Small Tanks	Platework items as per the mechanical equipment list
Tankage Field Erect	Tanks as per the mechanical equipment list
Mechanical Equipment	Items as per the mechanical equipment list. BQR enquiries used for major mechanical items. Costs for all other items taken from the Lycopodium database of current or recently completed projects
Plant Piping General	Factored from mechanical costs where the factor was determined from completed project data
Overland Piping	Sized by engineering including MTO and specification
Electrical Costs	Estimate provided by ECG based on MEL and layout
Commodity Rates – General	Appropriate rates taken from the Lycopodium database
Installation Rates – General	Appropriate rates taken from the Lycopodium database
Cranage	250t Crane hired for the project major lifts
Freight General	Combination of freight tons and percentages
Contractor Mobilisation / Demobilisation	Estimated based on projects of a similar size
EPCM	Percentage based on the EPCM controlled scope
Owner's Costs	Estimate provided by Endeavour
Compensation / CSR Costs	Estimate provided by Endeavour

21.1.4 Pricing Basis

Pricing has been identified by the following cost elements, as applicable, for the development of each estimate item.

Plant Equipment

This component represents prefabricated, pre-assembled, commonly available mechanical equipment.

Bulk Materials

This component covers all other materials, normally purchased in bulk form, for installation on the project.

Installation

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities.

Temporary Construction Facilities

Facilities will be capable of servicing the Owners and EPCM teams.

Heavy Lift Cranage

The hire of a 250 t capacity crane for the duration of major lifts has been allowed for in the estimate.

Contractor Distributables

Costs for mobilisation / demobilisation of labour and equipment to / from the Project site were based on projects of a similar size and adjusted to suit the project location.

21.1.5 Qualifications

The estimate is subject to the following qualifications:

- Mining contractor mobilisation and establishment costs provided by Endeavour.
- Overhead power line and grid connection cost provided by ECG. Back-up power station size and cost by ECG.
- All labour rates, materials and equipment supply costs are 3Q20. Contingency has been allowed based on the quality of the various estimate inputs, however no allowance for escalation has been included.
- Construction contractor rates include mobile equipment, vehicles, cranage (up to 100 t), fuel, construction power and consumables for the duration of construction. Hire of a single heavy lift crane for a short duration for specific lifts has been allowed. Potable water and raw water supply will be provided by the client and available at site for the use by contractors.
- Site construction offices will be containerised units only, with the intention that the permanent administration building construction schedule will be accelerated.
- Mobilisation / demobilisation / R&R flights for construction contractor personnel are incorporated in the contractor distributable labour rates on the basis of individual contractors.

- There is no allowance for blasting in the bulk earthworks.
- Contractor accommodation costs per day have been included in the individual contractor's rates.
- Meals and accommodation for the EPCM team, Owners Team and Senior Contractor's staff has been included in the estimate.
- Project spares are included as a percentage allowance of the mechanical supply cost, based on similar size projects.
- Owners vehicles and mobile equipment are to be purchased early for use by the EPCM team during construction.
- A commissioning assistance crew is included in the EPCM allowance.

21.1.6 Contingency

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, design growth, etc., or the listed qualifications and exclusions.

Contingency has been applied to the estimate as a deterministic assessment by assessing the level of confidence in each of the defining inputs to the item cost being engineering, estimate basis and vendor or contractor information. It should be noted that contingency is not a function of the specified estimate accuracy and should be measured against the project total that includes contingency.

A contingency analysis has been applied to the estimate that considers scope definition, materials / equipment pricing and installation costs. Contingency applicable to various Owners inputs have been specified by Endeavour.

The resultant contingency for the Project is 12.2%.

21.1.7 Exclusions

The following items are excluded from the overall project capital cost estimate:

- Duties / taxes / fees.
- Permits and licences.
- Duties / taxes / fees.
- Project sunk costs.

- Exchange rate variations.
- Project escalation.

21.1.8 Escalation and Foreign Exchange

Escalation

Escalation has been excluded from the estimate.

Exchange Rates

All items in the capital cost estimate have been expressed in United States Dollars and no allowances for exchange rate variations are included in the estimate.

21.1.9 Preproduction Costs

Preproduction costs that include first fills, opening stocks, preproduction labour and vendor representative costs have been included in the estimate.

21.1.10 Working and Sustaining Capital

Working capital costs have been allowed to cover the first six weeks of plant operation (in Owners cost) and mining (included in pre-strip allowance) while full operating costs are being incurred with no revenue.

Knight Piésold has estimated ongoing quantities for the embankment raises on the TSF. These will be a combination of mine waste overhaul and earthworks contracting.

21.2 Operating Cost Estimate

21.2.1 Process Operating Cost Estimate

Introduction

The operating costs have been compiled by Lycopodium based on costs developed by:

- Lycopodium - Processing costs.
- Endeavour / Lycopodium – Site general and administration costs.

Operating costs for Mining are detailed separately in Section 4.

The estimate is considered to have an accuracy of ±25%, is presented in United States dollars (US$) and is based on information obtained during the fourth quarter of 2020 (4Q20).

Process Plant Operating Costs

Processing operating costs have been developed by Lycopodium for fresh, transition, and oxide ore and a life of mine (LOM) blend of 63% fresh, 9% transition, and 28% oxide material. It is expected that the plant will operate on a range of ore blends. The LOM processing costs are a weighted average of the various ore type processing costs based on the LOM blend.

Processing operating costs have been developed for a plant with an annual throughput equivalent to 3,000,000 tonnes of fresh ore plant feed at a P_{80} grind size of 90 µm, based on a 24 hour per day operation, 365 days per year. The plant will operate at the following availabilities:

- 80% (7,008 h/y) – primary crushing and ore handling.
- 91.3% (8,000 h/y) – milling and all subsequent downstream processing plant sections.

The operating costs have been compiled from a variety of sources, including the following:

- Labour pay rates and manning as advised by Endeavour.
- Grid power costs as advised by Endeavour.
- Consumable prices from supplier budget quotations, Endeavour advice, and the Lycopodium database.
- Modelling by OMC for crushing and grinding energy and consumables, based on physical ore characteristics determined from comminution testwork for the various ore types.
- Reagent consumptions and gold and silver extractions based on the results from a metallurgical testwork programme.
- First principle estimates based on typical operating data / standard industry practice.

The processing operating cost estimate is summarised in Table 1.11-1. The process plant operating cost estimate by year based on the plant feed schedule is presented in Table 21.2-2. The relative proportions of each operating cost centre for the processing and process plant (processing plus G&A) operating costs is shown in Figure 21.2.1.

The process operating costs have been split into fixed and variable components to enable them to be used to derive annual costs for changing plant feed blends and/or throughput over the life of the project. The fixed and variable costs are considered valid for throughput variations within 25% of the design plant feed throughput.

The operating costs includes all direct costs to allow production of gold bullion. The battery limits for the processing operating costs are as follows:

- Ore delivered to the ROM bin.
- Tailings discharged from the tails pipeline to the tailings storage facility (TSF).
- Gold bullion in plant goldroom safe.

Qualifications

The operating cost estimate presented in this section is exclusive of the following:

- All costs associated with areas beyond the battery limits of Lycopodium's scope of work.
- Any impact of foreign exchange rate fluctuations.
- Any escalation from the date of the estimate.
- Any contingency allowance.
- All withholding taxes and other taxes.
- First fill and opening stocks costs (included in capital cost estimate).
- Gold refining and bullion transport and in-transit security of gold from site.
- Tailings storage facility future lifts and site rehabilitation.

The operating cost estimate includes the following:

- Import duties on consumable unit costs (in the consumables cost).
- Costs for MSA power and raw, fire and potable water supply (in the power and consumables costs).
- Costs for the preparation and assaying of 100 mine grade control samples per day and routine laboratory tests on the site water samples (in the laboratory cost).
- Selected G&A costs (travel to site for international and regional expats, international expat recruiting / relocation and camp, catering and cleaning) for mine (as well as administration and process plant) personnel (in the G&A costs).

Figure 21.2.1 Operating Cost Breakdown

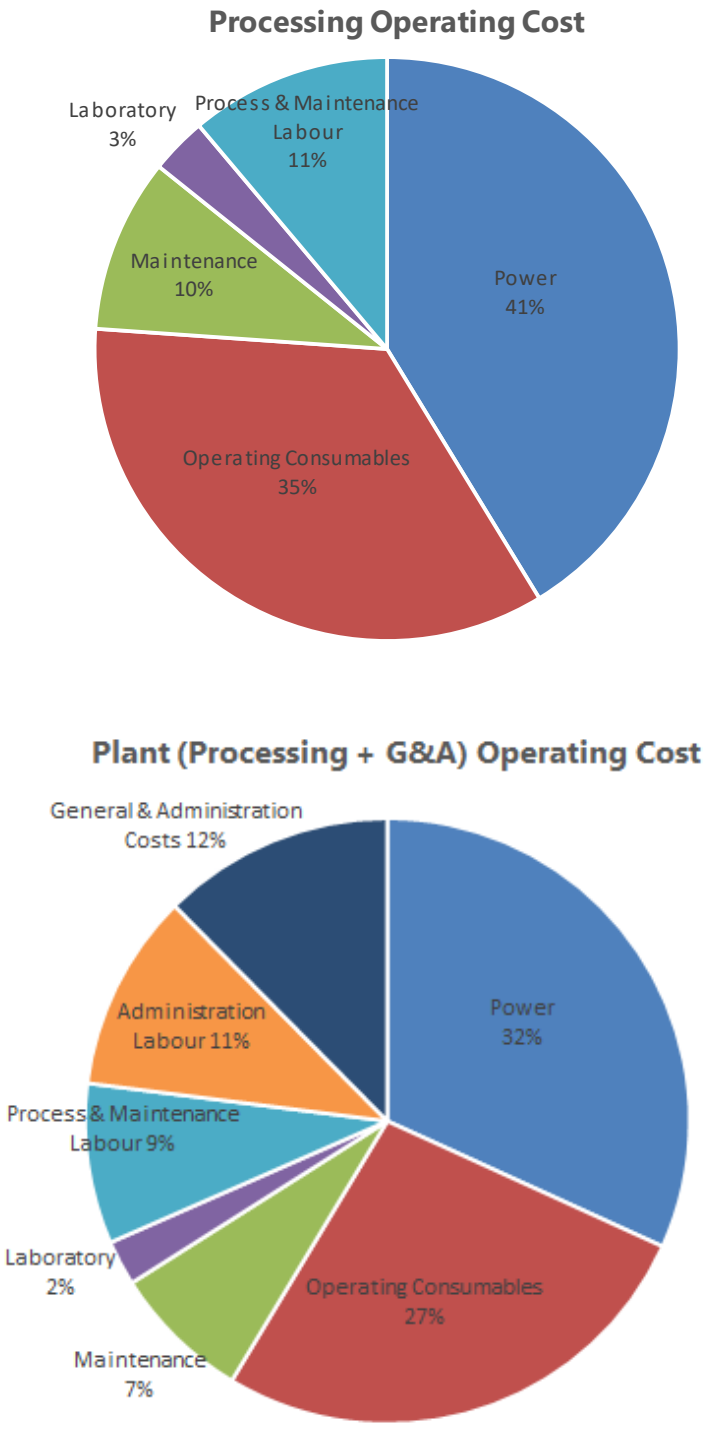


Table 21.2-1 Process Plant Operating Cost Summary (US$, 4Q20, ±25%)

	LOM Blend		Kalana Oxide		Kalana Transition		Kalana Primary		Kalanako Oxide	
Proportion of LOM	**100%**		**25.0%**		**9.1%**		**62.9%**		**3.0%**	
Plant Feed t/y	**3,233,100**		**3,800,000**		**3,100,000**		**3,000,000**		**3,800,000**	
Cost Centre	**US$/y**	**US$/t**	**US$/y**	**US$/t**	**US$/y**	**US$/t**	**US$/y**	**US$/t**	**US$/y**	**US$/t**
Power	17,744,016	5.49	13,289,347	3.50	17,698,983	5.71	19,133,743	6.38	13,289,347	3.50
Operating Consumables	14,965,302	4.63	14,111,618	3.71	16,098,312	5.19	15,076,941	5.03	13,013,152	3.42
Maintenance	4,144,389	1.28	4,027,976	1.06	4,063,782	1.31	4,212,975	1.40	3,920,971	1.03
Laboratory	1,342,572	0.42	1,342,572	0.35	1,342,572	0.43	1,342,572	0.45	1,342,572	0.35
Process Plant Labour	4,785,318	1.48	4,785,318	1.26	4,785,318	1.54	4,785,318	1.60	4,785,318	1.26
Total Processing	**42,976,879**	**13.29**	**37,556,831**	**9.88**	**43,988,967**	**14.19**	**44,551,549**	**14.85**	**36,346,641**	**9.57**
Fixed Cost US$/y	15,397,553		15,200,887		15,343,697		15,476,474		15,200,887	
Variable Cost US$/t		8.53		5.88		9.24		9.69		5.57
Administration Labour	5,905,813	1.83	5,905,813	1.55	5,905,813	1.91	5,905,813	1.97	5,905,813	1.55
General & Administration Costs	6,929,399	2.14	6,929,399	1.82	6,929,399	2.24	6,929,399	2.31	6,929,399	1.82
Subtotal G&A	**12,835,212**	**3.97**	**12,835,212**	**3.38**	**12,835,212**	**4.14**	**12,835,212**	**4.28**	**12,835,212**	**3.38**
Total Plant Including G&A	**55,816,808**	**17.26**	**50,392,042**	**13.26**	**56,824,178**	**18.33**	**57,386,760**	**19.13**	**49,186,571**	**12.94**
Fixed Cost US$/y	28,232,765		28,036,098		28,178,908		28,311,865		28,036,098	
Variable Cost US$/t		8.53		5.88		9.24		9.69		5.57

Exclusions: All Mining Costs except MSA power and water, mine grade control sample assaying and selected G&A costs for mining personnel.

