Lycopodium

Endeavour Mining Corporation

Lafigué Gold Project in the Fetekro Property, Côte d'Ivoire

Pre-Feasibility Study

National Instrument 43-101 Technical Report (Amended)

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1.0 SUMMARY

1.1 Introduction

This report was prepared to provide a NI 43-101 Technical Report on the Pre-Feasibility Study (PFS) completed for the Lafigué Gold Project ('Lafigué' or the 'Project') in Côte d'Ivoire, which is included in the PR 329 Fetekro Exploration Permit (the 'Permit' the 'Site', or the 'Property'), located approximately 500 km northeast of Abidjan.

La Mancha Côte d'Ivoire (LMCI) is 100% held by Ity Holdings, a fully owned subsidiary of Endeavour Mining Corporation (Endeavour) and has a 100% interest in the exploration permit. LMCI is a company incorporated in Côte d'Ivoire, with the registered address at 08 BP 872 Abidjan 08 - Cocody 180-logements Boulevard Latrille, Immeuble Palm Club, 2nd floor.

This report has been prepared by Lycopodium Minerals Canada Pty Ltd (Lycopodium) at the request of Endeavour and has been prepared in compliance and conformance with the disclosure requirements of the Canadian National Instrument 43-101 (NI 43-101) standards required by and in accordance with the requirements of Form 43-101 F1(F). The Report is intended to be used to demonstrate the results of the PFS.

In compliance with the requirements for the preparation of a NI 43-101 Technical Report, a Qualified Person (QP) has visited/worked at the Project site; however, due to the emergence of COVID-19 in 2020, not all QPs have been able to visit site at the time of filing this Report.

In addition, Lycopodium and other contributors have studied and constructed numerous projects in the region and are familiar with local conditions.

1.2 Contributors

Contributors to the preparation of this Report are summarised in Table 1.2.1. Lycopodium was responsible for compilation of the Report inputs and delivery of the Report to Endeavour.

Table 1.2.1 Report Contributions

Report Section	Contributors
Section 1 – Summary	Lycopodium/Knight Piésold/Snowden/Endeavour
Section 2 – Introduction	Lycopodium
Section 3 – Reliance on Other Experts	Lycopodium
Section 4 – Property Description and Location	Lycopodium
Section 5 – Accessibility, Climate, Local Resources and Infrastructure	Lycopodium
Section 6 – History	Endeavour
Section 7 – Geological Setting and Mineralisation	Endeavour
Section 8 – Deposit Types	Endeavour
Section 9 – Exploration	Endeavour
Section 10 – Drilling	Endeavour
Section 11 – Sampling Preparation, analysis and Security	Endeavour
Section 12 – Data Verification	Endeavour
Section 13 – Mineral Processing and Metallurgical Testing	Lycopodium
Section 14 – Mineral Resource Estimates	Endeavour
Section 15 – Mineral Reserve Estimates	Snowden
Section 16 – Mining Methods	Snowden
Section 17 – Recovery Methods	Lycopodium
Section 18 – Project Infrastructure	Lycopodium, Knight Piésold
Section 19 – Market Studies and Contracts	Endeavour
Section 20 – Required Permits and Environmental Considerations	Endeavour
Section 21 – Capital and Operating Cost	Lycopodium, with input from Knight Piésold/Endeavour
Section 22 – Economic Analysis	Endeavour
Section 23 – Adjacent Properties	Endeavour
Section 24 – Other Relevant Data and Information	Lycopodium/Knight Piésold/Snowden/Endeavour
Section 25 – Interpretation and Conclusions	Lycopodium/Knight Piésold/Snowden/Endeavour
Section 26 – Recommendations	Lycopodium/Knight Piésold/Snowden/Endeavour
Section 27 – Selected Reference	Lycopodium/Knight Piésold/Snowden/Endeavour

1.3 Property Access and History

Côte d'Ivoire is located along the coast of western Africa and is bordered to the east by Ghana, to the north by Burkina Faso and Mali, and to the west by Guinea and Liberia. The Property is located in the north-central part of Côte d'Ivoire approximately 500 km from Abidjan. The Property is located within the Dabakala Department, in the Hambol Region of the Vallée du Bandama District.

The Lafigué Project sits five kilometres from its namesake village, in the northeastern corner of the Fetekro Property.

Access to the Property is via sealed roads to Yamoussoukro and Bouaké, following the regional highway B412 east from Katiola. A 15 km unsealed track (to be upgraded pre-construction) extends to Lafigué. The general location of the Project, and its proximity to local towns is illustrated in Figure 1.3.1 and Figure 1.3.2 following.

An exploration permit was first sought for the area in 1993 by SODEMI which signed a partnership agreement with GENCOR (through its Ivoirian Company GATRO-CI). In 1999, the Compagnie Minière Or (COMINOR) took over La Source participation and the GATRO-CI contractual commitments under the Exploration Agreement. In 1999, COMINOR started an exploration programme on the permit, focusing mainly on the Lafigué area.

In 2000, COMINOR was transferred to Compagnie Générale des Matières Atomiques (COGEMA) which was incorporated into La Mancha Group in 2006, via a reverse takeover of La Mancha by Compagnie Française de Mines et Métaux (CFMM), a wholly owned subsidiary of AREVA group.

Field work progressed but was put on hold from 2002 to 2010 as a result of civil war. In 2013, AREVA sold its gold assets in Côte d'Ivoire to a private fund. In 2014, La Mancha Côte d'Ivoire S.a.r.l (LMCI) was incorporated in Côte d'Ivoire as a 100% subsidiary of COMINOR and took over the exploration activities of COMINOR managed by its Ivoirian branch, COMINOR CI, including COMINOR contractual commitments under the Exploration Agreement.

LMCI become a wholly owned subsidiary of Endeavour Group in November 2015. Subsequently, LMCI has been held 100% by Ity Holdings (Ity), a wholly owned subsidiary of Endeavour. Endeavour began exploration on the Property in March 2017, following a strategic assessment of its exploration tenements which ranked the Property as a top priority target.

Figure 1.3.1 General Property and Project Location

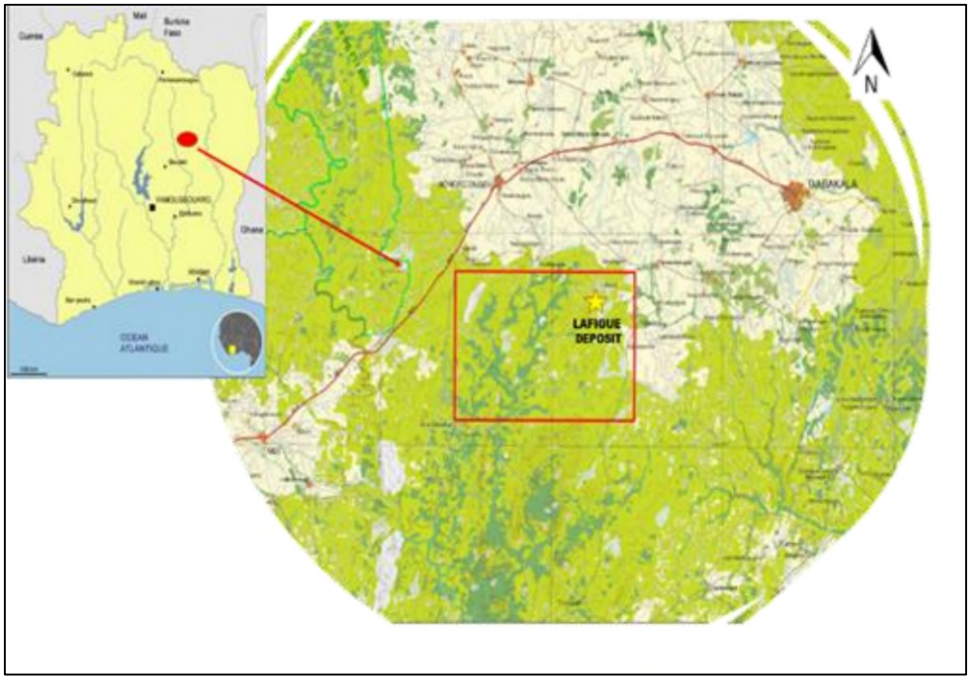
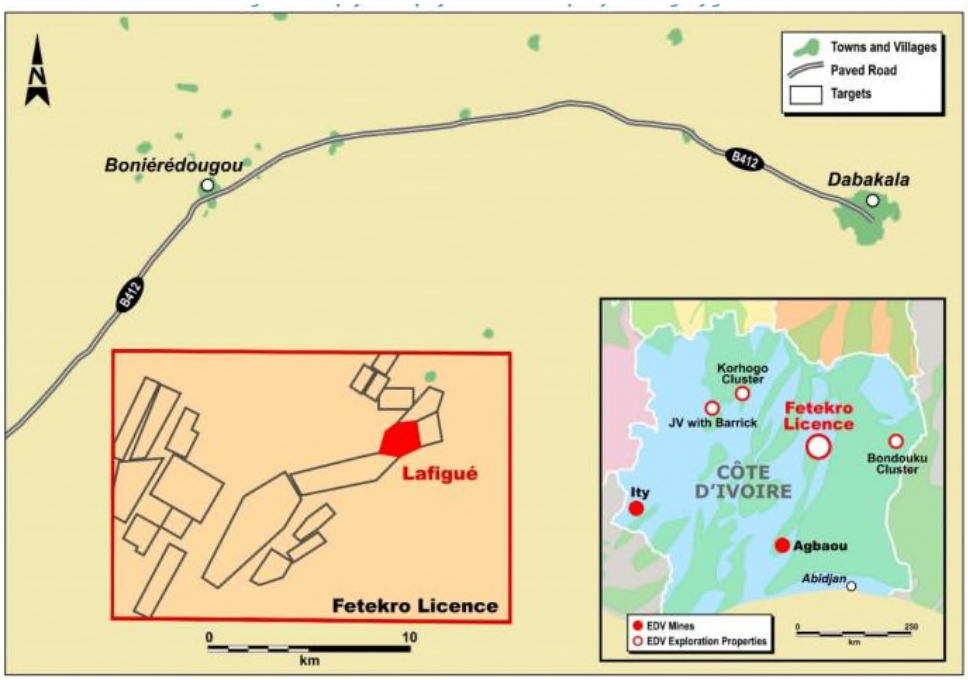


Figure 1.3.2 Lafigué and Endeavour Côte d'Ivoire Projects



1.4 Licence Status

The existing exploration permit PR 329 is owned by SODEMI and operated by La Mancha Côte d'Ivoire (LMCI).