Table 21.2-2 Process Plant Operating Cost Summary by Year (US$M, 4Q20, ±25%)

Plant Feed Schedule*	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	LOM	Prop'n
Oxide	2,761	2,543	1,948	1,294	314	111	1,540	5	0	5	14	10,535	29.6%
Transition	292	331	325	293	652	25	759	463	0	8	29	3,176	8.9%
Primary	71	832	1,284	1,800	2,229	2,896	1,213	2,610	3,000	2,990	2,960	21,886	61.5%
Total Ore Processed	**3,124**	**3,706**	**3,556**	**3,387**	**3,196**	**3,032**	**3,511**	**3,078**	**3,000**	**3,002**	**3,003**	**35,596**	100%
Annual Operating Costs (US$M)													
Power	12.8	15.7	16.6	17.5	18.8	19.0	17.1	19.2	19.2	19.1	19.1	194.1	
Operating Consumables	12.1	15.3	15.4	15.4	15.8	15.1	15.8	15.5	15.1	15.1	15.1	165.6	
Maintenance	4.1	4.2	4.2	4.2	4.2	4.1	4.2	4.1	4.1	4.1	4.1	45.7	
Laboratory	1.3	1.4	1.4	1.4	1.4	1.3	1.4	1.4	1.3	1.3	1.3	14.8	
Process & Maintenance Labour	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	52.6	
Total Processing*	**35.0**	**41.5**	**42.3**	**43.2**	**44.9**	**44.3**	**43.2**	**45.0**	**44.5**	**44.5**	**44.4**	**472.8**	
Administration Labour	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	65.0	
General & Administration Costs	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	76.2	
Total G&A	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**12.8**	**141.2**	
Total (US$M)*	**47.9**	**54.3**	**55.1**	**56.0**	**57.7**	**57.2**	**56.1**	**57.8**	**57.3**	**57.3**	**57.3**	**614.0**	

Exclusions: All Mining Costs except MSA power and water, mine grade control sample assaying and selected G&A costs for mining personnel.

*2020 Mine Schedule Version 1 A12

Power

The power cost estimate has been based on grid power at a unit cost of US$0.1442/kWh. The average continuous power draw and power cost for the LOM blend by plant area is summarised in Table 21.2-3.

The power cost the MSA is included in the processing operating cost. The power cost for the accommodation camp is included in the G&A operating cost.

Table 21.2-3 LOM Blend – Processing Power Cost Summary (US$, 4Q20, ±25%)

Plant Area	Installed Power (kW)	LOM Average Continuous Power Draw (kW)	LOM Power Cost (US$/yr)	LOM Power Cost (US$/t)
Primary Crushing	572	346	436,708	0.14
Stockpiling	55	40	50,022	0.02
Reclaim	118	71	89,491	0.03
Grinding Energy	12,000	7,821	9,879,473	3.06
Milling, Classification	1,876	733	925,743	0.29
Gravity & Intensive Cyanidation	726	340	429,688	0.13
Recycle Crushing	207	146	184,738	0.06
Pre-Leach Thickening	552	249	315,164	0.10
CIL	1,201	731	923,540	0.29
Elution & Gold Room	498	159	201,282	0.06
Tails Detox and Pumping	1,809	995	1,257,110	0.39
Reagents Mixing and Distribution	197	37	46,805	0.01
Water Services	842	345	435,904	0.13
Plant Services	54	16	19,996	0.01
Air Services	2,277	865	1,093,244	0.34
Fuel Storage & Distribution	3	0	152	0.00
Tailings Facility	321	166	209,613	0.06
Plant Buildings & Workshops	673	455	574,131	0.18
Accommodation Camp	861	384	485,602	0.15
Mine Services Area	362	147	185,611	0.06
Total	**25,202**	**14,047**	**17,744,016**	**5.49**
Fixed Component US$/y			6,497,968	
Variable Component US$/t				3.48

The power consumption for the SAG and ball mills has been calculated by OMC based on the physical ore properties determined from comminution testing. The power consumption for the remainder of the individual plant mechanical equipment items has been calculated from the installed power and typical drive efficiency and utilisation factors. Allowances for typical MSA and camp power requirements have been made.

Operating Consumables

Costs for processing operating consumables, including reagents, liners, fuels and process supplies have been estimated and are summarised for the various generic ore types and LOM blend by plant area in Table 21.2-4. Expected consumption rates for key consumables are summarised in Table 21.2-5.

Table 21.2-4 Consumables Cost Summary (US$, 4Q20, ±25%)

Plant Area	**Oxide**		**Transition**		**Primary**		**LOM Blend***	
Throughput	**3,800,000 t/y**		**3,100,000 t/y**		**3,000,000 t/y**		**3,233,100 t/y**	
% of LOM	**29.6%**		**8.9%**		**61.5%**		**100%**	
	US$/y	**US$/t**	**US$/y**	**US$/t**	**US$/y**	**US$/t**	**US$/y**	**US$/t**
Crushing	28,902	0.01	55,758	0.02	70,635	0.02	57,596	0.02
Milling	1,124,612	0.30	4,805,616	1.55	5,767,500	1.92	4,553,141	1.41
Leaching	5,506,862	1.45	5,500,662	1.77	4,006,526	1.34	4,517,886	1.40
Refining	1,568,430	0.41	1,567,907	0.51	1,567,714	0.52	1,567,920	0.48
Tailings	4,405,985	1.16	2,787,203	0.90	2,297,065	0.77	2,869,404	0.89
Water	8,094	0.00	8,094	0.00	8,094	0.00	8,094	0.00
Other	14,900	0.00	14,900	0.00	14,900	0.00	14,900	0.00
Diesel	1,453,833	0.38	1,358,172	0.44	1,344,507	0.45	1,376,362	0.43
Total	**14,111,168**	**3.71**	**16,098,312**	**5.19**	**15,076,941**	**5.03**	**14,965,302**	**4.63**

* Weighted average

Table 21.2-5 Expected Process Plant Consumption Rates

Consumables Usage	Units	Oxide	Transition	Primary	LOM Blend*
Jaw Crusher Liners	sets/y	2.34	2.80	3.50	3.11
Pebble Crusher Liners	sets/y	0.00	1.67	2.08	1.46
SAG Mill Liners	sets/y	0.18	0.87	0.99	0.75
Ball Mill Liners	sets/y	0.54	1.55	1.92	1.50
SAG Mill Media	t/y	38	1,085	1,275	976
Ball Mill Media	t/y	524	1,550	1,902	1,561
Sodium Cyanide	t/y	1,342	1,366	1,146	1,221
Quicklime	t/y	3,293	3,307	1,340	1,968
Carbon	t/y	147	140	139	142
Flocculant	t/y	95	78	75	81
Hydrochloric Acid	t/y	582	582	582	582
Sodium Hydroxide	t/y	930	930	930	930
Sodium Metabisulphite	t/y	1,330	1,085	1,050	1,132
Copper Sulphate	t/y	874	403	240	409
Ferrous Sulphate	t/y	1,064	868	840	905
Diesel	kL/y	1,888	1,764	1,746	1,787

* Weighted average

The consumables cost for mining is included in the mining operating cost.

The consumption of reagents and other consumables has been calculated from laboratory testwork and comminution circuit modelling at average ore properties, calculated from first principles, or has been assumed based on experience with other operations. No additional allowance for process upset conditions and wastage of reagents has been made.

Reagent costs have been sourced from a combination of budget quotations, Client supplied costs, and in-house data relating to similar projects in the region. Transport and freight to site and import duties and taxes have been added based on vendor information.

Cyanide destruction cost has been based on the INCO Air / SO_2 method, with the treatment of CIL tailings containing 150 g CN_{WAD}/m^3 down to 50 g $CN_{WAD}/m^{3\ \text{after cyanide destruction}}$. The cost of reagents for cyanide destruction and for the subsequent arsenic precipitation process have been derived from metallurgical testwork data.

A diesel price, delivered to site, of US$0.77 per litre has been used, as provided by the Client. Diesel will be used in plant mobile equipment, for the carbon elution heater circuit, and for the carbon regeneration kiln. The diesel consumption for plant mobile equipment is based on industry standard vehicle consumption rates and estimated equipment utilisation, while the diesel usage for carbon treatment has been calculated from equipment anticipated running times and vendor data.

Allowances have been made for water treatment reagents and operator supplies. Lubricants are excluded from the consumables costs and have been included under the maintenance cost centre.

Maintenance

The plant maintenance cost allowance has been factored from the capital supply cost using factors from the Lycopodium database and is summarised for the LOM Blend in Table 21.2-6.

The allowance covers mechanical spares and wear parts, but excludes crushing and grinding wear components, grinding media and general consumables which are allowed for in the consumables cost.

The maintenance cost excludes payroll maintenance labour which is included in the labour cost. Contract labour has been allowed for primary crusher and mill liner changes and plant shutdowns.

Allowances for plant mobile equipment, plant building maintenance and general maintenance expenses have been made.

The mobile equipment allowance is based on unit costs for maintenance of the light vehicles, portable generators and other mobile equipment for the process plant.

The plant infrastructure allowance includes maintenance on plant and MSA buildings.

General maintenance expenses include specialist maintenance software, maintenance manuals, training costs, vendor visits, and control system maintenance and licence fees.

Table 21.2-6 LOM Blend – Maintenance Cost Summary (US$, 4Q20, ±25%)

Plant Area	Oxide		Transition		Primary		LOM Blend*	
Throughput	3,800,000 t/y		3,100,000 t/y		3,000,000 t/y		3,233,100 t/y	
% of LOM	28.0%		9.1%		62.9%		100%	
	US$/y	US$/t	US$/y	US$/t	US$/y	US$/t	US$/y	US$/t
Plant Maintenance								
Crushing / Milling / Gravity	1,779,122	0.47	1,779,122	0.57	1,779,122	0.59	1,779,122	0.55
Thickening / Leaching / Elution	690,457	0.18	690,457	0.22	690,457	0.23	690,457	0.21
Tails Treatment & Pumping	235,115	0.06	128,110	0.04	128,110	0.04	154,861	0.05
Reagents	53,488	0.01	53,488	0.02	53,488	0.02	53,488	0.02
Plant Services	419,275	0.11	419,275	0.14	419,275	0.14	419,275	0.13
Water Supply	3,448	0.00	3,448	0.00	3,448	0.00	3,448	0.00
Plant Infrastructure	70,055	0.02	70,055	0.02	70,055	0.02	70,055	0.02
Mobile Equipment	328,530	0.09	328,530	0.11	328,530	0.11	328,530	0.10
Maintenance General	280,000	0.07	280,000	0.09	280,000	0.09	280,000	0.09
Contract Labour	168,486	0.04	311,296	0.10	460,491	0.15	365,153	0.11
Total Cost	**4,027,976**	**1.06**	**4,063,782**	**1.31**	**4,212,975**	**1.40**	**4,144,389**	**1.28**
Fixed Cost, US$/y	3,088,329		3,231,139		3,380,333		3,284,995	
Variable Cost, US$/t		0.25		0.27		0.28		0.27

* Weighted average

Labour

The labour rates, manning levels and rosters used to determine the labour operating cost estimate have been agreed with the Client and are based on benchmarking within Mali and other similar West African gold operations.

The manning numbers and labour costs for administration and plant operations and maintenance are summarised in Table 21.2-7 and Table 21.2-8.

Mining labour numbers, as shown in Table 21.2-7, have not been included in the estimation of selected G&A costs.

The plant labour cost includes all labour costs associated with site based administration, plant operations and maintenance personnel. The plant labour cost excludes all mining personnel (included in the Mining cost category) and all head office (included in the General Overheads costs).

The site laboratory will be operated on a contract basis with the personnel included in the process labour count but the labour costs included in the contract laboratory cost. Camp management / catering / cleaning, site security and the medical clinic will be operated on a contract basis with the personnel included in the administration labour count but the respective labour costs included in the camp contract cost, the security contract cost and the medical services contract cost.

Table 21.2-7 Manning Summary

Department	Number of Personnel				
	International Expatriate	Regional Expatriate	Local National	Contract	Total
Administration	7	10	159	261	437
Plant Operations	4	3	80	54	141
Plant Maintenance	3	3	52	0	58
Total Plant and Admin	**14**	**16**	**291**	**315**	**636**
Mine	12	40	72	395	519
Total Personnel	**26**	**56**	**363**	**710**	**1,155**

Table 21.2-8 Administration and Process Plant Labour Cost (US$, 4Q20, ±25%)

Department	Annual Labour Cost (US$)*			
	International Expatriate	Regional Expatriate	Local National	Annual Cost
Administration	1,627,480	1,191,287	3,087,046	5,905,812
Plant Operations	1,174,424	287,698	1,365,232	2,827,353
Plant Maintenance	808,960	309,749	785,335	1,904,044
Total	**3,610,864**	**1,788,733**	**5,291,533**	**10,691,130**

* Excludes Contract Labour Costs

The estimate of the labour contingent has been based on a three shift operation (two shifts working 12 hours per day, one rotation shift), to provide continuous coverage for the plant operation with allowance for leave and absenteeism coverage. Provision has been made in the manning numbers to accommodate annual and sick leave requirements.

The roster is based on all expatriate personnel working 6 weeks on site and 3 weeks off site with all other personnel working 10 days on site and 4 days off site with 4 weeks annual leave and two weeks sick leave per year.

Unit rates for labour have been based on Client advice and include the base salary and an overheads allowance. The overhead cost includes allowances for housing, travel, overtime and shift work, medical health insurance, life and disability cover, leave provisions and production bonuses. Camp and transportation costs for the workforce are excluded from the labour cost category and are included in the G&A costs.

Laboratory Costs

A contract laboratory will provide sample preparation and assay services for plant and environmental samples and 100 mine grade control samples per day.

Laboratory costs have been based on contract laboratories operating at other similar Client West African gold operations. The laboratory cost allows for the supply of the laboratory equipment, mobilisation and all ongoing costs (laboratory labour, equipment and consumables) comprising a fixed monthly cost and a variable cost related to the number of samples being processed. The laboratory building has been included as a separate item in the capital cost estimate.

The laboratory costs and the estimated samples are summarised in Table 21.2-9.

Table 21.2-9 Laboratory Cost Summary (US$, 4Q20, ±25%)

Plant Area		Samples / Month	Samples / Period		US$ / Month	US$ / Year
Fixed Fee					69,106	829,272
Variable Fee					42,775	513,302
Mine Exploration and Grade Control		3,042	100	per day		
Plant Solids (Assay, Moisture, Sizing)	Dry	30	1	per day		
	Slurry	243	8	per day		
Plant Solutions (Assay)		243	8	per day		
Plant Carbon (Assay)		122	4	per day		
Bullion (Au & Ag)		17	4	per week		
Environmental (CN_{WAD}, Total CN)		30	1	per day		
Environmental (pH, TSS, TDS, E.coli, Assay)		4	1	per week		
Total		**4,006**			**111,881**	**1,342,574**

Services and Utilities

Mobile Equipment

Plant mobile equipment requirements have been agreed with the Client. Mobile equipment costs provide for the fuel and maintenance of the mobile equipment fleet (excluding the mining fleet and mining light vehicles). The purchase cost of this equipment has been included in the capital cost estimate.

The fuel and maintenance costs for the mobile equipment are included in the consumables and maintenance cost centre, respectively.

An allowance has been made for a FEL to be routinely used in the plant to feed stockpiled ore into the mill feed bin when the primary crushing circuit is off-line, and to deliver SAG mill grinding media to the same mill feed bin, as required.

Water Supply

Water supply costs are based on operation and maintenance of the pit dewatering facilities to fulfil the process plant and infrastructural water requirements. Consequently water supply costs have not been estimated separately as they have been included in the other cost centres:

- Raw water supply pumping power is included in the power cost.
- Maintenance costs associated with raw water supply are included in the maintenance cost.
- Although no operations labour will be required for the raw water pumping facilities, security personnel will man the area with the labour cost of the security personnel included in the general overheads cost.

The consumables cost estimate includes allowances for elution and potable water treatment.

21.2.2 Site General and Administration

The general and administration costs have been estimated based on information provided by the Client and are summarised in Table 21.2-10. Administration labour costs are also shown in Table 21.2-10 for reference, although the labour costs and manning are included in the labour cost category.

Table 21.2-10 Site General and Administration Cost Summary (US$, 4Q20, ±25%)

Area	Annual Costs US$/y	LOM Cost US$/t
Site Office	523,200	0.16
Insurances	750,000	0.23
Financial	346,000	0.11
Government Charges	250,000	0.08
Personnel	1,338,000	0.41
Contracts	3,620,199	1.12
Community Relations	102,000	0.03
Total General & Administration	**6,929,399**	**2.14**
Administration Labour*	**5,910,090**	**1.83**
Total G&A, including Administration Labour	**12,839,488**	**3.97**

*Administration Labour included in Labour cost centre.

The general and administration expenses include the following ongoing operating expenses:

- Site office expenses including communications and communication maintenance, office equipment and supplies, computer supplies and software licenses.
- Insurance expenses covering industrial special risks, third party liability, motor vehicle and bullion transport. Labour associated insurances (medical, death and disability and workers liability insurances) are included in the labour costs.
- Financial expenses including banking charges, legal fees, auditing costs and accounting consultants and bullion selling. Bullion refining and royalties are excluded as they are included in the general overheads cost category.
- Government charges including permits and environmental inspection fees.
- Personnel expenses such as first aid and medical costs, safety supplies, travel and accommodation, expatriate travel, international expat recruiting / relocation costs, training, recreational and local facilities costs, professional memberships and subscriptions, and entertainment allowances. The allowances for expatriate travel (international and regional) and international expat recruiting / relocation include all personnel (mining as well as administration and process plant).
- Contract costs for personnel transport (in country charter flights and on site bussing), camp, catering and cleaning, environmental compliance testing, OH&S and other consultants. The camp, catering and cleaning contract cost includes all personnel (mining as well as administration and process plant).
- Community relations expenses including general expenses, community projects and scholarships.

21.2.3 Process Plant Pre-Production and Working Capital Costs

The costs incurred by operations during the latter stages of construction and commissioning are included in the capital cost estimate, but are derived in this estimate. The pre-production cost estimate for site administration and processing is summarised in Table 21.2-11. Pre-production costs associated with mining are excluded.

Table 21.2-11 Processing Pre-Production Cost Summary (US$, 4Q20, ±25%)

Description		US$
Mine		Excluded
Site Administration	Labour	1,375,895
	Expenses	1,598,058
	Subtotal	**2,973,953**
Process Plant	Labour	852,236
	First Fill	1,429,527
	Opening Stocks	1,618,809
	Vendor Representatives	429,500
	Training	170,000
	Working Capital	5,292,906
	Subtotal	**9,792,978**
Total		**12,766,931**

Pre-Production Labour

Pre-production site administration and processing labour costs reflect the need to recruit key operating personnel in time for them to set up and establish operating procedures and undergo training as required. It is envisaged that manning will commence to build-up eight months preceding plant start-up.

Pre-Production Site Administration Expenses

The pre-production site administration expenses cost covers the establishment of operations during the eight months preceding start-up and includes provision for power consumption, mobile equipment and other expenses during this period.

First Fill Reagents and Opening Stocks

Costs have been allowed to purchase the consumables and reagents required for the process plant first fill and opening stocks.

Sufficient first fill reagents and consumables have been estimated to fill the reagent tanks, charge the ball mill with media, fill the CIL circuit with carbon, and for other plant consumable requirements. Opening stocks refer to the purchase of the reagents and consumables required to sustain the operations for 6 weeks, which is the minimum on-site start-up storage quantity nominated by the Client.

Quantities allowed have been based on either consumption over the minimum period or minimum shipping quantities, considering package size.

Vendor Representatives

This cost allows for specialist vendor representatives to oversee commissioning of their processing equipment and include allowances for labour, airfares, and expenses.

Training

The training allowance covers the cost of providing pre-production training for process plant operations and maintenance staff, but not their salaries as these are covered in the pre-production labour costs.

Working Capital

Working capital covers the cost of operating the process plant before the first receipt of revenue from bullion sales. The working capital calculation is based on treating 100% oxide for the initial six weeks of operation at 80% of the design throughput rate.

21.3 Mining Operating Costs

The mining operating costs were estimated from first principals with equipment costs calculated based on hours derived from calculated productivities and haul profiles. Life of mine mining costs (including pre-strip) are shown in Figure 21.3.1.

Figure 21.3.1 Life of Mine Mining Costs by Activity

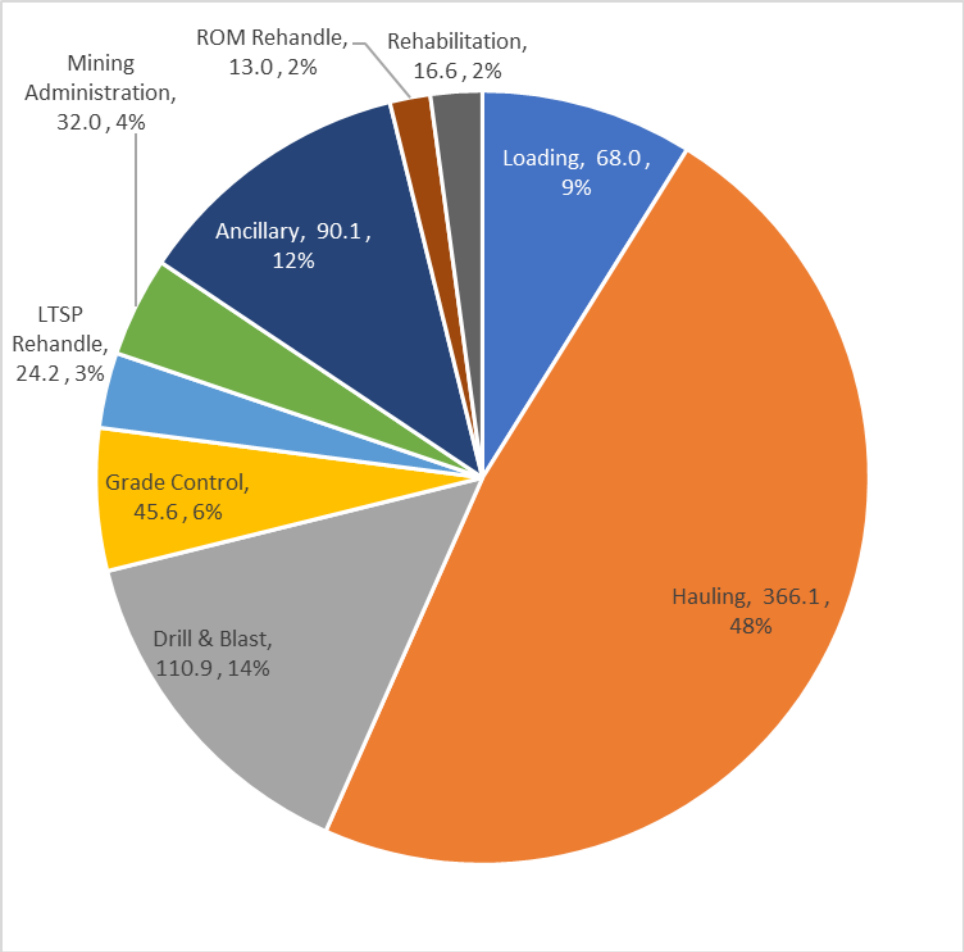


Table 21.3-1 summarises the annual mining operating expenditures by activity. The costs associated with hauling account for almost half the mining operating cost.

Table 21.3-1 Annual Mining Operating Expenditures by Activity

Activity	0	1	2	3	4	5	6	7	8	9	10	11	Total
Grade control	0.5	5.6	5.1	5.6	4.1	4.6	4.7	2.4	4.2	4.9	3.4	0.4	45.6
Drill and blast	0.0	2.7	4.5	8.1	7.7	16.5	18.5	6.2	17.7	19.4	8.6	1.0	110.9
Loading	0.9	6.7	7.2	7.3	7.3	7.3	7.0	6.9	6.9	7.0	3.1	0.4	68.0
Hauling	3.8	26.2	31.7	37.2	42.9	40.0	37.2	40.4	41.9	42.8	19.3	2.5	366.1
Ancillary	1.1	8.0	8.9	9.5	10.1	9.8	9.2	9.4	9.5	9.7	4.3	0.7	90.1
ROM rehandle	-	1.1	1.4	1.3	1.2	1.2	1.1	1.3	1.1	1.1	1.1	1.1	13.0
LT stockpile rehandle	-	1.3	-	-	1.1	0.1	0.5	8.0	-	-	1.6	11.6	24.2
Administration	0.5	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	0.3	32.0
Rehabilitation	0.2	1.7	1.8	1.8	1.8	1.8	1.6	1.8	1.6	1.6	0.7	0.1	16.6
Total	**7.1**	**56.4**	**63.7**	**73.9**	**79.4**	**84.2**	**82.9**	**79.6**	**86.1**	**89.6**	**45.3**	**18.1**	**766.2**
Unit cost ($/t ex-pit)	**1.90**	**1.98**	**2.08**	**2.47**	**2.58**	**2.88**	**3.10**	**2.62**	**3.14**	**3.44**	**3.99**	**13.28**	**2.78**

22.0 ECONOMIC ANALYSIS

22.1 Introduction

A financial model has been built to include the relevant study results in order to estimate and evaluate project cash flows and economic viability. The evaluation method takes into account mill feed tonnages and grades (including dilution) for the ore and the associated recoveries, gold price (revenue), operating costs, bullion transport and refining charges, government royalties and capital expenditures (both initial and sustaining). The project has been evaluated on a 100% ownership basis, with no debt financing.