The permit was assigned to SODEMI by decree N°2013-410 on June 6, 2013. Recently, pursuant to a sale of exploration permit, PR 329 was transferred from SODEMI to LMCI by Ministerial order N° 00174/MMG/DGMG of December 18, 2020. The Permit covers an area of 249.5 km², as illustrated in Figure 1.4.1.

Figure 1.4.1 Property Location

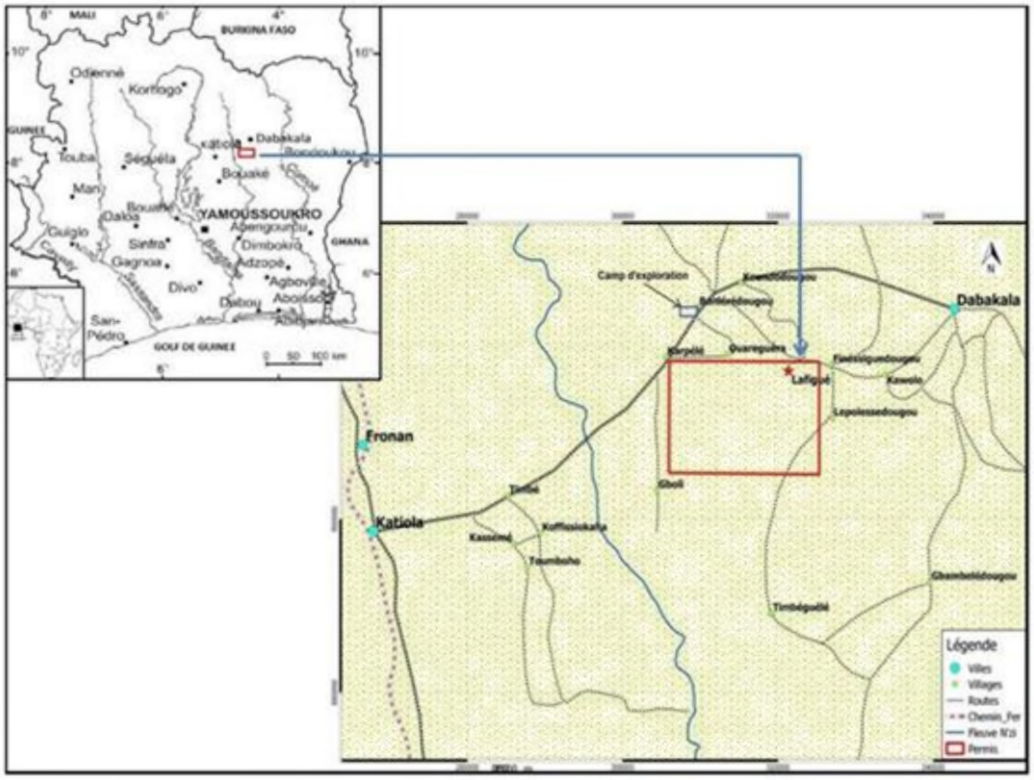


Table 1.4.1 Property Coordinates

Point	Latitude North	Latitude East
A	8° 18′ 12.00″	4° 45′ 40.00″
B	8° 18′ 12.00″	4° 35′ 10.00″
C	8° 11′ 12.00″	4° 35′ 10.00″
D	8° 11′ 12.00″	4° 45′ 40.00″

According to the 2014 Mining Code in Côte d'Ivoire, an exploration permit is granted by a Presidential Decree. The exploration permit is valid for an initial period of four years, it may be renewed for two consecutive periods of three years and a final exceptional renewal for a two-year period, provided the titleholder complies with the rights and obligations set by the Mining Legislation. At each renewal, at least 25% of the original area must be relinquished, however the titleholder has the option to maintain the right over the full area by paying an 'option fee'.

The Mining Code gives the exploration permit holder the exclusive right to explore for the minerals requested, on the surface and in the subsurface, within the boundaries of the permit.

The application for a permit must be filed along with a programme of exploration works that the applicant contemplates to carry out during the first four years, along with the related budget of such programme. The titleholder of a permit must start exploration works within the covered perimeter no later than six months from the date of validity of the permit and must continue actively carry out work during this time.

Once an Industrial Operating Permit (i.e., a mining or exploitation licence) covering the Lafigué deposit is granted, Endeavour will be entitled to an 80% stake in 'Société des Mines de Lafigué', the exploitation entity which will mine the Lafigué deposit while SODEMI and the Government of Côte d'Ivoire will each retain 10% of the remaining 20% stake respectively. Endeavour, through its exploration subsidiary La Mancha Côte d'Ivoire, will retain the full ownership of the remaining area of the Fetekro exploration licence, until such time as it is converted into mining licences, subject to the discovery of new deposits, or the permit expires.

1.5 Geology and Mineralisation

The Fetekro Property, which includes the Lafigué gold deposit, is located at the northern end of the Lower Proterozoic Oumé-Fetekro Greenstone Belt, a north-south elongated belt which is approximately 300 km long and 20 km wide. The Oumé-Fetekro Greenstone Belt is composed mainly of Birimian volcano-sediments, consisting mainly of mafic to intermediate metavolcanics, felsic metavolcanics, and clastic metasediments, that are bound and intruded by granitoid complexes. Known gold deposits such as Bonikro and Agbaou are hosted within the Oume-Fetekro Belt.

The Lafigué gold deposit has been traced over two kilometres along an east-northeast (ENE) axis and over one kilometre down-dip. It is a Birimian volcanic complex predominately composed of mafic rocks, namely metagabbros/metanorites and metabasalts and felsic intrusives (granodiorite or tonalite) that lies in the western part of the Fetekro Property. This volcanic complex is affected by a transpressive deformation and intruded by granodioritic bodies and quartz-porphyry dykes. Regional foliation varies in strike from north-south to N070°, with gentle to intermediate/steep dips to the east and south (25° to 65°).

The gold mineralisation is mesothermal in origin and occurs as free gold in quartz vein stockworks and zones of silicification, associated with tourmaline, calcite, ankerite and pyrite. The gold mineralisation is found in linear zones in or near the contacts between two different rock types (metabasalt and metagabbro or metabasalt and intrusive). This contact shows evidence of shearing. Alteration is weak to severe depending on the development of the system.

1.6 Exploration and Drilling

An exploration permit was first sought for the area in 1993 by SODEMI, with initial site exploration works commencing in 1996.

In 1996, an exploration, development and operating agreement was entered into by SODEMI (SODEMI/BRGM/La Source), the title holder, and GENCOR (through its Ivoirian company GATRO-CI) for the Project (the Exploration Agreement). GATRO-CI completed classical stream sediment then soil geochemical sampling, pits, trenching and limited drilling (14 diamond drill (DD) holes for 1,446 m and 37 reverse circulation (RC) for 1,549 m). Four main targets were identified, including Lafigué. A

first preliminary polygonal mineral estimation (non NI 43-101 compliant) for internal use was done by GATRO-CI in February 1998. No top cut was applied. GATRO-CI found 1,595 Mt at 2.87 g/t Au in the overburden and in the oxidised zone.

In 1999, COMICOR took over La Source participation and the GATRO-CI contractual commitments under the Exploration Agreement. In 1999, COMINOR started an extension exploration program in Lafigué to find new resources to replace Angovia's production after its depletion.

In 2002, COMINOR realised 1,803 m of rotary air-blast (RAB) drilling, 1,281 m of RC drilling and 461 m of DD drilling, which showed that the mineralisation is not continuous between Lafigué Centre and Lafigué Nord, and felsic dykes play a role in mineralisation control.

In 2002, COGEMA reviewed the GATRO-CI preliminary estimation using a geostatistical method (Grove 2002). The prospect was split between the Southern, Central and Northern zones finding 3.5 Mt at 2.44 g/t Au in oxide and sulphide zone. Field works were put on standby from 2002 to 2010 due to the Ivorian civil war.

A revised estimation was completed in 2003 by COGEMA with new densities measured and an updated geological model. COGEMA calculated 3.7 Mt at 2.43 g/t Au in oxide and sulphide zone.

The above historical resource estimates were not reported publicly, or within any regulatory environment. The estimates were intended for the internal use of the companies involved only. These resource estimates have not been evaluated by the QPs for this Report, are not current, and are reported for background information only.

In 2010, COMINOR drilled 11 RC holes for 1,109 m and four DD holes for 396 m to verify the mineralisation extension downdip in the central area.