The financial evaluation presents results for Net Average Value ('NAV') at a 5% discount rate and the Internal Rate of Return ('IRR'). The economic assessment has been prepared with the input from Snowden (mining and processing schedules and mining costs), Lycopodium (processing plant and surface infrastructure operating costs, capital costs) and Endeavour (site G&A costs and financial analysis).

22.2 Summary

The economic model shows robust results. Applying a long term gold price of $1,500/oz on a flat line basis from the commencement of production, the pre-tax NAV5% is $498 million and the IRR is 59%. The life of mine average cash cost per ounce is $785.

Project gold production is 1,655 koz over the life of the project using a metallurgical recovery of 90%.

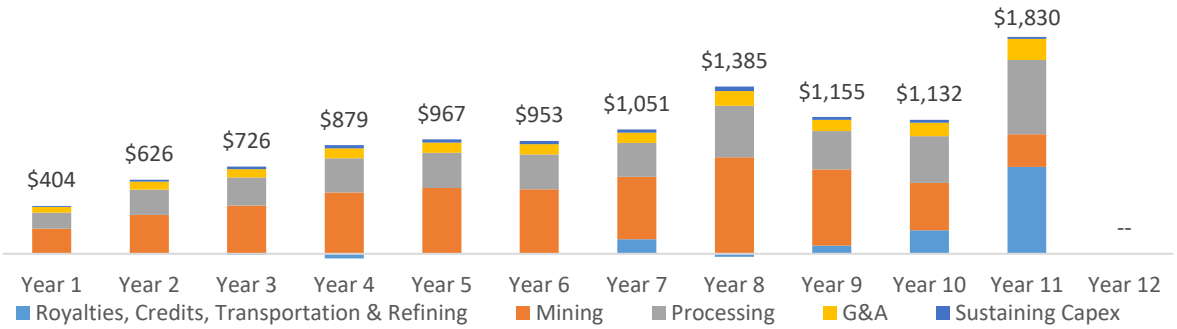
Table 22.2-1 presents a summary of the production information on which the cash flow model is based and the key project financial measures.

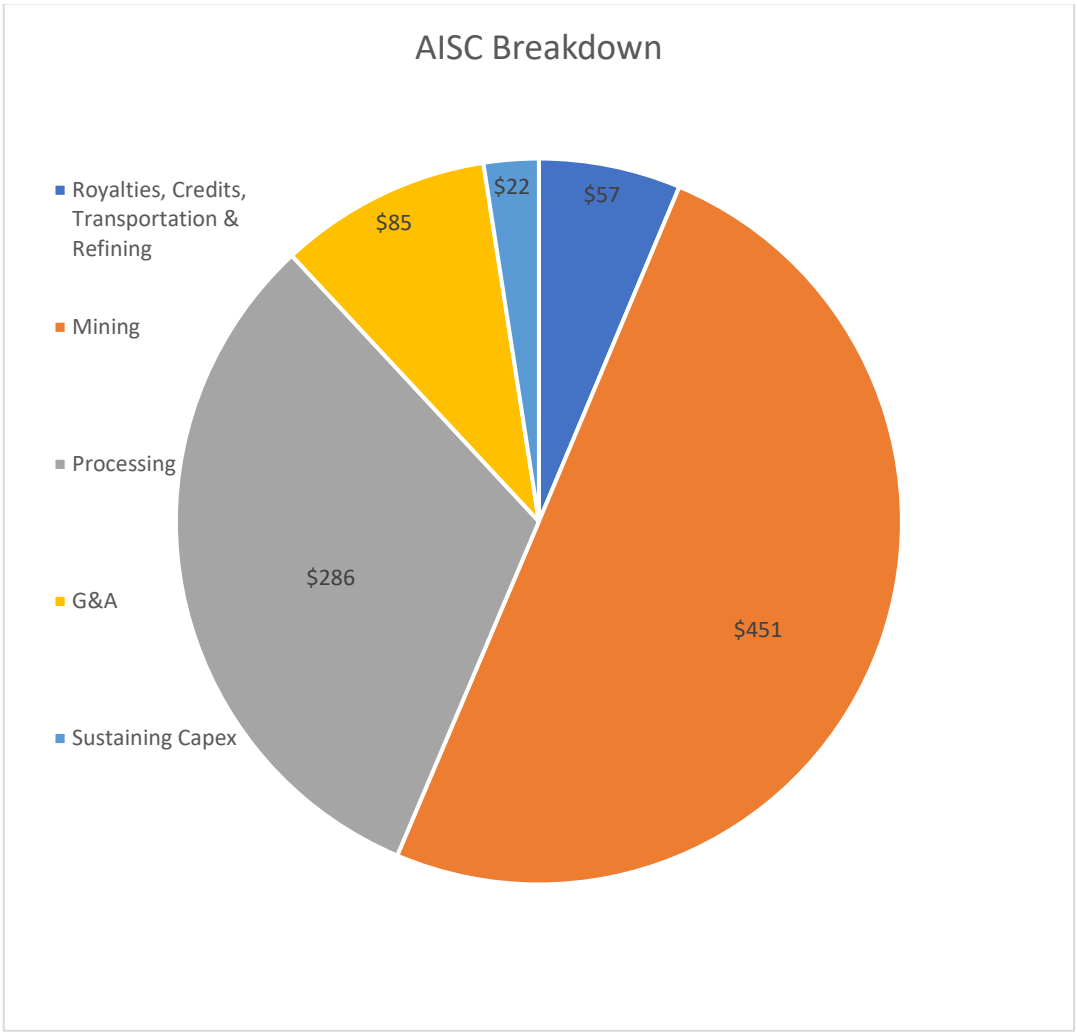
Figure 22.2.1 shows the All-In Sustaining Costs (ASIC) per ounce of gold sold by year and cost centre for the life of mine. The life of mine average all-in Sustaining Cost (ASIC) is $901/oz.

Table 22.2-1 Summary of Financial Analysis Results

Financial Summary	Units	Value
LOM tonnage ore processed	kt	35,595
LOM strip ratio	w:o	6.7
LOM feed grade processed	Au g/t	1.60
LOM gold recovery	%	90.5%
LOM gold production	koz	1,655
Upfront capital cost	$M	297
Pre-Tax:		
Internal rate of return	%	59%
NAV - 5% discount rate	$M	498
Post Tax:		
Internal rate of return	%	49%
NAV - 5% discount rate	$M	331

Figure 22.2.1 AISC Breakdown ($/oz sold)





22.3 Principal Assumptions and Inputs

The economic evaluation of the Project was based upon:

- Capital cost estimates prepared by Lycopodium, ECG, Knight Piésold and Endeavour.
- Mine schedule and mining operating cost estimates based on contract mining prepared by Snowden with inputs from Endeavour
- Process operating cost estimates prepared by Lycopodium.
- General and administration (G&A) cost estimates prepared by Lycopodium based on Endeavour input.
- Metallurgical performance characterised by testwork conducted on variability and composite samples from the Kalana deposits, as detailed in Section 5 of the report.
- Typical sustaining capital cost estimates for the infrastructure and process plant including Tailings Storage Facility lifts as advised by Knight Piésold.
- 3% Malian government royalties are based on the taxation code agreed between operating company SOMIKA and the state of Mali.
- The cash flow analysis excludes any effects due to inflation.
- A gold price of $1,500/oz.
- Construction completed over an 18 month period shown in the "Pre-production" period
- The financial assessment has been undertaken in United States Dollars (US$).

22.3.1 Basis of Estimate

- Annual tonnage, strip ratio and head grade have been based on the mining schedule as discussed in Section 4.
- The mining schedule and mining contractor operating cost estimate used as basis for the annualised production and cost model is presented in Section 4.
- The processing and administration costs are based on the operating cost estimate discussed in Section 12.
- The overall gold recovery figures utilise a LOM average 90.5% recovery based on the metallurgical testwork. Metallurgical recovery is discussed in Section 5.

- The capital cost estimate used as the basis for the cash flow model is discussed in Section 11. Project capital expenditure are in pre-production period.

22.3.2 Mining Contractor Costs

- Mining contractor costs are averaged for each material type moved per bench, including for drill and blast costs. It is noted that only approximately 30 to 40% of the material moved will require no or limited blasting (laterite and Saprolite material), and the associated costs are reduced accordingly.

- The anticipated contractor margin is included in the mining costs for the activities completed by the contractor. A benchmark comparison was done with other Endeavour sites using mining contractors or with recent tenders received, and the mining cost developed for the Kalana. project was very similar.

- Crusher feed costs and stockpile rehandle costs are averaged and applied to 100% of ore tonnes.

22.3.3 Sustaining Capital

A nominal allowance has been made for sustaining capital to include for:

- Tailings storage facility construction and ongoing embankment lifts.
- Allowances for road maintenance.
- Major shutdown maintenance and repair crew costs.
- Pit dewatering if required.
- Light vehicle replacement costs.
- Administrative capital budget.
- Provision for waste capitalisation

Provision for mining fleet replacement will not be required with contract mining operations.

22.3.4 Depreciation

Provision has been made for depreciation using a straight-line method for a period of five years for all capital.

22.3.5 Company Tax

Corporate tax rate of 35% of gross profit has been used which is in line with the taxation agreement between operating company SOMIKA and the state of Mali.

22.3.6 Refining Costs

- A typical refinery gold payable rate of 100% and an additional $3.00/oz freight plus insurance charge has been used based on historical costs.

22.3.7 Silver Credits

- No silver credits have been included in the financial model.

22.3.8 Royalties

- Gold production is subject to a government royalty which is based on a percentage of the spot gold price, a rate of 3% has been applied.

22.3.9 Working Capital

- The mining pre-strip is included in the capital estimate as it occurs prior to process plant commissioning. These tonnes are included in the mine schedule, however so for the purposes of the model, the pre-strip is deducted from the operating cost.

- Working capital, first fills and spare parts have been included in the upfront capital amount, with ongoing working capital movements estimated based on maintaining a rolling quarterly balance in line with historical levels observed at operations within the Endeavour Group.

22.3.10 Closure Costs

- In the economic model, demolition and closure activities occur after mining and processing activities have ceased, with an estimate of $29.5m provided by Lycopodium

22.3.11 Other

The cash flow model is based on 100% project ownership i.e. no allowance for minority shareholders and government free carry.

No provision has been made for interest or cost of capital.

No provision has been made for escalation or inflation.

Allowance has been made in the capital costs for Import Duties.

22.4 Cash Flow Analysis

The following table provides the details of the cash flows anticipated over the entire life of the mine.

Table 22.4-1 Summary of Cash Flow Analysis

Economic Analysis Summary		Total	Pre-prod.	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mine Total															
Total Material Mined	t	275,774,664	3,724,207	28,446,876	30,440,240	29,961,231	30,736,001	29,235,840	26,716,423	30,365,431	27,407,991	26,071,436	11,306,161	1,362,827	--
Waste	t	240,178,673	3,146,831	24,207,865	26,347,101	25,535,548	27,388,738	25,660,586	23,134,408	28,528,565	24,216,001	22,295,916	8,678,310	1,038,805	--
Ore	t	35,595,991	577,376	4,239,011	4,093,139	4,425,683	3,347,263	3,575,254	3,582,015	1,836,866	3,191,990	3,775,520	2,627,851	324,022	--
Grade	g/t	1.60	1.42	2.11	1.66	1.52	1.68	1.48	1.74	1.73	1.16	1.43	1.40	1.61	--
Contained Gold	**oz**	**1,828,838**	**26,430**	**287,818**	**219,019**	**215,865**	**180,434**	**169,880**	**200,410**	**102,056**	**119,214**	**173,100**	**117,868**	**16,743**	**--**
Strip Ratio	*Waste: Ore*	*6.75*	*5.45*	*5.71*	*6.44*	*5.77*	*8.18*	*7.18*	*6.46*	*15.53*	*7.59*	*5.91*	*3.30*	*3.21*	*0.00*
Processing Total															
Ore Processed	t	35,595,991	--	3,123,685	3,705,865	3,556,033	3,387,334	3,195,688	3,031,873	3,511,386	3,078,499	3,000,000	3,002,455	3,003,173	--
Ore Grade	g/t	1.60	--	2.75	1.76	1.70	1.53	1.64	1.73	1.47	1.18	1.60	1.32	0.83	--
Contained Gold	oz	1,828,838	--	275,855	209,874	194,822	166,647	168,672	169,116	165,455	116,548	154,599	127,069	80,183	--
Recovery	%	90%	0%	94%	92%	91%	90%	89%	89%	91%	89%	89%	89%	89%	0%
Recovered Gold	**oz**	**1,654,629**	**--**	**259,806**	**192,736**	**177,976**	**150,000**	**150,000**	**150,000**	**150,000**	**103,479**	**136,974**	**112,591**	**71,067**	**--**