In 2014, LMCI drilled 23 DD holes (1,864 m) and 54 RC holes (4,634 m) to obtain structural data and to check extension in the north area of Lafigué prospect. In 2015, LMCI completed a LIDAR survey.

A summary of exploration work completed prior to LMCI prior to November 2015 is presented in Table 1.6.1.

Table 1.6.1 Drilling Summary 1997 to 2014

Year	Area	Drill Type	No. Drill Holes	Metres
1997	Lafigué	RC	37	1,549
1997	Lafigué	DD	14	1,446
2002	Lafigué	RAB	75	1,803
2002	Lafigué	RC	32	1,281
2002	Lafigué	DD	11	461
2002	Target 8	RAB	19	294
2010	Lafigué	RC	11	1,109
2010	Lafigué	DD	4	396
2014	Lafigué	RC	54	4638
2014	Lafigué	DD	23	1,864
Total			**280**	**14,548**

Between 2017 and 2020, 39 diamond drill (DD) holes for 6,598 m, 50 reverse circulation (RC) holes for 80,151 m and 164 RC-DD holes for 51,322 m were drilled on the Lafigué deposit.

In addition, 14 DD holes for 2,244 m and 182 holes for 22,301 m were drilled on the exploration targets (excluding Lafigué) within the Fetekro Property,

These drill campaigns are summarised in Table 1.6.2 and Table 1.6.3 following.

Table 1.6.2 Lafigué Deposit Drilling Summary 2017-2020

Year	Drill Type	No Drill Holes	Metres	Company
2017	RC	119	6,582	FORACO
	DD	6	762	FTE
2018	RC	44	7,610	GEODRILL
	RC-DD	8	2,662	GEODRILL
	DD	18	3,293	FORACO-GEODRILL
2019	RC	172	30,628	GEODRILL
	RC-DD	27	7,804	GEODRILL
	DD	15	2,543	FORACO-GEODRILL
2020	RC	165	35,331	GEODRILL
	RC-DD	129	40,856	FORACO-GEODRILL
Total		**703**	**138,071**	

Table 1.6.3 Fetekro Property (excl. Lafigué) Drilling Summary 2017-2020

Year	Area	Drill Type	No Drill Holes	Metres
2017	Target 1	RC	10	1,224
	Target 1	DD	7	897
	Target 2	RC	15	1,506
	Target 2	DD	2	290
	Target 3	RC	3	715
	Target 3	DD	1	237
	Target 4	RC	10	1,211
	Target 4	DD	1	252
	Target 5	RC	5	378
	Target 6	RC	7	461
	Target 8	RC	2	590
	Regional	RC	8	643
	Subtotal		**71**	**8,404**
2018	Target 1	RC	16	2,013
	Target 2	DD	1	327
	Target 3	RC	3	369
	Target 4	RC	14	1,759
	Target 7	RC	10	726
	Target 8	DD	2	241
	Target 9	RC	2	248
	Target 10	RC	4	484
	WA01	RC	10	1,203
	WA03	RC	2	235
	Subtotal		**64**	**7,605**
2019	Target 2	RC	5	984
	Target 7	RC	6	720
	Target 8	RC	9	1,093
	WA02	RC	12	1,641
	WA06	RC	11	917
	WA10	RC	2	202
	WA13	RC	9	1,118
	WA14	RC	2	330
	Subtotal		**56**	**7,005**
2020	Target 1	RC	2	568
	Target 2	RC-DD	2	640
	Target 7	RC	1	323
	Subtotal		**5**	**1,531**
Total			**196**	**24,545**

1.7 Mineral Resource

The statistical analysis, geological modelling and resource estimation were prepared by Kevin Harris, CPG. Mr Harris is Endeavour Mining's Vice-President Resources and a Qualified Person as defined by NI 43-101.

The Lafigué resource model was developed in GEOVIA Surpac™ software. A total of 24 mineralised zones were defined from the current drilling data and geologic interpretations across Lafigué South, Centre, and North areas. The gold assays from the drill holes were composited to one metre intervals within the mineralised wireframes and capped from 15 g/t to 30 g/t Au. Spatial analysis of the gold distribution within the mineralised zone using variograms indicated a good continuity of the grades along strike and down dip of the mineralised zones.

Density was measured in 2,282 core samples within the various rock types then averaged within the model by the weathered zones. The laterite density is 2.0 g/cm^3, the saprolite density is 1.80 g/cm^3, the transition is 2.4 g/cm^3, and the fresh rock is 2.80 g/cm^3.

The gold grade was estimated using ordinary kriging constrained within the mineralised domains. The grade was estimated in multiple passes to define the higher confidence areas and extend the grade to the interpreted mineralised zone extents.

The grade estimation was validated with visual analysis and comparison with the drilling data on sections and with swath plots comparing the block grades with the composites.

The mineralised domains were classified into Indicated and Inferred Mineral Resource classifications, depending on the sample spacing, number of samples, confidence in mineralised zone continuity, and geostatistical analysis. The Indicated classification was generally applied to blocks within the mineralised zoned defined by a minimum of seven samples from at least three drill holes with a 50 m search. The Inferred classification was defined by a minimum of three samples within a 75 m search from two drill holes. The classification was cleaned to minimise some of the isolated blocks within the areas of good continuity.

A preliminary pit optimisation exercise based on assumed costs and recoveries (refer Section 3.2.11) was conducted to allow reporting within an optimised pit shell to demonstrate that the Mineral Resource has reasonable prospects for future economic extraction. The Resource was constrained by a $1,500/oz Au pit shell and 0.50 Au g/t cut-off.

Table 1.7.1 Lafigué Mineral Resource as of July 31, 2020

Indicated			Inferred		
Tonnes (kt)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kt)**	**Grade (g/t)**	**Au (koz)**
32,030	2.40	2,471	820	2.52	66

Notes: Effective date July 31, 2020. 0.5 Au g/t cut-off grade. US$1,500/oz Au MII optimised Whittle pit shell constrained. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

1.8 Mineral Reserve

Snowden completed pit optimisation, mine design and developed the Life of Mine (LOM) schedules based on the Endeavour Lafigué deposit block model (fetekro_bm_july202.dm). The deposit is near surface and is amenable to open pit mining, which does not preclude future underground mining. Although the underground potential for further expansion was assessed during the PFS, Snowden determined it was not feasible given the extents of the current resource. Further expansion of the orebody at depth would require re-evaluation of the underground potential.

Reference mining costs summarised in Table 1.8.1, used included allowances for load and haul, drill and blast, dewatering, mining labour, fuel, and mining mobile equipment maintenance. Processing costs used in pit optimisation are shown in Table 1.8.2.

Table 1.8.1 Pit Optimisation Reference Mining Costs by Weathering Type

Weathering Type	Above Reference Level (US$/dt)	Reference Level (mRL)	Below Reference Level (US$/dt)
Laterite	2.70 - 0.0022*level	380	2.70 + 0.0022*level
Saprolite	2.70 - 0.0022*level	370	2.70 + 0.0022*level
Saprock	3.03 - 0.0022*level	310	3.03 + 0.0022*level
Fresh	3.03 - 0.0022*level	210	3.03 + 0.0022*level

Table 1.8.2 Pit Optimisation Processing Costs

Cost	Laterite (US$/dt)	Saprolite (US$/dt)	Saprock (US$/dt)	Fresh (US$/dt)
Processing - fixed	3.28	3.28	3.48	3.83
Processing - variable	6.09	5.95	7.40	8.90
G&A	5.74	5.74	5.74	5.74
Total Ore Cost	**15.11**	**14.97**	**16.62**	**18.47**

Other parameters used in pit optimisation included a royalty of 4.0% and stamp duty of 0.5% applied to all sales based on the price. A transport and refining charge of US$4/oz was also applied. 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. Except for mining mobile equipment, which was applied as part of the mining operating cost, no capital or taxation costs were included.

The Probable Mineral Reserve estimate for the Lafigué Project is 31.9 Mt at 2.0 Au g/t as reported in Table 1.8.3. Inferred material within the pit design was treated as waste for scheduling and reporting purposes.

Table 1.8.3 Lafigué Mineral Reserve as of December 31, 2020

Item	Proven	Probable	Total
Tonnes (Mdt)	0	31.9	31.9
Gold grade (g/t)	0	2.0	2.0
Gold contained (Moz)	0	2.05	2.05

1.9 Mining

1.9.1 Mining Inventory

At the economic cut-off grades determined, a total of 31.9 Mdt at 2.0 Au g/t was included in the pit designs. Table 1.9.1 summarises the pit design inventory by Mineral Resource category, with Inferred material treated as waste for scheduling and reporting purposes.

Table 1.9.2 summarises the total pit design inventory by weathering type. Table 1.9.3 shows the marginal cut-off grades determined by weathering type.

Table 1.9.1 Pit Design Inventory by Resource Category

Item	Measured	Indicated	Inferred	Total
Tonnes (Mdt)	-	31.9	0.5	32.4
Gold (g/t)	-	2.0	2.0	2.0

Table 1.9.2 Pit Design Inventory by Weathering Type

Item	Laterite (Mdt)	Saprolite (Mdt)	Saprock (Mdt)	Fresh (Mdt)	Total (Mdt)
Total	1.3	18.2	23.0	319.4	361.8
Waste	1.0	16.9	22.0	290.0	329.9
Mineralised	0.2	1.2	1.0	29.4	31.9
Gold Grade (g/t)	2.3	1.7	1.7	2.1	2.0

Table 1.9.3 Pit Optimisation Cut Offs

Weathering Type	Au (g/t)
Laterite	0.34
Saprolite	0.34
Saprock	0.38
Fresh	0.43

1.9.2 Production Schedule

Figure 1.9.1 summarises the total ex-pit material movement classified by ore and waste as well as high, medium and low grade ores. Peak production year is Year 2, with the ore feed by material type shown in Figure 1.9.2. The process plant feed rate for the first two years of operation is higher than the design throughput rate of 3.0 Mtpa, primarily due to the predominantly oxide feed source.