Cashflow Summary		Total	Pre-prod.	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Net Revenues	**US$m**	**$2,482**	--	$390	$289	$267	$225	$225	$225	$225	$155	$205	$169	$107	--
Less: Cash Operating Costs															
Mining	US$m	($747)	--	($55)	($62)	($72)	($78)	($82)	($81)	($79)	($84)	($88)	($45)	($20)	--
Purchased ore	US$m	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Processing	US$m	($473)	--	($35)	($41)	($42)	($43)	($45)	($44)	($43)	($45)	($44)	($44)	($44)	--
General & Administrative	US$m	($141)	--	($13)	($13)	($13)	($13)	($13)	($13)	($13)	($13)	($13)	($13)	($13)	--
Other (royalties,inventory movement…etc)	US$m	($28)	--	($0)	($1)	$2	$18	$4	($0)	$24	$11	($10)	($22)	($52)	--
Operating EBITDA	**US$m**	**$1,094**	**--**	**$287**	**$172**	**$141**	**$109**	**$88**	**$86**	**$114**	**$24**	**$51**	**$44**	**($22)**	**--**
Less: Sustaining Capital	US$m	($103)	--	($2)	($3)	($4)	($16)	($9)	($4)	($46)	($12)	($3)	($3)	($1)	--
(Including Sustaining Exploration)	US$m	--	--	--	--	--	--	--	--	--	--	--	--	--	--
All-In-Sustaining Costs	US$m	($1,491)	--	($105)	($121)	($129)	($132)	($145)	($143)	($158)	($143)	($158)	($127)	($130)	--
Sustaining Margin	**US$m**	**$991**	**--**	**$285**	**$169**	**$138**	**$93**	**$80**	**$82**	**$67**	**$12**	**$47**	**$41**	**($23)**	**--**
Less: Non Sustaining Capital	US$m	($51)	--	--	($6)	($13)	($3)	--	--	--	--	--	--	($30)	--
(Including Closure Costs)	US$m	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mine Direct Cash Flow	**US$m**	**$939**	**--**	**$285**	**$163**	**$125**	**$90**	**$80**	**$82**	**$67**	**$12**	**$47**	**$41**	**($53)**	**--**
Less: Working Capital Movement	US$m	--	--	($4)	($5)	($10)	($14)	($9)	($10)	$19	($15)	($0)	$11	$52	($15)
Less: Taxes	US$m	($215)	--	--	($34)	($36)	($24)	($19)	($24)	($30)	($25)	($4)	($12)	($8)	--
Less: Lease Principal Repayment	US$m	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mine Free Cash Flow	**US$m**	**$724**	**--**	**$281**	**$124**	**$79**	**$52**	**$52**	**$48**	**$57**	**($29)**	**$44**	**$41**	**($10)**	**($15)**
Less: Growth Projects	US$m	($297)	($289)	($8)	--	--	--	--	--	--	--	--	--	--	--
Less: Other outflows	US$m	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Net Cash Flow	**US$m**	**$427**	**($289)**	**$273**	**$124**	**$79**	**$52**	**$52**	**$48**	**$57**	**($29)**	**$44**	**$41**	**($10)**	**($15)**
Project NAV (Post Tax)	**US$m**	**$331**													
Project NAV (Pre Tax)	**US$m**	**$498**													
Project IRR (Post Tax)	**%**	**49%**													
Project IRR (Pre Tax)	**%**	**59%**													

22.5 Sensitivity Analysis

22.5.1 Gold Price

Table 22.5-1 shows the impact on the pre-tax NPV @ 5% and pre-tax IRR of a change in the gold price

Table 22.5-1 Pre-tax NPV and Pre-Tax IRR Gold Price Sensitivity

Gold Price ($/oz)	NPV ($M)			IRR (%)
	0%	5%	10%	
$1,000	-158	-133	-123	N/A
$1,250	240	180	132	32%
$1,500	638	498	387	59%
$1,750	1036	808	642	83%
$2,000	1434	1121	897	106%

22.5.2 Operating Costs

Table 22.5-2 shows the impact on the pre-tax NPV @ 5% and pre-tax IRR of a change in the operating costs

Table 22.5-2 Pre-tax NPV and Pre-Tax IRR Operating Cost Sensitivity @ $1,500 Gold Price

Opex Change (%)	NPV ($M)			IRR (%)
	0%	5%	10%	
-15%	843	648	507	67%
-7.5%	740	571	447	63%
0%	638	498	387	59%
7.5%	535	417	327	55%
15%	433	340	267	50%

22.5.3 Capital Costs

Table 22.5-3 shows the impact on the pre-tax NPV @ 5% and pre-tax IRR of a change in all capital costs (initial, sustaining and non-sustaining capital).

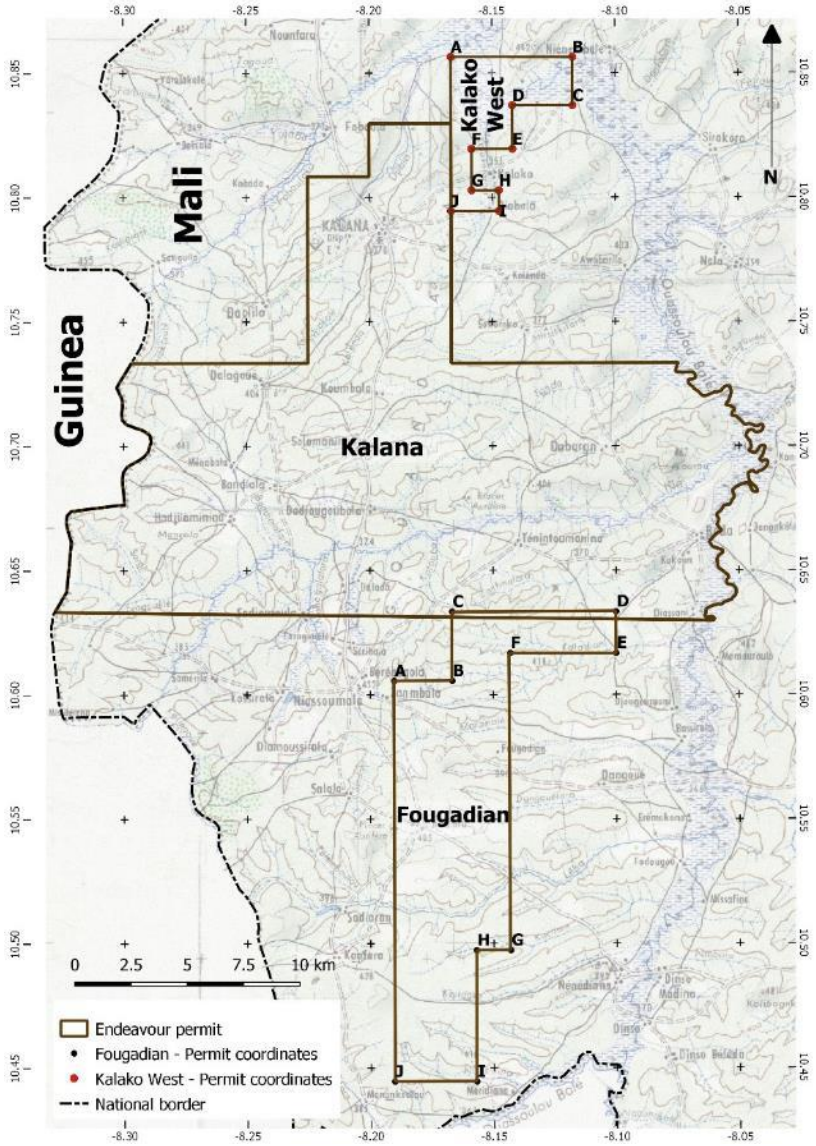
Table 22.5-3 Pre-tax NPV and Pre-Tax IRR Capital Cost Sensitivity @ $1,500 Gold Price

Capex Change (%)	NPV ($M)			IRR (%)
	0%	5%	10%	
-15%	691	546	437	76%
-7.5%	664	520	412	67%
0%	638	498	387	59%
7.5%	611	468	362	53%
15%	584	443	337	47%

23.0 ADJACENT PROPERTIES

Two exploration permits neighbouring Kalana were granted to Endeavour Mining in 2018; Fougadian located immediately to the south and Kalako West adjacent to the northwest, Figure 23.0.1.

Figure 23.0.1 Adjacent Endeavour Mining Exploration Permits



23.1 Fougadian

In May 2018, the Fougadian exploration permit covering 100 km² was granted to Endeavour Mining; no work has been performed to date.

Table 23.1-1 Fougadian Permit Coordinates

Point	Comment
A	Intersection between parallel 10°36'20''N and meridian 08°11'25''W. From point A to point B following parallel 10°36'20''N.
B	Intersection between parallel 10°36'20''N and meridian 08°10'00''W. From point B to point C following meridian 08°10'00''W.
C	Intersection between parallel 10°38'00''N and meridian 08°10'00''W. From point C to point D following parallel 10°38'00''N.
D	: Intersection between parallel 10°38'00''N and meridian 08°06'00''W. From point D to point E following meridian 08°06'00''W.
E	Intersection between parallel 10°37'00''N and meridian 08°06'00''W. From point E to point F following parallel 10°37'00''.
F	Intersection between parallel 10°37'00''N and meridian 08°08'35''W. From point F to point G following meridian 08°08'35''.
G	Intersection between parallel 10°29'50''N and meridian 08°08'35''W. From point G to point H following parallel 10°29'50''N.
H	Intersection between parallel 10°29'50''N and meridian 08°09'25''W. From point F to point G following meridian 08°09'25''W.
I	Intersection between parallel 10°26'40''N and meridian 08°09'25''W. From point I to point J following parallel 10°26'40''N.
J	Intersection between parallel 10°26'40''N and meridian 08°11'25''W. From point J to point A following meridian 08°11'25''W.

Fougadian was previously granted to Avnel Gold, who signed a Joint Venture (JV) agreement with IAMGOLD in December 2010. Before the JV, Avnel did soil geochemistry in 2007 and 5,000 m RC and 230 m of diamond drilling in 2008. Following the JV, IAMGOLD became the operator between 2009 and 2014.

During their period in Fougadian, IAMGOLD performed various works such as termite mound geochemistry and airborne geophysics (magnetic and radiometric). Following the reconnaissance work, IAMGOLD undertook several drilling campaigns, including AC and RC drilling on multiple targets for a total of 550 AC holes for 37,610 m and four RC holes totalling 532 m.

In 2014, IAMGOLD ended the JV and ceased activities on the permit which was subsequently released.

23.2 Kalako West

Kalako West encompasses 21 km² and was granted to Endeavour Mining in December 2018. Since its allocation, Endeavour has not performed any significant work except for some mapping.

Table 23.2-1 Kalako West Permit Coordinates

Point	Comment
A	Intersection between parallel 10°51'23''N and meridian 08°10'00''W. From point A to point B following parallel 10°51'23''N.
B	Intersection between parallel 10°51'23''N and meridian 08°07'02''W. From point B to point C following meridian 08°07'02''W.
C	Intersection between parallel 10°50'13''N and meridian 08°07'02''W. From point C to point D following parallel 10°50'13''N.
D	Intersection between parallel 10°50'13''N and meridian 08°08'30''W. From point D to point E following meridian 08°08'30''W.
E	Intersection between parallel 10°49'10''N and meridian 08°08'30''W. From point E to point F following parallel 10°49'10''N.
F	Intersection between parallel 10°49'10''N and meridian 08°09'30''W. From point F to point G following meridian 08°09'30''W.
G	Intersection between parallel 10°48'10''N and meridian 08°09'30''W. From point G to point H following parallel 10°48'10''N.
H	Intersection between parallel 10°48'10''N and meridian 08°08'50''W. From point F to point G following meridian 08°08'50''W.
I	Intersection between parallel 10°47'40''N and meridian 08°08'50''W. From point I to point J following parallel 10°47'40''N.
J	Intersection between parallel 10°47'40''N and meridian 08°10'00''W. From point J to point A following meridian 08°10'00''W.

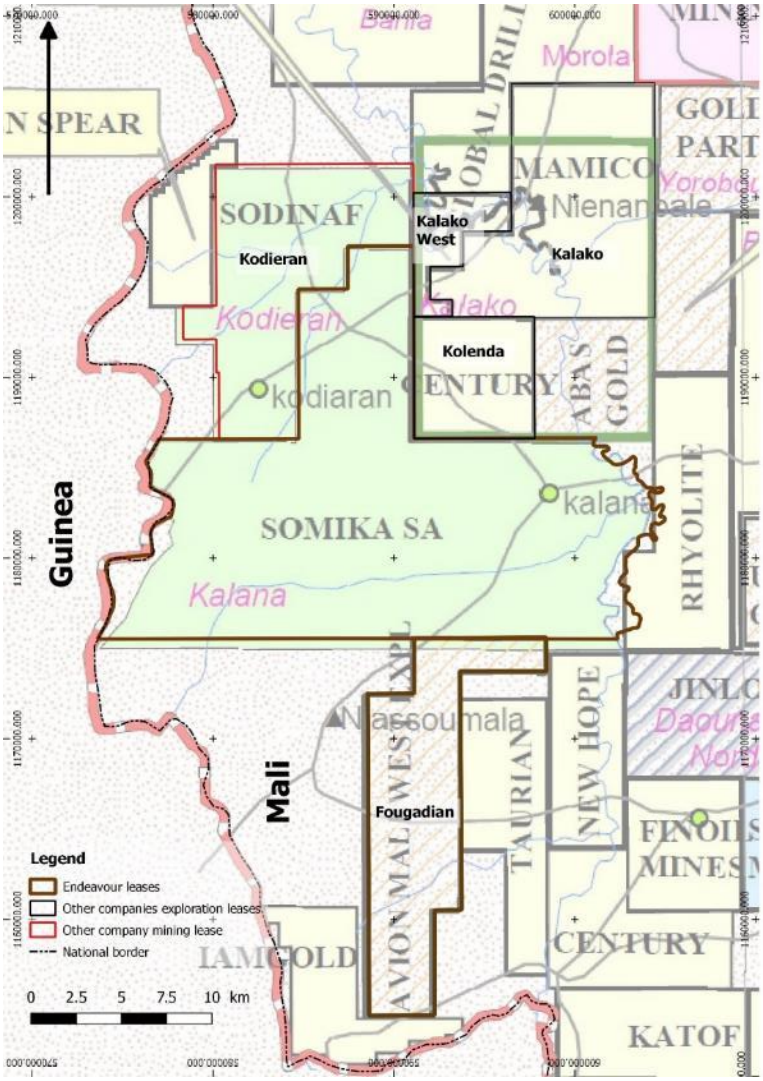
IAMGOLD held the permit between September 2011 and February 2013 and completed a termite mound geochemistry campaign and 45 RC holes totalling 5,560 m.

23.3 Other Permits

Exploration activity around Kalana is low. To the east of Kalana, the exploration permits of Kolenda and Kalako are currently held by Malian and Czech companies. These permits were explored by SONAREM-SOGEMORK in the 1970's and 1980's who defined a few targets with similar geology to the Kalana deposit. As part of their geological study of the area, the Russians identified various diorite bodies and took some grab sampling of quartz material (including one at 97 Au g/t) and drilled a few auger or diamond drillholes. There has not been any activity in the last five years.

The Kodieran Mining Lease, located north of Kalana, has an operating mine and is privately owned by a Malian company, Sodinaf/Wassoul'or, which was recently acquired by a Middle Eastern Fund. There are no publicly disclosed Mineral Resources and Mineral Reserves issued for the property. The Kodieran deposit is geologically similar to Kalana.

Figure 23.3.1 Other Adjacent Exploration Permits



24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Introduction

Risks are defined as the effect of uncertainty on objectives, with the deviation being either negative (a 'risk') or positive (an 'opportunity'). Objectives can have different aspects such as financial, health and safety, and environmental goals and can apply at different levels, such as strategic, organisation-wide, product, project and process. Risk is often characterised by reference to potential events, a consequence, or a combination of these and how they can affect the achievement of objectives. Risk is expressed in terms of a combination of the consequences of an event or a change in circumstances and the associated likelihood of occurrence.

At Pre-Feasibility Study level, risk assessment focuses on addressing those risks facing the project as a whole. The project risk review endeavours to identify major risks that could have a significant impact on the project, to allow these to be addressed early in the design.

24.2 Risks

24.2.1 Geology

Poor grade control, the impact of misallocation of ore and waste rock, was considered Significant. Poor grade control (sampling, analysis, interpretation, etc.), excessive mining dilution or ore loss, uncontrolled / not monitored blast movement, etc. resulting in ore being classed as waste and vice versa, will have a significant impact on the grade of the feed ore to the plant and gold production.

The actual grade of the ore feed varying by more than 10% over one year's production from the Resource model was identified as a risk. The coarse nature of the gold and geometry of the quartz veins constrains the level of accuracy in the gold grade estimate. It is considered that the grade estimate is a base case, conservative estimate. There is significant upside potential that the plant grade will be higher than the Mineral Resource and Grade Control estimates (the historical Mine Call Factor was approximately 130%).

24.2.2 Geotechnical

The hydrology of the orebody due to the saprolite being weak and having low permeability was identified as a risk.

The highly oxidised and weathered saprolite material is weak with low shear strength properties. This makes slopes developed in this material unstable, potentially leading to slope failures. In order to maintain stability, slopes developed in the saprolite material have been flattened. The slopes will also require active pit dewatering with either vertical wells or sub-horizontal holes to depressurise them.

The availability of poor-quality aggregate and sand locally and / or none being available was identified as a risk.

Suitable aggregate and sand for construction work is not available on site. Geotechnical / civil investigations will be conducted to identify sites within the project area from which to draw good quality aggregate and sand for construction purposes.

24.2.3 Mining

The issue of flyrock from the blasting going outside the exclusion zone was identified as a risk. The exclusion zone is 350 m, smaller than the Endeavour standard of 500 m. The 350 m was determined by SOMIKA to be safe with assessments covering air quality, noise, blast overpressure, vibration and flyrock, surface water and hydrogeology. Endeavour will develop a drill and blast procedure to ensure safety of the community. Other gold mines have operated safely with buffer zones of as little as 250 m.

The issue of high vibrations from the blasting causing damage to structures in the nearby Kalana town was identified as a risk. The 350 m exclusion zone was determined by SOMIKA to be safe with regard to blast overpressure and vibration. Endeavour will develop a drill and blast procedure to ensure safety of the community. Vibration and structural monitoring will be undertaken, if deemed necessary. Other gold mines have operated safely with buffer zones of as little as 250 m.

The potential for excessive dilution causing lower grade material to be fed through the mill was identified as a risk. As for each mining operation, and given the type of mineralisation of the Kalana deposit (gold bearing quartz vein with sometime relatively low thickness), the risk of higher than planned dilution exists. Endeavour will develop an appropriate grade control drilling program in order to have good definition of the limits between ore and waste. Endeavour also intends to implement a selective mining approach to mine this deposit, and to take advantage of the experience gained at the other operations managed by the company. Finally, the approach taken to develop the mineral resource estimate at Kalana is considered conservative, thereby further mitigating the risk of seeing lower grade material delivered to the process plant.

The possibility of the mining contractor underperforming by 15% or more was identified as a risk Given the fairly high mining rate needed at the Kalana operation, having good performance on the maintenance and operation of the mining equipment will be key. Endeavour has recent experience in mining contractor selection for several of its operations, and its technical team has developed good systems to work in conjunction with the mining contractors to assess the contractor's performance. On the contracting strategy, some incentives will be in place with the contractor to make sure expected performance is achieved.

24.2.4 Tailings Storage Facility

The effects of a seismic event on the TSF was identified as a risk. A seismic event greater than design parameters may result in a risk of dam failure but this is considered unlikely due to the downstream embankment configuration. In the event of dam (stability) failure, significant volumes of water and tailings solids could be mobilised. A liquor breach would flow downstream via the site water diversions and the topographical features within an existing water course adjacent to the site. It is probable that water and solids carried by the outflow would report to the Ouassoulou River (Bale River) some 14 km downstream of the TSF. It is noted that the Ouassoulou River is a tributary of the Niger River (the third longest river in Africa). In the event of failure of the TSF embankment resulting in release of solids, tailings would progress down the valley towards the Kalana Pit. It is anticipated that the Kalana Pit diversion bund and channel would be overtopped, resulting in tailings inundating the Kalana Pit base. The TSF will be designed to the correct stability design parameters (which include design earthquakes based on the population at risk) prescribed by ANCOLD. A warning system will be developed whereby the Kalana Pit will be evacuated in the event of dam failure (or warning signs are identified).

The consequences for a wall failure in the TSF were identified as a risk. This item covered failure modes including, but not necessarily limited to, piping failure, stability failure, or overtopping failure. A significant failure of the embankment would result in a release of tailings and water depending on the location of the breach, its size and the cause. In the event of dam failure, significant volumes of water and tailings solids are expected to be mobilised with consequences as described above for seismic failure. It is noted that the TSF will be constructed using downstream raise construction methods, which reduces risk of dam failure (based on historical information). A detailed geotechnical investigation will be completed during subsequent design phases to ensure stability parameters used for the TSF are appropriate.

24.2.5 Water

The unknown water quality from the pit was identified as a risk. Physicochemical analysis of pit water indicates high levels of As, Mn and Fe beyond allowable limits. The water is currently stagnant in the pit and leaching more than would normally be possible when abstraction matches inflow. It is expected that when dewatering is properly established, the level of contaminants will drop to acceptable levels. Additional test work will be undertaken to confirm this during the DFS. In addition, if contaminant levels are still high, samples will be sent for test work to ensure that they do not affect gold recoveries. If contaminants still exceed allowable levels for discharge to the environment, a treatment plant will be installed.

24.2.6 Pipelines

The leaking of the tails line to the TSF was identified as a risk. The tails line will be installed in a HDPE-lined trench that reports to either the site event pond or the TSF, mitigating the risk of tailings discharge to the environment.

24.2.7 Power

Prolongation of the negotiations with the authorities could result in a delay in constructing the HV power supply grid. The negotiations to tie into the Yanfolila substation must be completed in time for construction and commissioning.

The grid power reliability and security of supply were identified as risks. ECG have identified a technical solution to reduce the potential effects of power fluctuations in the supply grid.

24.2.8 Metallurgy

The contamination of the ore by arsenic, copper and other elements was identified as a risk. The design includes an arsenic precipitation circuit. The contamination risk can be further mitigated by having the geologist identify potential high areas for arsenic and then blend these ores with normal ROM to lower the arsenic grade being fed into the process plant.

24.2.9 Process Plant

The risk analysis identified six risks within the process plant; however, after the proposed new control measures were considered, the risks were all classified as low.

24.2.10 Environmental and Social

A delay in issuing the environmental impact assessment report was identified as a risk. The ESIA amendment will be undertaken in early 2021 and is forecast to take less than four months to complete. This is well within the current overall Project schedule.

The potential for spillage of hazardous materials outside the operations i.e. diesel or cyanide on route to the plant, was identified as a risk. Supply and transport contractors will be selected with comprehensive and compliant hazardous materials transport and handling procedures, including a documented and well-implemented incident response plan.

Dust from the operation was identified as a risk to the local community. Baseline dust monitoring is ongoing as part of the ESIA. Ongoing monitoring during operations will determine if the operation creates additional dust issues. The orientation of the plant and main areas of dust emission (crushing / stockpiling) is such that prevailing winds will not direct any dust emissions towards the populated zones. Haul roads will be watered more often during the dry season to reduce dust, including the rental of additional water trucks if required.

Noise from the operation was identified as a risk to the local community. Baseline noise monitoring is ongoing as part of the ESIA. Noise impact assessments confirm that no additional parties would need to be resettled as a result of noise impacts. Ongoing monitoring during operations will determine if allowable limits are exceeded and mitigating measures will be implemented.

Extreme rainfall was identified as a risk. Rainfall events greater than the design criteria for the various infrastructure items may result in water ingress into the Kalana Pit (stopping the mine operations or placing mine workers at risk) and possibly impacting the outskirts of Kalana town. The TSF stormwater storage capacity was confirmed by ensuring that volumes generated by storm events prescribed by ANCOLD based on the population at risk downstream of the TSF. Design of the site water management structures (pit diversion channels bunds) and haul road crossings were sized for agreed storm event parameters, with contingency freeboards applied as required. It is recommended that any further crossings installed across the river during operation be designed to ensure no impact to existing drainage systems. Trigger levels should be established at the pit diversion bunds for evacuation of the pit.

The implementation of the Resettlement Action Plan (RAP) was identified as a risk. The implementation plan will be integrated with the resettlement schedule to ensure that resettlement impacts on the critical path are well understood and adequately managed. International Finance Corporation Performance Standards will be complied with in accordance with Malian law. Malian authorities will be engaged throughout the process to support Endeavour in engaging the population to be resettled. A stakeholder engagement plan has been established and there is an established remediation committee to address concerns raised in timely manner. The RAP will include a grievance management procedure.

The potential of the RAP to overrun the budget was identified as a risk. The RAP estimate has been prepared by an established and experienced local consultant that has been involved in several resettlements. The numbers have been reviewed by Endeavour project personnel and an adequate level of contingency (15%) has been included.

The availability and cost of land for the RAP was identified as a risk. A suitable area has already been identified and initial consultation commenced. There have been no significant issues raised in acquiring this land. Malian authorities will be engaged throughout the process to support Endeavour's acquisition of the land.

The operation offering insufficient opportunities for employment and training of locals was identified as a risk. A local employment procedure will be implemented to ensure that priority for employment is given to project affected populations where the requisite skill is available. This will also be covered in the Stakeholder Engagement Plan.

24.2.11 Permitting

The approval of the Environmental Impact Study was identified as a risk. The ESIA will be prepared by a reputable consultant that is familiar with the local, national and international guidelines for an ESIA. The previous ESIA was approved and a permit granted (since expired). This application will simply be an amendment to the initial ESIA to cover changes due to the Project progression. Delays are unlikely as the Malian government is supportive of the Project and keen to see it progressed.

24.2.12 Country and Sovereign Risk

Changes in government policy due to a new government bring elected was identified as a risk. The current transition government is due to hold elections in 1Q22. It is expected that any elected government will be supportive of the project and the mining industry in general. Endeavour conducts regular, quarterly engagements with the government to ensure that they are updated on the project progress and remain supportive.