Figure 1.9.1 Total Ex-pit Movement by Material Type

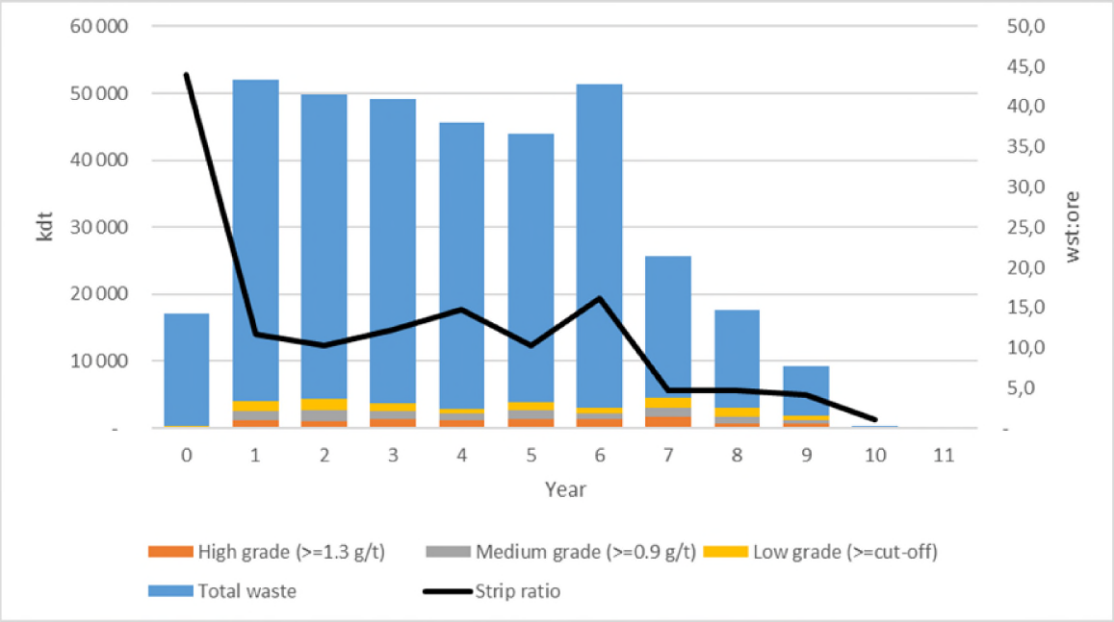
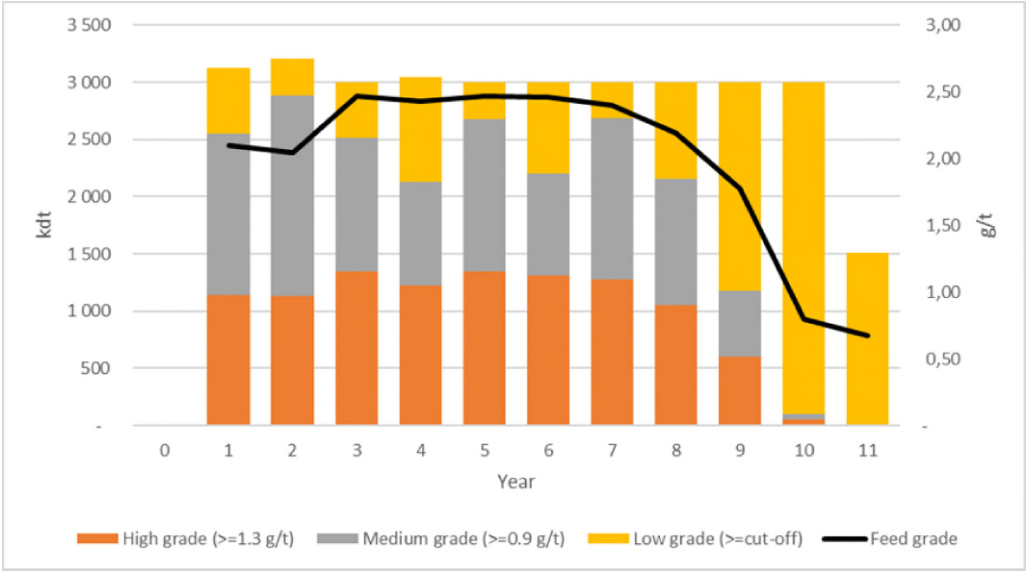


Figure 1.9.2 Total Ore Feed by Material Type



1.9.3 Mining Equipment and Labour Requirements

A contract mining strategy is preferred for the deposit at this stage, and was the considered option in the PFS. It is expected however, that all material will be mined using 100 t and 200 t hydraulic excavators (i.e. backhoe configuration) to maximise equipment productivities. This would include but not be limited to potential equipment manufacturers such as Caterpillar 6015 and 6020B, and Komatsu PC1250 and PC2000.

All ex-pit material will be hauled using 90 t capacity rigid rear-dumping trucks. Examples of this size and type of equipment include Komatsu HD785-7, Caterpillar 777G and Hitachi EH1700-3.

Early engagement with an experienced mining contractor will be undertaken during the Feasibility Study (FS) to determine and confirm equipment and labour numbers. As an indication, the PFS has estimated the total mining (Endeavour plus Contractor) personnel requirements of 519, as follows:

- International expatriate management and professional - 12.
- Regional expatriate management and professional - 40.
- Local National - 72.
- Mining Contactor - 395.

1.10 Metallurgy

1.10.1 Scouting Testwork 2018

A scouting testwork programme was initiated in 2018 and was undertaken by Endeavour at ALS Global laboratory in Perth, Western Australia on ten variability and two comminution samples. The ten variability samples covered the three main ore types from the prospect; oxide (three samples), low grade fresh (five samples) and high grade fresh (two samples) ore.

The key outcomes of the comminution programmes were as follows.

- The low Axb and high Drop Weight Index (DWi) value for Composite 2 indicates a competent ore with a high breakage energy requirement. The Bond ball work and abrasion indices are moderate.
- The low Axb (and high DWi) value for comminution Composite 2 indicates a competent ore with a high breakage energy requirement. The Bond ball work and abrasion indices are moderate. Composite 1, from a shallower depth and may be more weathered, was less competent and had lower ball work and abrasion indices.
- Detailed head assays showed few deleterious elements for gold leaching with low levels of base metals and arsenic. Silver head assays for the variability samples were low.

- At a grind size of 80% passing (P_{80}) 75 µm, the variability samples ranged from 20% to 85% gravity gold recovery.
- All overall gold extractions were greater than 96%.
- After removal of the gravity gold, leach kinetics were fast with the bulk of gold leached within four to eight hours.
- Reagent requirements were low for the fresh samples and low to moderate for the oxide sample.
- The economic optimum grind appeared to be a P_{80} of 106 µm with similar gold recoveries and lower operating costs when compared with P_{80} of 75 µm.
- Rheology testwork indicated that the fresh composites had low to moderate viscosities, while the oxide composite viscosity was extremely high.

1.10.2 2019 Detailed Testwork Programme

The 2019 testwork programme built on the scouting testwork and involved a detailed metallurgical and comminution testwork programme on 36 variability samples and 12 comminution samples, representative of the 2019 Mineral Resource.

The programme design had the following key objectives:

- Evaluate the variability in comminution parameters for the range of ore types.
- Perform detailed head assays and a mineralogical investigation.
- Confirm the suitability of the selected gravity / leach processing route.
- Generate optimised processing conditions for the selected route.
- Evaluate the variability in metallurgical performance for the range of ore types.
- Determine metallurgical recoveries and operating cost input data for the selected route and range of ore types.
- Determine physical slurry characteristics and engineering parameters for process plant design.

The key outcomes were as follows:

- Comminution testwork results were reasonably consistent with low Axb values and the DWi values high (90th percentile in the JKTech database) for the fresh ores. This indicates that the fresh ore is very competent with a high breakage energy requirement. The Bond Rod mill work index (RWi) and Bond Ball mill work index (BWi) while not as extreme, were still moderately high. The abrasion index is considered low to moderate.

- As per the scouting programme, triplicate gold head assays typically showed significant variation due to the high gravity gold content and silver assays were consistently low.

- There were no deleterious elements for gold leaching present with low levels of base metals, antimony and tellurium. Mercury and arsenic levels were also low and should not present an environmental or occupational health risk in the elution or electrowinning circuits.

- The ore appeared to be relatively insensitive to grind with only a 1% difference in gold extraction over the size range tested. The optimum grind size was confirmed at P_{80} 106 µm.

- The gravity gold content was very high and ranged from 25 - 92% with an average of 61% GRG.

- Overall gold extractions were very high for both fresh and oxide ores with an average gravity / leach gold extraction of >97%.

- Gold extraction after gravity removal was fact and essentially complete within eight to 12 hours.

- Leach optimisation testing on the master composite samples indicated that high gold extractions were achieved with air only sparging, low cyanide dosing and high fresh ore slurry densities (up to 55% solids w/w). Testwork on oxide ore was conducted at 40% solids w/w.

- Whole of ore leaching (without gravity gold removal) had a much slower leaching rate and overall gold extraction.