COVID-19 is an issue both during the construction and the operation. The ongoing presence of COVID-19 could have a detrimental effect on the construction duration, costs and ongoing operation of the mine. The Endeavour COVID Management Plan will cover the safe and efficient construction and operation of the mine.

24.2.13 Project Implementation

The planned production start-up could be delayed by any delay in resettling the population currently within the planned exclusion zone and this was identified as a risk. The implementation plan will be integrated with the resettlement schedule to ensure that resettlement impacts on the critical path are well understood and adequately managed. Local and national Malian authorities will be engaged throughout the process to support Endeavour in engaging the population to be resettled. A stakeholder engagement plan has been established and there is an established remediation committee to address concerns raised in timely manner. The RAP will include a grievance management procedure. The relocation affects one section of the pit and it is anticipated early commencement of the reallocation will eliminate all potential schedule delays.

24.3 Project Implementation Schedule

24.4 Key Construction Milestones

Key critical path activities / project risks are as shown in the following Table 24.4-1.

Table 24.4-1 Key Milestones

Item	Target Date	Duration from Award	
		Weeks	Months
Funding Approval	30-Nov-21	-	-
Proceed with Detailed Engineering & Drafting and Procurement	01-Dec-21	0	0
Award SAG & Ball Mill Supply	07-Dec-21	1	0
Start Bulk Earthworks	19-Apr-22	20	4.5
Start Civils Construction	21-Jun-22	29	6.5
Start SMP Installation	27-Oct-22	47	10.5
Start Electrical Installation	20-Dec-22	55	12.5
Start Plant Commissioning	26-Jul-23	86	20
First Gold Pour	31-Oct-23	100	23

24.5 Critical Path / Schedule Risk

Key critical path activities / project risks are listed below, and the main critical path components are shown below.

- Primary Critical Path:
  - Funding approval to proceed with design and procurement.
  - SAG and ball mill order, procurement and delivery.
  - SAG and ball mill installation.
  - Installation of associated milling area mechanical, piping and electrical works.
  - Subsequent commissioning through to gold pour.
- Secondary Critical Path:
  - Appointment of the bulk earthworks contractor.
  - Bulk earthworks sufficiently complete to provide access for mill foundation construction.
  - Construction of the mill foundations to be complete in time for the mill installations.

Additionally, the supply of permanent power for commissioning should be considered a risk.

24.6 Schedule Calendars

- Off-site activities such as engineering, design, drafting, procurement and manufacturing / fabrication are generally assigned to an 8-hour day x 5-work day week calendar.
- Transport / delivery activities are assigned to a 7-day week calendar.
- On-site construction activities are assigned to a 10-hour day x 6-workday week calendar.
- Some allowance has been made for public holidays and a non-work period between Christmas and New Year in the calendars.
- No specific allowance has been made for inclement weather.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Interpretation and Conclusions

The Kalana Gold Project PFS has identified, defined and costed a relatively low technical risk conventional open cut mine, and carbon-in-pulp processing facility. The flowsheet is based on unit operations that are proven in industry.

Commercial gold mining has been established in West Africa for over 100 years and Mali and surrounding countries have varying levels of established mining infrastructure.

Sovereign risk is understood with Endeavour having considerable experience in existing operations in other neighbouring West African countries.

Residual environmental risks are low and manageable, however the social issues aspirated with a large relocation of people from the local town will require careful management in future development stages.

Work undertaken to date is sufficient to support PFS level design and cost estimating. The Project is economically viable at current gold prices, with strong economic viability as detailed in Section 22.

25.2 Mineral Resource

The Kalana Gold Project is composed of the Kalana and Kalanako orogenic mesosthermal deposits and two Tailing Storage Facilities (TSFs) generated by historical underground mining with a gravity-only recovery plant.

The Mineral Resource models on which the Project is based is considered to be robust, with the key outcomes being:

- The Kalana geology is well understood and the mineralisation shows good continuity ~~over~~. The drill spacing of 25 m to 50 m over the majority of the resource area supports classification into the Indicated and Inferred Mineral Resource categories.
- The June 2020 Kalana Mineral Resource is reported above a grade cut-off of 0.50 Au g/t and inside a $1,500 optimised pit shell. Reporting within an optimised pit shell satisfies the requirement for the Mineral Resource to have "reasonable prospects for future economic extraction".
- Resource classification is based on the magnitude and standard of the exploration undertaken, sample density, the nature of the gold mineralisation and experience of the QPs. The drilling techniques, survey, sampling, sample preparation, analytical techniques and database management / validation exceed industry standards. The database represents an accurate record of the drilling undertaken in the Project.

- Geological continuity is understood with high confidence. The classification of the Mineral Resource reflects the level of geological confidence and the effect of the coarse gold.
- The estimation technique and parameters used are appropriate for the style of mineralisation.
- It is the Qualified Person's opinion that the Kalana Gold Project resource documented in this Report meets NI 43-101 standards and JORC Code guidelines.

25.3 Mineral Reserve

The Mineral Reserve Estimate on which the Project is based is considered to be robust, with the key outcomes being:

- The Probable Mineral Reserve is 34.8 Mt at 1.60 Au g/t. The strip ratio is 6.9:1.
- Annual gold production for the first three years of production is above 150 koz/a. This is followed by four years at 150 koz/a, with the remaining years below this level.
- Twelve year mining life.

25.4 Mineral Processing

The metallurgical and ore processing studies have demonstrated the Project to be viable and attractive for development. The key outcomes of the Project are as follows:

- Whilst testwork has not yet been completed on ore from the expanded resource at the bottom of the pit, it is expected that the metallurgical and comminution characteristics of this material will be similar to the Kalana ore already tested.
- The flowsheet is based on unit operations that are well proven in industry.

High recoveries are achievable from all major ore types that are planned to be processed.

25.5 Environmental and CSR

Endeavour considers the Project is being undertaken with due consideration of biophysical, social and economic factors, as well as the relevant Mali legislative requirements, EPs and IFC Performance Standards. The economic benefits of such a development are numerous, however, as in any mining project of this nature, there are also negative impacts which will require planning, monitoring and mitigation during construction, operation and decommissioning phases. None of the impacts identified are considered as fatal flaws and as indicated, high significant impacts, after implemented mitigation measures, if implemented, will be of low to medium significance.

26.0 RECOMMENDATIONS

A number of recommendations have been identified by Endeavour and their consultants, which will be progressively investigated and potentially implemented as the Project progresses.

26.1 Mineral Resources and Geology

The Mineral Resource Estimate demonstrates the economic potential of the Kalana and Kalanako deposits. The following points summarise key recommendations arising from the Project.

- Resource Definition Drilling:

  The Kalana deposit 3rd order veins and stockwork mineralisation have not been wireframed / modelled as separate, distinct mineralisation styles but as 'blow-outs' to the 1st and 2nd order veins. Endeavour recommends that the 3rd order veins are modelled separately in any future work, which will require additional diamond core drilling at variable orientations.

  For Kalanako, it is recommended to undertake some close spaced drilling within a 'typical' region of the deposit to improve the definition of grade continuity and to improve confidence in modelling the continuity of grade. Any such drilling should be based on section lines that are perpendicular to the overall deposit strike and separated by no more than 20 m.

- Additional Deposit Exploration:

  Exploration and resource delineation drilling should be continued to the southwest of Kalanako, in the Kalanako NorthExtension area, Bandiala, Tonda, etc. where interesting gold showings and historical artisanal workings have been identified in conjunction with soil and geophysical anomalies.

  The Djirila deposit is within a reasonable trucking distance (22 km southwest) from the Kalana Plant and should be reviewed in regard to identifying possible additional mineral resources.

- Regional Exploration:

  Exploration should continue on the Kalana Project permits. Discovery of additional deposits may provide an extension of the Kalana Mine operation. In addition, potential high-grade free-digging soft material (i.e. saprolite) may provide flexibility in the overall Kalana operation, should the price of gold fall or OPEX increase, for example. The Kalana Project permits have the potential to develop a strong pipeline of exploration prospects.

26.2 Environmental and Social

The pit inflows and overall water balance for the Project was based on the available hydrogeological and climate information. Actual pit inflows as well as the efficiency of the out of pit dewatering holes and the ability of these holes to produce the projected volumes of clean water will be confirmed once the boreholes have been established and a sustainable yield tests completed on the boreholes. This may require the establishment of additional boreholes and a review and update of the water balance once the testwork has been completed.

The expected water quality and ultimate salt balance is based on a series of geochemical tests undertaken on the available material subjected to the short and long term testwork completed. It is recommended that additional testwork be undertaken, and that a program be implemented to review and update the model based on results as they become available, together with actual water quality monitoring.

Measures recommended for minimising and managing the current and LoM rehabilitation and closure liabilities include:

- Minimising the footprint of mining and related activities.
- Confirmation and monitoring of the geochemistry of the tailings and waste rock.
- Confirmation of the geohydrology of the mined areas and also the likely quality and direction of flow of water in the workings subsequent to closure.
- Confirming that collection and treatment of contaminated water originating from the open pit or residue deposits will not be required.
- Ensuring that the necessary environmental monitoring data is collected in order to enable assessment of the extent of rehabilitation works required and the design of those works.

The Project impacts on three cemeteries; local communities recommended their relocation. To do this, an authorisation from the Ministry of the Environment is required after consultation with three other ministries, namely Health, Territorial Administration and Town Planning. The application file for this authorisation must contain documentation of the process that resulted in the agreement between the community and the company for the relocation of cemeteries.

The realisation of the Project requires the relocation of part of Kalana town. This partial resettlement is the fundamental social stake in financial terms and in terms of social risk. The Project will generate both physical and economic (agricultural) displacement.

The loss of cultivated land constitutes a major social risk in the context of the land crisis in the area. The loss of land and fruit trees caused by the project could disrupt the production systems in the area if we take into account the moratorium which means that since 2015, no field owner could plant trees in his field.

An update of the entire Resettlement Action Plan must be completed during the DFS Phase, and prior to make a full investment decision on the Project. This should include a detailed analysis of a staged approach (three phases) of the resettlement, to reduce the upfront capital cost.

26.3 Pit Geotechnical

Groundwater management during the saprolite mining must be managed effectively to ensure wall stability. A ground control management plan should be prepared prior to commencing mining operations, with provision and installation of basic slope stability monitoring instrumentation networks. A geotechnical design review should be conducted once new data is available from the working areas. Snowden also recommends a void management plan to control the hazards posed by mining through old workings.

26.4 Mining

The Kalana Mineral Reserve Estimate has demonstrated economic viability under the economic assumptions outlined. Adherence to a number of the key inputs and assumptions underpinning this economic viability is pivotal to realising the expected Project cash flows.

If the mine schedule and stockpiling strategy are followed accordingly, it will ensure that tonnes mined, ore delivered and mining costs are within the estimated parameters. To implement this accurately, Snowden recommends detailed short-term planning be completed as soon as practically possible following a Project development decision.

Designing and implementing best practise ore control procedures in the Kalana open pit will ensure that the grade of ore delivered to the mill is within the estimated ranges. Snowden recommends ore control personnel be appointed who are suitably trained and qualified. This task should be initially undertaken by geologists rather than general 'ore spotters' to ensure effective management during the critical early phases of development is adequately managed.

A system of mine to mill reconciliation should be developed prior to production that will adequately capture and store information.

In addition, Snowden recommends that suitably qualified and experienced open pit blasting personnel be engaged to ensure the highest possible standards of blasting are adhered to in order to limit both dilution and potential community impacts.

Potential exists to improve the haulage productivities by introducing a 'hot-seat-change' for the operators, utilising an additional 1.5 hours of operating time. Further potential exists to optimise the ramp widths, allowing a slightly reduced stripping ratio. Snowden recommends both points be investigated.

There is a potential northern extension of the Kalana open pit. A larger open pit will impact on the Kalanako stream and a major diversion will be required. The volume of waste mined will potentially increase and additional space for the waste dumps will need to be identified. Exploration drilling should be undertaken to test the potential mineralisation to the north of the Kalana open pit.

26.5 Mineral Processing and Metallurgy

In October 2017, a series of drill core samples were identified and selected for subsequent comminution and metallurgical testing as part of the 2018 Kalana Gold Project Definitive Feasibility Study (DFS) in support of a 1.5 Mtpa plant throughput of primary ore. The results of this testwork campaign formed the basis for the development of the design criteria for the 2018 DFS. These same series of criteria were also used for the subsequent 2020 Pre-Feasibility Study (PFS), which considered a revised 3.0 Mtpa plant throughput of primary ore and resulted in a preliminary pit design for the PFS of ~320 m – the DFS testwork campaign only used drill core samples retrieved from a depth of ~200 m. Despite geological opinion that it would be highly unlikely that any new and anomalous geological ore types would have been introduced as a result of the deeper pit, it is recommended that a comminution and metallurgical infill testwork programme be included in a subsequent DFS to confirm the metallurgical response and/or physical characteristics of the primary ore at these deeper levels (i.e. between ~200 m and ~320 m).