- Variability leach testwork on 31 samples achieved >95% gold extraction for all but five samples. Increased cyanide addition and extra leach residence time to 36 hours improved extractions to expected levels.

- Reagent consumption was low. Cyanide consumption for the fresh ore will be 0.13 kg/t while the oxide requires 0.16 kg/t. These figures include a free cyanide excess loss of 100 ppm NaCN. Plant lime usage will be 0.17 kg/t for the fresh ore while the oxide requires 1.65 kg/t.

- On the basis of treating Lafigué fresh and oxide ores via gravity and direct cyanidation, an overall gold recovery of 95.8% is recommended. This recovery is based on the median of the variability testwork results after allowing for the likely soluble gold loss equivalent of 1% Au.

- Rheology testwork indicated that both the fresh and oxide composites had low to moderate viscosities. The extremely high oxide viscosity noted in the 2018 testwork was not present in the 2019 testwork.

- Engineering parameter testwork was completed and was used for the PFS engineering design.

1.10.3 Current Programme

The Lafigué deposit comprises a network of mineralised high grade shear zones. Visible gold can be observed in these veins. Metallurgical and comminution testwork has been undertaken for the Lafigué Gold Project on representative samples sourced from the expected minable pit based on the July 2019 Mineral Resource Estimate.

It was recognised that the 2019 Resource had upside potential due to the mineralised zone continuing at depth to the southeast. Further drilling and geological investigations increased the 2020 Mineral Resource Estimate significantly.

Whilst testwork has not yet been completed on ore from the expanded Resource, it is expected that the metallurgical and comminution characteristics of this material will be similar to the Lafigué ores already tested. Therefore, the available testwork is considered a suitable basis for this PFS. To confirm this, additional testwork on ores representative of the additional Resource is planned by Endeavour for early 2021 with results for all material to be available for the FS phase.

1.11 Process Plant

1.11.1 Design Philosophy

The process plant design for the Lafigué Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimal operating costs. The flowsheet is based on unit operations that are well proven in industry.

The Lafigué plant will process fresh and oxide ores and is expected to operate on either 100% fresh ore (possibly with a small portion of oxide) or 100% oxide ore. The key project and ore specific design criteria that the plant design must meet are as follows:

- 3,000,000 t/y of fresh ore. The plant is expected to process 3,500,000 t/y of oxide ore.

- When processing fresh ore, mechanical availabilities of 70% for the closed-circuit secondary crushing plant, 86.7% for the closed circuit HPGR crushing plant 91.3% for the remainder of the plant, supported by crushed ore storage and standby equipment in critical areas.

- When processing oxide ore, mechanical availability of 88.0% with direct feed of primary crushed ore to the ball mill.

- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

1.11.2 Selected Flowsheet

The treatment plant design incorporates the following unit process operations:

- Primary jaw crushing to produce a coarse crushed product.

Fresh Ore

- Secondary cone crushing in closed circuit with a dry sizing screen to produce an intermediate crushed product.

- A live secondary crushed ore stockpile, providing coarse crushed ore storage and reclaim to feed the HPGR crushing circuit.

- Tertiary HPGR crushing in closed circuit with a wet sizing screen with undersize slurry reporting to the milling circuit via the mill discharge hopper.

Oxide Ore

- Direct feeding of primary crushed ore to the ball mill feed chute.

Oxide and Fresh Ores

- A ball mill in closed-circuit with hydrocyclones to produce a grind size P_{80} of 75 µm (micron).

- 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.

- Trash screening to remove any wood trash or oversize material prior to cyanidation.

- Pre-leach thickening of the trash screen underflow to produce a higher solids concentration leach feed to reduce leach and adsorption tankage and reagent requirements.

- A leaching circuit to leach gold from the milled ore with one pre-leach oxidation tank and five leach tanks providing a total of 22 hours (oxide ore) to 36 hours (fresh ore) residence time.

- A CIP circuit to adsorb leach solution gold onto activated carbon in seven tanks operating in carousel.

- A split AARL elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré.
- Tails wash thickening of the CIP tails to produce a higher solids concentration tails and to recover process water and cyanide from the tails slurry.
- Dilution of the tails thickener underflow with decant return / raw water in order to meet the target plant tails cyanide discharge level.
- Tailings pumping to the tailings storage facility (TSF).

1.11.3 Key Process Design Criteria

The key process design criteria listed in Table 1.11.1 form the basis of the detailed process design criteria and mechanical equipment list.

Table 1.11.1 Summary of Key Process Design Criteria

	Unit	Fresh	Oxide[1]	Source [2,3,4,5,6,7,8,9]
Plant Throughput	t/y	3,000,000	3,500,000	EDV
Life of Mine (LOM) Ore Blend		92%	8%	Snowden
Design Gold Head Grade	g Au/t	2.5	2.0	Assumed
Gravity Gold Recovery	%	60	60	EDV
Design Overall Gold Recovery	%	97	97	Testwork
Primary Crushing Plant Utilisation	%	70	88	Lyco / OMC
Secondary Crushing Plant Utilisation	%	70	-	Lyco / OMC
HPGR Crushing Plant Utilisation	%	86.7	-	Lyco / OMC
Milling / Leaching Plant Utilisation	%	91.3	88	Lyco / OMC
ROM Ore Top Size	mm	800	500	OMC
Ore SG		2.83	2.70	Test / OMC
Comminution Circuit		Prim/Sec/HPGR Crush / Reverse fed Ball Mill	Primary Crush / Direct fed Ball Mill	Endeavour / OMC
Crush Size, P_{80}	mm	7.2	101	OMC
Milling Circuit Top Size, P_{100}	mm	4	101	OMC
Target Grind Size, P_{80}	µm	75	75	Testwork
Cyclone Overflow Density	% solids w/w	25	30	Testwork
Pre-leach Thickener Solids Loading	$t/m^2.h$	0.8	0.5	Testwork / Lyco
Pre-leach Thickener Underflow Density	% solids w/w	65	50	Testwork / Lyco
Leach Feed Slurry Density	% w/w	55	45	Testwork / Lyco
Leach Residence Time	h	36	27	Testwork / Lyco
Number of Leach Tanks		1 Pre-Ox / 5 Leach	6 Leach	Lyco / EDV
Average Cyanide Consumption6	kg/t	0.13	0.21	Testwork
Average Quicklime Consumption7	kg/t	0.19	1.91	Testwork
Adsorption Circuit		Carousel CIP	Carousel CIP	EDV / Lyco
Number of Adsorption Tanks		7	7	Kemix
Carbon per CIP Tank	t/tank	7.5	7.5	Kemix
Carbon Cycle Time	days	1	1	Lyco
CIP Carbon Loading	g Au+Ag/t	3,893	4,017	Kemix / Lyco
Elution Circuit Type		Split AARL		EDV
Elution Circuit Capacity	t/strip	7.5		Lyco
Frequency of Elution	strips/week	7		Lyco
Tailings Thickener Solids Loading	t/m2.h	0.8	0.5	Testwork / Lyco
Tails Thickener Underflow Density	% solids w/w	65	55	Testwork / Lyco

Notes:
1. Oxide includes oxide and transition ores
2. 'Testwork' refers to metallurgical testwork conducted.
3. 'EDV' refers to advice / agreement from Endeavour Mining
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 100 ppm residual NaCN in the CIP tail solution.
7. Lime consumption based on 90% CaO.
8. 'Kemix' refers to carousel CIP vendor.
9. 'Snowden' refers to advice from Snowden Group.

1.12 Infrastructure

The overall project site layout is shown in Figure 1.12.1. The layout shows the major features of the Project and infrastructure including the process plant, tailings storage facility (TSF), accommodation camp, airstrip, roads, power, open pit mine and mining waste dumps.

The plant has been located to suit the available usable area within a restrictive topography, while providing appropriate access for all supporting infrastructure including mining, power supply and TSF. Due consideration for the interaction of these activities is critical for the effective operation of the processing plant and is captured in the overall plant layout.

Where practical, general project facilities will be located around the process plant site to maintain a single centre of operations. The main exception to this is the accommodation camp, which is deliberately somewhat remote from the process plant, to minimise noise and light interference from the process plant.

The main access route to the processing plant and project site will be from the northeast, and the layout provides easy access for personnel and material movements.

1.12.1 Buildings and Infrastructure

The Project site will be fenced to prevent animal access and deter access by unauthorised persons. Road access into the fenced area will be through a manned checkpoint. Site security is based on concentric lines of fencing / control. Security fencing will surround the accommodation camp and general site infrastructure. Monitored high security fencing will surround the process plant.

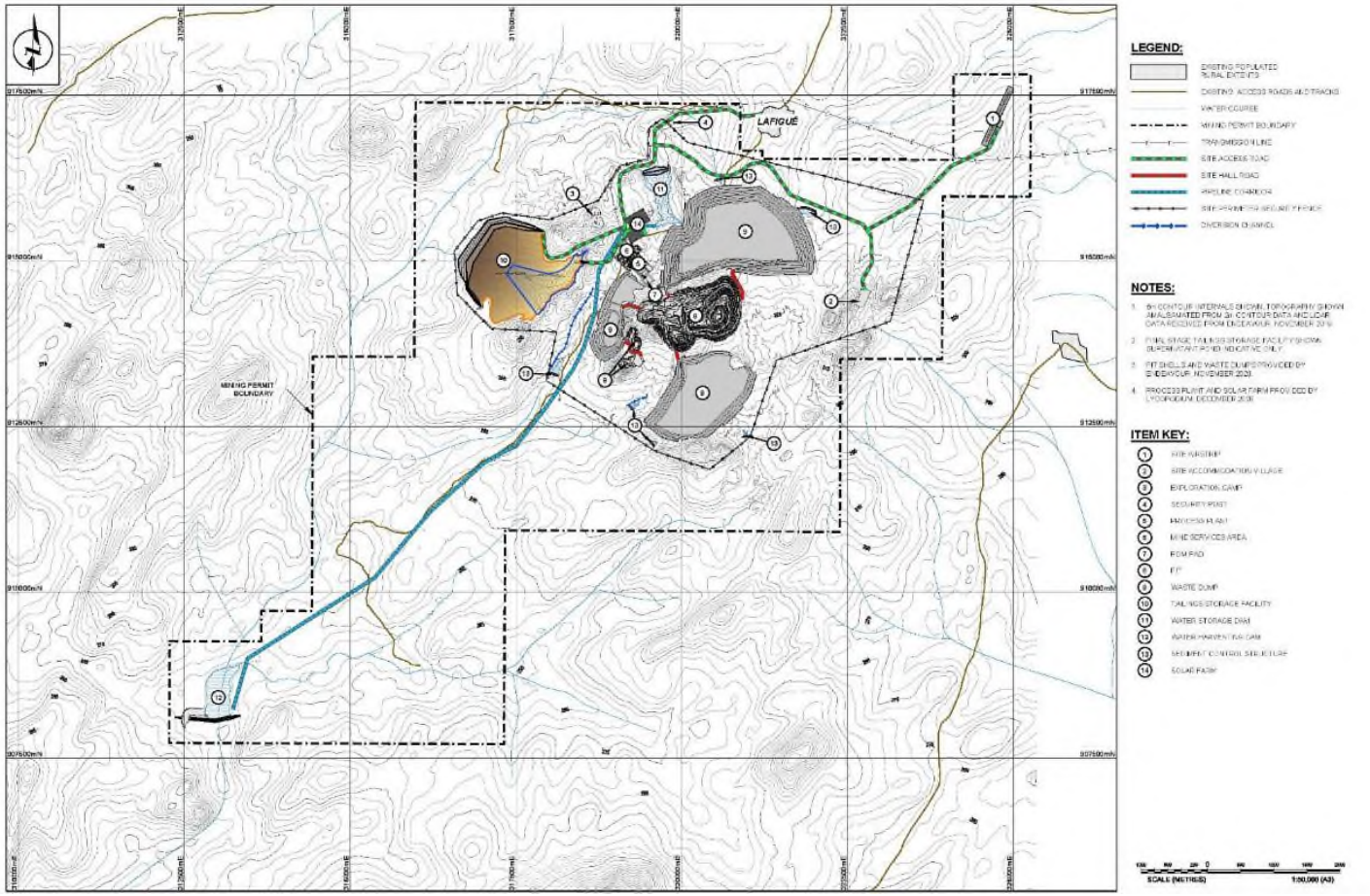
The site access road and major on-site roads will be all weather, free draining, and consist of two 3.5 m width running lanes with a one metre shoulder each side of the road, for a total formation width of nine metres. This will allow access for the delivery of equipment, materials and services to site.

A hybrid grid/solar option has been selected for the Project. The closest connection point to grid power is a connection to the national grid at the Dabakala Substation approximately 28 km from site. La Société des Energies de Côte d'Ivoire (CI-E) owns the National Interconnected Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government.

A connection will be facilitated by extending the existing 225 kV bus, adding a 225 kV transmission line feeder bay, construction of a 225 kV single circuit lattice tower transmission line, and constructing a substation at Lafigué site. The Lafigué Substation would be owned and operated (CI-E) and the Project would take a 225 kV tariff metered feeder, install a 225/11 kV transformer in their substation and take an 11 kV feeder to the plant main 11 kV switchboard.

Grid power will be supplemented by an onsite 7 MW solar array. The Plant is estimated to have a connected load of approximately 23.4 MW with a Maximum Demand of 16.6 MW and an expected energy consumption of 133 GWh/y.

Figure 1.12.1 Lafigué Project Layout



A permanent accommodation village will be located approximately two kilometres east of the plant site and approximately four kilometres by road. A short separate access road off the main access road to the plant site will be developed for access to the village. In addition, a security barracks for 60 staff will be located at the entrance to the site. Unskilled staff and trades will be sourced from the local villages and will travel by bus to site on a daily basis.

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.

A packaged treatment plant will process daily sewage from up to 200 personnel at the accommodation village site. Effluent will discharge to a leach drain system. A separate, similar treatment plant will be located near the process plant.

A preliminary non-instrument design has been allowed for the Project airstrip, based on the nominated airstrip location 3.5 km north of the accommodation village. A fully automated weather station is included together with refuelling facilities.

The following facilities will be located in the fenced area adjacent to the process plant:

- Security gatehouse and change room.
- Main administration office.
- Clinic / first aid and emergency response building.
- Plant meeting and training room.
- Plant change room and ablutions.
- Plant offices and control room.
- Plant kitchen and diner.
- Reagent stores.
- Plant warehouse.
- Plant workshop.
- Electrical workshop.

In addition, provision has been made for two laboratories. The first will be located inside the plant security fence and will analyse process plant samples only. The second will be located outside the fence and will be used for exploration and mine grade control samples using fire assay.

A microwave tower and fibre network will provide outside communications and in-plant data communications. Endeavour will engage a local mobile service provider with regard to extending coverage to the mine site.

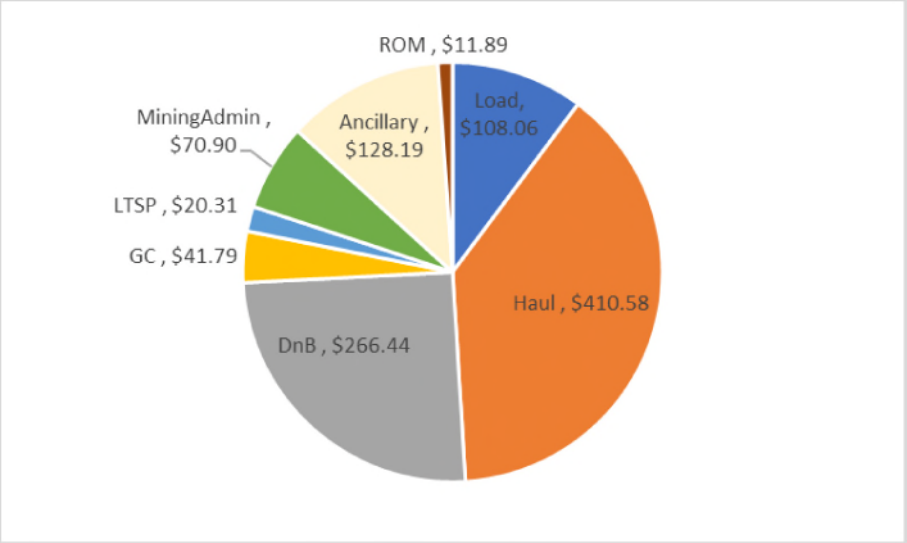
Provision has been made for a Mine Services Area (MSA), with the design and construction of this to be coordinated with the successful mining contractor.

1.13 Operating Costs

1.13.1 Mining Operating Costs

Endeavour calculated mining costs based on haulage paths provided by Snowden. Figure 1.13.1 shows the life of mine operating costs by area.

Figure 1.13.1 LOM Mining Operating Costs by Area, US$ M



1.13.2 Process Plant Operating Costs

The processing operating costs for treating fresh and oxide (oxide/transition) ores have been estimated by Lycopodium, as presented in Table 1.13.1. The plant will process either 100% fresh ore (possibly with a small portion of oxide) or 100% oxide ore. The operating costs have been developed based on a nominal capacity of 3.0 Mtpa of fresh ore and 3.5 Mtpa of oxide ore. The life of mine (LOM) blend operating costs were estimated based on the weighted average of the fresh and oxide costs.

The fresh and oxide ore fixed and variable cost components can be used, along with the plant feed schedule, to estimate the operating cost variation on an annualised basis as well as the LOM operating cost.

Mining costs, excepted as noted, are excluded from the costs presented in Table 1.13.1. Crusher feed and stockpile rehandle costs are included under mining operating costs.

Table 1.13.1 Lafigué Processing Cost Summary (US$, 4Q20, ±25%)

COST CENTRE	Fresh		Oxide / Transition		LOM Blend	
Proportion of LOM	92%		8%		100%	
Plant Feed t/y	3,000,000		3,500,000		3,039,614	
	US$/y	US$/t	US$/y	US$/t	US$/y	US$/t
Power	9,845,239	3.28	6,292,359	1.80	9,563,750	3.15
Operating Consumables	9,224,661	3.07	9,745,105	2.78	9,265,895	3.05
Maintenance	3,473,286	1.16	2,635,940	0.75	3,406,944	1.12
Contract Laboratory	1,342,574	0.45	1,342,574	0.38	1,342,574	0.44
Process & Maintenance Labour	6,134,550	2.04	6,134,550	1.75	6,134,550	2.02
Total Processing	**30,020,309**	**10.01**	**26,150,529**	**7.47**	**29,713,713**	**9.78**
Fixed Component US$/y	**13,394,496**		**11,936,603**		**13,278,990**	
Variable Component US$/t		**5.54**		**4.06**		**5.41**
Administration Labour	9,748,187	3.25	9,748,187	2.79	9,748,187	3.21
G&A Costs	8,828,943	2.94	8,828,943	2.52	8,828,943	2.90
Total G&A	**18,577,130**	**6.19**	**18,577,130**	**5.31**	**18,577,130**	**6.11**
Total Process Plant	**48,597,439**	**16.20**	**44,727,658**	**12.78**	**48,290,842**	**15.89**
Fixed Component US$/y	**31,971,626**		**30,513,732**		**31,856,119**	
Variable Component US$/t		**5.54**		**4.06**		**5.41**

1.14 Capital Costs

The Project capital cost estimate (CCE) summarised in Table 1.14.1 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, Owners costs and HV power supply.