26.6 Process Plant

26.6.1 Site Geotechnical Investigations

No detailed geotechnical information was available for the PFS plant site and infrastructure design. A detailed site geotechnical program should be conducted with results available for commencement of detailed design.

26.6.2 Sterilisation Drilling

Site sterilisation drilling for process plant and infrastructure should be completed as soon as practicable to reduce the risk of siting plant and infrastructure on potential mineralisation.

26.6.3 Engineering

As discussed above, testwork is at PFS level only and this feeds into the design criteria and mass balance. Following completion of the more comprehensive testwork program described above, the flowsheet and mass balance will be reviewed. This may lead to revised flowsheets and mechanical equipment selection. In addition, further work on infrastructure design will be completed.

26.7 Groundwater Investigations

A desktop level assessment was undertaken, the hydrogeological concept revised and a numerical groundwater model constructed to estimate potential dewatering requirements. This work was based on data available from local wells as well as groundwater recovery data collected when dewatering operations from previous operations were halted. A model derived pit inflow rate in the order of 600 m^3/hour is not seen to be significant enough to impact mining operations. This assumption should be confirmed however in a detailed hydrogeological study.

A groundwater investigation should be conducted for the Kalanako Pit area and be incorporated into the hydrogeological study for the Project.

26.8 Infrastructure

26.8.1 Roads

The Malian government requirements for the Kalana Bypass Road should be agreed prior to commencement of the next design phase, with a view to confirming the design parameters.

26.8.2 Tailings Storage Facility (TSF)

Findings from the geotechnical investigation will be incorporated into the seepage and stability assessments for the TSF Dam Breach Assessment**.**

A population census should be completed for settlements in proximity to the project site (Kalana and Daraolia) to determine the population and areal extents of the settlements, to inform the dam breach assessment and ANCOLD consequence classification of the TSF.

26.9 Implementation

The existing schedule will be progressively updated based on updated information on environmental and mining approvals and lead times for equipment. The implementation approach will be refined to take account of market conditions and appropriate timing for the design and construction of the facilities.

27.0 REFERENCES AND OTHER INFORMATION

27.1 References

The following references were used in the preparation of this report:

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7. Epoch Resources (Pty) Ltd., Project Ref. 192-001, "Environmental and Social Impact Assessment Report for the Kalana Gold Project", March 2016.

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28.0 QUALIFIED PERSONS

Lycopodium

CERTIFICATE OF QUALIFIED PERSON

I, David John Gordon, as an author of this report entitled "NI 43-101 Technical Report – Pre-Feasibility Study of the Kalana Gold Project, Republic of Mali", which is effective as of December 31m 2020, and issued on April 1, 2021 (the "Technical Report") prepared for Endeavour Mining (the "Company"), do hereby certify that:

1) I am Group Manager - Process with Lycopodium Minerals Pty Ltd. My office address is Level 5, 1 Adelaide Terrace, East Perth, Western Australia 6004.

2) I am a graduate of the Western Australian Institute of Technology with a B. App. Sc. (Bachelor of Applied Science) degree in Engineering Metallurgy, 1983.

3) I am a Member of Australian Institute of Mining and Metallurgy, membership number 108413, and registered as a Fellow with the Institute. I have worked as a metallurgist, operations manager, and process engineer/manager for a total of thirty seven years since my graduation. My relevant experience for the purpose of the Technical Report is;

   - managed and interpreted results from numerous leaching and mineral processing testwork programs on gold ores
   - involved in the process design of treatment plants for over 18 years

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Kalana Project site, however a representative of Lycopodium did visit the site in 2017.

6) I am responsible for all of preparation of Sections 13, 17 18.6 to 18.15, 18.21, 21.1 and 21.2, and have contributed to Sections 1, 24, 25 and 26 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report on the Kalana Deposit.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this

1st day of April 2021

"Signed and Sealed"

David John Gordon, FAusIMM (108413)

CERTIFICATE OF AUTHOR

To accompany the Report entitled “NI 43-101 Technical Report – Pre-Feasibility Study of the Kalana Gold Project, Republic of Mali” which is effective as of December 31, 2020 and issued on April 1, 2021 (the “Technical Report”) prepared for Endeavour Mining (the “Company”).

I, *David Morgan,* do hereby certify:

1. I am the Managing Director of Knight Piesold Pty Limited, with an office at Level 1, 184 Adelaide Terrace, East Perth, WA 6004.
2. I am a graduate of University of Manchester, (BSc, Civil Engineering, 1980), and the University of Southampton (MSc, Irrigation Engineering, 1981).
3. I am a registered member of AusIMM (Australasian Institute of Mining and Metallurgy), membership #202216 and a chartered Professional Engineer and member of the Institution of Engineers Australia (Australia, 974219).
4. I have worked as a Project Manager, Senior Mining Engineer or Project Director continuously since my graduation from university. I have been continuously employed since my graduation in 1981. I have gained relevant experience on projects similar to the Lafique Deposit, including:
    a) Work on gold mining operations and projects located throughout West Africa.
    b) I have supervised and contributed to many tailings management engineering studies for different projects at various stages of development. Hands-on experience for gold in Ivory Coast, Mali, Burkina Faso, Australia and Canada;
    c) Design, supervision and implementation of mining programs;
    d) Review, audits of tailings management systems;
    e) Experience on several projects in tropical conditions (West Africa, Indonesia); and
    f) Participation in the preparation of parts of NI 43-101 compliant Technical Reports.
5. I have read the definition of “qualified person” set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101;
6. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

7. I have participated in the preparation of this Technical Report and am responsible for Sections 18.1 to 18.5, 18.16 to 18.20 and have contributed to Sections 1, 24, 26 and 27.

8. I last visited the property site in 2015/2016.

9. I have had no prior involvement with the property that is the subject of the Technical Report.

10. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report;

11. I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 1st day of April 2021

"Original Signed and sealed"
David Morgan, AusImm (#202216).
Managing Director

CERTIFICATE OF AUTHOR

To accompany the Report entitled “NI 43-101 Technical Report – Pre-Feasibility Study of the Kalana Gold Project, Republic of Mali”, which is effective as of December 31, 2020 and issued on April 1, 2021 (the “Technical Report”) prepared for Endeavour Mining (the “Company”).

I, Allan Earl, do hereby certify that:

1. I am an Executive Consultant of Snowden Mining Industry Consultants (Pty) Limited, 580 Hay St Street, Perth, Western Australia.

2. I graduated with an Associateship in Mining Engineering from the Western Australian School of Mines in 1977.

3. I am a Fellow of the AusIMM, with membership number 110247.

4. I have worked as a mining engineer continuously for over 40 years since my graduation. I have been involved in mining and resource evaluation practice consulting practice for over 20 years, including scoping studies, prefeasibility studies and feasibility studies including reserve estimation for open pit gold mines for at least five years.

5. I have read the definition of “qualified person” set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101

6. I am independent of the issuer as defined in Section 1.5 of the Instrument.

7. I am responsible for the preparation of Sections 15, 16 and 21 (pertaining to mining capital and operating cost); and have contributed to parts of sections 1, 2, 3, 24, 25 and 26 of the Technical Report.

8. I visited the Kalana Main Project in 2015 and in 2018.

9. I have had prior involvement with the property that is the subject of the Technical Report. I have reviewed work and technical reports of the same Project completed in 1999 and in 2014.

10. I have no personal knowledge as of the date of this certificate of any material fact or change,

which is not reflected in this Report;

11. I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia, on the 1st day of April 2021

"Original Signed and sealed"

Allan Earl AWASM FAusIMM
Executive Consultant

CERTIFICATE OF QUALIFIED PERSON

Paul Blackney
Optiro Pty Ltd
16 Ord St, West Perth WA 6005, Australia
Telephone: +61 (0)8 9215 0000
Email: pblackney@optiro.com

To accompany the Technical Report entitled: "*Technical Report – Pre-Feasibility Study of the Kalana Gold Project, Republic of Mali*" which is effective as of December 31, 2020 and dated as of April 1, 2021 (the "Technical Report") prepared for Endeavour Mining (the "Company").

I, Paul Blackney, BSc (Hons), MAIG, MAusIMM, do hereby certify that:

1. I am a Principal Consultant at Optiro, a minerals industry advisory organisation based in West Perth, Western Australia.
2. I am a graduate of the University of Tasmania, Australia, with a BSc (honours) in geology completed in 1982. I have practised my profession continuously since 1983. For the purposes of the Technical Report, I have over 35 years experience in the minerals industry worldwide, with a substantial emphasis on gold, focussing on mining geology and Mineral Resource estimation.
3. I am a Member of the Australian Institute of Geoscientists and Australasian Institute of Mining and Metallurgy.
4. I have read the definition of "Qualified Person" set out in National Instrument 43- 101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I have had no prior involvement with the properties that are the subject of the parts of the Technical Report for which I am taking responsibility.
6. I am responsible for the parts of Section 1 and 14 of the Technical Report dealing with the Kalana and Kalanako Mineral Resource Estimates.
7. I fulfil the definition of an independent Qualified Person within the meaning of section 1.5 of National Instrument 43-101 – Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators.
8. I have read NI 43-101 and Form 43-101F1 and have read the previously mentioned sections of the Technical Report, in compliance with NI 43-101 and Form 43-101F1.
9. That, at the effective date of this Technical Report December 31st 2020, to the best of my knowledge, information, and belief it contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

This 1st day of April 2021

Original Signed

"Original Signed and sealed"

P C J Blackney BSc (Hons) Geology
Principal Consultant, Optiro

CERTIFICATE OF AUTHOR

To accompany the Report entitled "*NI 43-101 Technical Report – Pre-Feasibility Study for the Kalana Gold Project*" which is effective as of December 31, 2020 and issued on April 1, 2021 (the "Technical Report") prepared for Endeavour Mining (the "Company").

I, *Helen Oliver FGS CGeol.,* do hereby certify:

1. I am the Group Resource Geologist at Endeavour Mining, with an office at 5 Young Street, London W8 5EH, United Kingdom.
2. I am a graduate of Camborne School of Mines, University of Exeter in the UK, with a M.Sc. in Mining Geology obtained in 1995.
3. I am a Fellow of the Geological Society (FGS) of London and a Chartered Geologist (CGeol.), membership number 1012491.
4. I have worked as a geologist continuously since my graduation from university and have been employed since 1996. My relevant experience for the purpose of this Technical Report includes:
    - Three years with Endeavour Mining as their Group Resource Geologist.
    - Working as a consulting or contract geologist for over ten years reviewing and reporting on numerous exploration and mining operations, predominately in Africa.
    - Employed in the Southern African mining and exploration industry for seven years.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
6. I am not independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
7. I have participated in the preparation of this Technical Report and am responsible for Sections 4 to 12, parts of Section 14, Section 23 and parts of Sections 24 to 27.
8. I regularly visited the property site between October 2017 and October 2019 (the last visit was on 4th – 12th October 2019).
9. I have had no prior involvement with the property that is the subject of the Technical Report.

10. I have no personal knowledge as of the date of this Certificate of any material fact or change, which is not reflected in this Report.

11. I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the Report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 1st day of April 2021

"Signed & Sealed"

Helen Oliver FGS CGeol.
Group Resource Geologist

CERTIFICATE OF AUTHOR

To accompany the Report entitled “NI 43-101 Technical Report – Pre-Feasibility Study of the Kalana Gold Project, Republic of Mali” which is effective as of December 31, 2020 and issued on April 1, 2021 (the “Technical Report”) prepared for Endeavour Mining (the “Company”).

I, *Patrick Pérez, P. Eng.,* do hereby certify:

1. I am Director of Technical Studies with Endeavour Mining, with an office at 100 Boulevard Alexis Nihon, 7th Floor, Saint Laurent (Montreal), Canada.
2. I am a graduate from “Ecole Nationale Supérieure de Géologie de Nancy”, in France, with a M.Sc. in Geological Engineering obtained in 2003.
3. I am a registered member of APEGS (Association of Professional Engineers and Geoscientists of Saskatchewan), membership #16131.
4. I have worked as a Mining Engineer, Senior Mining Engineer or Project Manager continuously since my graduation from university. I have been employed since my graduation in 2003. I have gained relevant experience on deposits and projects similar to the Kalana Project, including:
    a) Work on gold mining operations and projects hosted in Birimian greenstone belt formations in west Africa, in both Open Pit and Underground environment
    b) I have also participated and supervised several mineral resource estimates or engineering studies for different projects at various stages of development. Hands-on experience for gold in Ivory Coast, Mali, Burkina Faso, Australia and Canada;
    c) Design, supervision and implementation of mining programs;
    d) Review, audits, interpretation of geoscientific data;
    e) Experience in on several projects in weathered terranes under tropical conditions (West Africa, New Caledonia); and
    f) Participation in the preparation of parts of NI 43-101 compliant Technical Reports.
5. I have read the definition of “qualified person” set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101;

6. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

7. I have participated in the preparation of this Technical Report and am responsible for Sections 19, 20, 21.3, 22 and parts of Sections 1, and 24 to 27.

8. I have not visited the property site.

9. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report;

10. I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 1st day of April 2021

"Original Signed and sealed"
Patrick Pérez, P. Eng.
Director of Technical Studies