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 based on rates obtained at the end of January 2020:

- 1 AUD = 0.7100 USD
- 1 EUR = 1.1833 USD
- 1 GBP = 1.3097 USD
- 1 ZAR = 0.061 USD

Table 1.14.1 Capital Cost Estimate Summary (US$, 3Q20, +20/-10%)

Main Area	Capital (US$ M)
Treatment Plant	74.4
Reagents & Plant Services	14.4
Infrastructure	70.4
Mining	50.3
Construction Distributables	23.5
Subtotal	**233.0**
Management Costs	24.5
Owners Project Costs	34.2
Working Capital	4.8
Subtotal	**296.5**
*Contingency	37.7
DFS Costs	3.8
Project Total	**338.0**

1.15 Project Execution

The Project is still at an intermediate stage of development and there are several key activities to be planned and executed going forward. These include the following:

- Completion of a Feasibility Study and update to the NI 43-101 document.
- Completion of associated studies and final reporting for an Environmental and Social Impact Assessment (ESIA).
- Application for a Mining Permit.
- Completion of any compensation negotiations and agreements.
- Board Approval of full capital commitment.
- Detailed process plant and infrastructure design and procurement.
- Power supply negotiations, procurement and construction.
- Early site works including roads, power line, bulk earthworks, accommodation village, water harvesting and storage facilities.

- Process plant and site infrastructure construction.
- Appoint and mobilise mining contractor.
- Operational readiness planning.
- Process commissioning.

Endeavour intends to engage a suitable Engineering, Procurement and Construction Management (EPCM) contactor for design and construction management of the process plant and infrastructure, which will then be handed over to an Owner's operating team. An experienced mining contractor will develop the mine infrastructure under the management of an Owner's mine technical team.

The Owner's team will be progressively expanded to widen its skills and knowledge base to meet the needs of the Project. The team of full- and part-time personnel will manage both the onshore and offshore activities of the principal EPCM contractor and specialist subcontractors as well as providing specialist technical input into the Project design.

Estimated key dates and durations to complete future studies, engineering and construction, leading to the first Project gold pour are summarised in Table 1.15.1.

Table 1.15.1 Lafigué Project Timeline

Activity	Month Start	Duration (Months)	Month Complete
PFS			Jan 2021
DFS	Mar 2021	7	Oct 2021
FEED and Long Lead Items Phase	Oct 2021	19	Jul 2023
Funding Commitment Early Works			Nov 2021
Operating Licence Approved			Aug 21
Full Funding Commitment			Sep 2021
Approval to Place Long Lead Orders			Nov 2021
Detail Engineering Design	Apr 2022	12	Feb 2023
Procurement	May 2022	16	Aug 2023
Bulk Earthworks (Site Works)	Mar 2022	12	Mar 2023
Construction	May 2022	20	Dec 2023
Plant Dry Commissioning	Oct 2023	3	Dec 2023
Plant Wet Commissioning	Nov 2023	3	Jan 2024
First Gold Pour			Jan 2024
Plant Ramp-up	Jan 2024	3	Apr 2024
Project Completion			Apr 2024

1.16 Environmental

The Mining Code of Côte d'Ivoire stipulates that the award of an operating license must result from a technical and economic feasibility study and be the subject of an environmental and social impact assessment (ESIA).

To comply with this provision of the Mining Code and in application of the Environmental Code, Endeavour Mining has engaged the services of Cabinet ENVAL Côte d'Ivoire, an environmental consulting company approved by the National Environment Agency (ANDE) to carry out the ESIA of the Lafigué area.

Endeavour has engaged ENVAL and Trust International consultants to undertake environmental studies and investigations for the Project. These studies commenced in 2019 and will culminate in the publishing of the Environmental and Social Impact Assessment (ESIA) in 2020.

These studies began with the exhaustive inventory of the physical, human and social environment of the area covering the Exploration License as well as the future area of the Mining Permit to be requested, considering the standards of the Environmental Code and of the Performance Standards of the International Finance Corporation (IFC).

The social component of the study was undertaken by Trust International who developed the stakeholder engagement plan to identify the actors involved in, and likely to contribute to the future Project.

The studies are being conducted using appropriate scientific methods with data systematically collected on that basis. The outcomes of these studies will provide the basis for a draft ESIA report which will be submitted to the National Environmental Agency (ANDE) for assessment.

Public inquiries and consultation will be undertaken with local communities and prefectures, with meetings open to elected officials, the customary authorities, the administrative authorities of these departments, and the concerned populations for ten working days, and endorsed by two reports with the opinion of the populations on the project.

Upon completion and approval of the technical review of the ESIA, an order of authorisation will be signed by the Minister of the Environment.

Once the operating license is obtained, a mining convention will be submitted to the Ministry of Mines and analysed by the cabinet meeting, which will authorize its signing.

The collection of field data was carried out between 16 November 2019 and 16 August 2020 to cover the two seasons (dry and rainy seasons).

1.17 Economic Analysis

A financial model has been built by Endeavour to include the relevant study results to estimate and evaluate project cash flows and economic viability. The evaluation method considers 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 US$ 1,500/oz on a flat line basis from the commencement of production, the pre-tax NAV5% (Net Average Value ('NAV') at a 5% discount rate) is US$ 663 million and the IRR is 38%. The life of mine average cash cost per ounce is US$ 684.

The LOM average all-in sustaining cost (ASIC) is US$ 831/oz.

1.18 Conclusions and Recommendations

Work undertaken to date is sufficient to support PFS level design and cost estimating; however, several recommendations for future work have been identified by QPs and Endeavour which require further evaluation prior to, or as part of any subsequent FS phase, with key points summarised below.

1.18.1 Mineral Resource and Geology

- A structural study is required to improve the understanding of mineralisation controls.
- The future campaign should test the junction of Lafigué Centre – Lafigué North.
- Update the Resource with an independent consultant to validate the Resource.
- Review the need for additional drilling to provide a Measured Resource category in the initial mining phase areas.

1.18.2 Environmental and Social

In compliance with Ivorian regulations and IFC performance standards, an Environmental and Social Impact Assessment ((ESIA) has been completed. Two major phases remain for the ESIA validation, which include the public inquiry, held on 15 to 30 December 2020, and a technical meeting (validation workshop) will be organised in January 2021, and be held in Abidjan.

Both of these phases are organised and facilitated by the National Agency for the Environment (ANDE) and Cabinet consultants. All technical ministries involved in the process of granting an Environmental License for a mining project will take part in this meeting. A formal meeting report will be signed by all technical Ministries, and based on this, it is anticipated that the Ministry of Environment will grant Endeavour the Environmental License no later than mid-March 2021.

This Report will be used by Endeavour in February 2021 to introduce the Request for the Lafigué Mining License to the Ministry of Mines and Geology.

Water, noise and dust monitoring baseline data collections is required to start as soon as possible. In addition, a weather station will be installed in the new exploration camp. Other programmes and plans to be developed and implemented by Endeavour.

These would include compensation and livelihood restoration programme (LRP) to enable affected people to recover alternative livelihoods, as well as a range of community development programmes will need to be implemented to build and subsequently reinforce Endeavour's social license to operate.

1.18.3 Pit Geotechnical

Dedicated geotechnical investigation boreholes would be required to be drilled to increase the levels of confidence and reliability of current geotechnical assessments. The new boreholes will target zones without either previous geotechnical holes or resource holes. Currently available drillhole data and analysis does not include the 2020 Resource expansion. Geotechnical data obtained from future drilling should be cross-checked with findings and assumptions presented in this preliminary assessment report.

1.18.4 Mining

The current LOM plan and Mineral Reserve is based entirely on an Indicated Mineral Resource. To increase the confidence (on both grade and tonnage) of the resource, Endeavour intends to complete a drilling programme to increase a portion of the Lafigué deposit that will be mined in the early part of the mine life to Measured category.

While the level of details provided as part of the PFS concerning the mine operating costs is sufficient, early engagement is recommended with one or more potential Mining Contractor experienced in West African operations to obtain firm a budget price estimate for the open pit mining operations.

1.18.5 Processing and Metallurgy

While the metallurgical testwork conducted to date is sufficient to support the PFS design, additional work has been recommended, and is currently being planned with results required expected to be available prior to FS.

Currently available testwork data from completed programs does not include significant ore zones added to the resource in 2020, and therefore it is unknown whether testwork performed and results achieved are representative of the expanded resource.

In addition to this, the PFS design includes a HPGR as opposed to the previous Scoping Study (SS) designed SABC circuit. Therefore, design criteria for the HPGR are currently based on assumptions, typical and database information only. Additional recommended testwork will provide the necessary input FS level design.

1.18.6 Infrastructure

Geotechnical Data

No site geotechnical evaluations have been undertaken to date to confirm the site conditions for major structures including the process plant and TSF. This should be undertaken as soon as final locations are fixed for in particular larger load areas such as crushers, HPGR, comminution area, larger tanks and TSF.

Power

Sufficient capacity exists within the Existing 225 kV network at Dabakala substation to service the Project's power demand. An acceptable power supply of sufficient quality and fault level is seen as a key component to the success of the Project. Discussions should commence early with CI-E to introduce the project and timing of construction, understand the grid supply limitations and expansion plans, and to discuss the Project's planned connection to the grid.

Roads

Road conditions vary considerably in Côte d'Ivoire. A detailed transport route survey should be undertaken during FS and prior to detailed design. An upgrade to the existing approximately 15 km section of the road / track from Bounadougou on the B412 has been allowed for in the estimate and is planned to be implemented early by Endeavour to improve access and transport safety to the Project.

2.0 INTRODUCTION

2.1 Terms of Reference

Endeavour Mining Corporation (Endeavour) is developing the Lafigué deposit (the Project) located in Côte d'Ivoire. Endeavour is a Canadian reporting issuer, and through its exploration subsidiary La Mancha Côte d'Ivoire (LMCI), has an 100% interest in the Fetekro exploration permit, PR329, (the Permit), in which the Project is located.

Endeavour engaged Lycopodium Minerals Canada Pty Ltd (Lycopodium) to coordinate and prepare an independent Technical Report on the Lafigué 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 historical and recent data generated by Endeavour. Endeavour has engaged several specialist consultants and information from reports prepared by previous independent consultants which have been utilised in the compilation of this Report. Section 27 provides a list of references relied upon in preparation of this Report.

2.3 Issuer

Endeavour was incorporated on July 25, 2002 under the laws of the Cayman Islands under the name Endeavour Mining Capital Corp. On July 16, 2008, it changed its name to Endeavour Financial Corporation and then on September 14, 2010 it changed its name to Endeavour Mining Corporation (hereinafter "Endeavour" or the "Company" or the "Group").

Endeavour's Registered Office is located at 27 Hospital Road, George Town, Grand Cayman, Cayman Islands. The Corporate Office is located at 5 Young Street, London, United Kingdom and its Executive Office is located at 7 Boulevard des Moulins, Monaco. Endeavour's ordinary shares are listed on the Toronto Stock Exchange ("TSX") under the symbol "EDV" and quoted in the United States on OTCQX International under the symbol "EDVMF".

Endeavour is a senior gold producer and the largest in West Africa, with producing mines in Côte d'Ivoire and Burkina Faso and a strong portfolio of advanced development projects and exploration assets on the highly prospective Birimian Greenstone Belt across West Africa.

As of December 31, 2020, Endeavour's mining operations comprised the Ity Gold Mine and the Agbaou Gold Mine located in Côte d'Ivoire; and the Houndé Gold Mine, the Karma Gold Mine, the Mana Gold Mine and Bongou Gold Mine located in the Republic of Burkina Faso ('"Burkina Faso'"). Endeavour also has an extensive exploration portfolio (totalling approximately 6,000 km^2) in Côte d'Ivoire (including the Fetekro Property with the Lafigué Project), Burkina Faso, the Republic of Mali (including the Kalana Project) and the Republic of Guinea.

2.4 Contributing Consultants

Contributing consultants and co-authors to sections of this Technical Report are summarised in Table 2.5.1 following. Relevant industry experience by consultant, is summarised below, whilst the respective QPs are as detailed in Section 2.5.

- Lycopodium Minerals Pty Ltd (Lycopodium) has provided engineering and project management services to the international mining industry for over 20 years, has extensive experience in West Africa, and was the lead consultant for the Technical Report
- Snowden Optiro (Snowden) is a resource and mining consulting group based in Perth, Western Australia. Snowden provided the mining design and the Mineral Reserve estimate.
- Knight Piésold Pty Limited (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 Technical Report for the Project, with the participation of specialised consultants. Table 2.5.1 following provides a list of QPs, and the respective sections/subsections for which they took responsibility. If a QP took responsibly for an entire section, it is given that they also took responsibility for the associated subsections. Further definition is provided in the 'Certificates of Qualified Persons' (Section 28). All consultants contributed to Section 27.

For the reasons stated in Section 2.7, only one of the QPs listed below has been to site, namely Silvia Bottero of Endeavour.

Table 2.5.1 Qualified Persons and QP Section / Subsection Responsibilities

Name	Position	Company	Sections/Subsections
David Gordon (FAusIMM)	Group Process – Manager	Lycopodium	1.2, 1.10, 1.11, 1.12.1, 1.13.2, 1.14, 1.15, 1.18.5, 1.18.6, 2, 3.1, 12.1, 12.5, 12.9.1, 13, 17, 18.2.1, 18.7, 18.8, 18.9, 18.10, 18.11, 18.12, 21.1, 21.1.2, 21.2, 21.3, 21.5, 24, 25.1, 25.4, 25.5, 26.5 and 27
Allan Earl (AWASM, FAusIMM)	Executive Consultant	Snowden Optiro	1.8, 1.9, 1.13.1, 1.18.3, 1.18.4, 3.1, 12.4, 15, 16, 21.1.1, 25.3, 26.3, and 26.4
David Morgan (CPEng, MAusIMM)	Managing Director	Knight Piésold	1.12, 3.1, 12.9.2, 12.9.3, 12.9.4, 12.9.5, 18.1, 18.2.2, 18.2.3, 18.2.4, 18.3, 18.4, 18.5, 18.6, 26.6.1, 26.6.2 and 26.6.3
Silvia Bottero (Pr. Nat. Sci.)	VP Exploration, Côte d'Ivoire	Endeavour	1.3, 1.4, 1.5, 1.6, 1.18.1, 4, 5, 6, 7, 8, 9, 10, 11, 12.2, 23 and 26.2
Kevin Harris (CPG)	VP Resources	Endeavour	1.7, 12.3, 14 and 25.2,
Patrick Pérez (P.Eng)	Director Technical Studies	Endeavour	1.1, 1.16, 1.17, 1.18, 1.18.2, 3.2, 3.3, 12.6, 12.7, 12.8, 19, 20, 22, 26.1, 26.6.4 and 26.6.5

2.6 Effective Date and Declaration

The effective date of this Technical Report is December 31, 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

The status of site visits, up to the 'effective date' of this Technical Report by QPs and other relevant parties, is as defined herein. The onset of the COVID-19 pandemic prior to, and over the course of the PFS study, halted/restricted travel to site. Additional detail by QP, is described more fully in the accompanying QP Certificates (Section 28.0)

- Ms Silvia Bottero (Pr. Nat. Sci.) was La Mancha's Exploration Manager for the Project from 2014 to 2016 and Endeavour's Exploration Manager from 2017 to August 2020 and over this period, was at the Site extensively. In September 2020, Ms Bottero was appointed Endeavour's VP for Exploration for Côte d'Ivoire and has continued her involvement with the Site, with her last site visit being December 14 to 17, 2020.

- Mr Mark Giddy, Principal Process Engineer with Lycopodium (not a listed QP in this Technical Report), undertook a site visit to Lafigué in May 2019 to supervise the metallurgical sample selection and packing for shipment. Lycopodium has previously completed studies and projects in Côte d'Ivoire and is familiar with the country / region.

- Mr Patrick Pérez (P.Eng), the Technical Director of Studies at Endeavour Mining, has had no prior involvement with the Project and has not visited site due to travel restrictions associated with COVID 19.

- Mr Kevin Harris (CPG), VP Resources at Endeavour Mining, has been involved with the Project since 2017, where he was responsible for developing NI 43-101 compliant Mineral Resource estimates in 2017, 2018, 2019 and 2020. Mr Harris has not visited site because of travel restrictions associated with COVID 19.

- Mr David Gordon (FAusIMM), Lycopodium's Group Process Manager, has had no prior involvement with the Project and has not visited the site because of travel restrictions associated with COVID 19.

- Mr David Morgan (CPEng, MAusIMM), a Managing Director of Knight Piésold, has previously completed studies and projects in Côte d'Ivoire and is familiar with the country/region and has routinely travelled to other Endeavour assets. Mr Morgan has not visited the site because of travel restrictions associated with COVID 19.

- Mr Allan Earl (AWASM, FAusIMM) an Executive Consultant with Snowden Optiro has previously completed studies and projects in Côte d'Ivoire and is familiar with the country/region and has routinely travelled to other Endeavour assets. Mr Earl has not visited site because of travel restrictions associated with COVID 19.

2.8 Units and Currency

Unless stated otherwise, Le Système International d'Unités (SI) units have been used throughout the reports. (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 (US$), 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
Dt	Dry 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	General and Administration
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
mamsl	Metres above mean 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
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 autogenous 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 Project Ownership, Mineral Tenure, Permits and Agreements

External QPs, namely David Gordon, Allan Earl and David Morgan, have relied on information provided by Julie Blot [Legal and Public Affairs Counsel] of Endeavour [via email, 10 December 2020], relating to Property ownership, mineral tenure, permits and agreements (Section 4.0). The information has not been independently verified and no opinion is offered in this area.

3.2 Taxes and Royalties

The QP (Mr Patrick Pérez) has fully relied upon information provided by Julie Blot [Legal and Public Affairs Counsel] of Endeavour [via email, 10 December 2020], pertaining to the taxation and royalty regime applied in this Project. The information used is based on the current Côte d'Ivoire Mining Code, and is discussed and applied in Sections 4.0 and 22.0 respectively

3.3 Environmental and Permitting

The QP (Mr Patrick Pérez) has obtained information regarding the environment, social and permitting status of the Project from data and a report supplied by experts retained by Endeavour. The QP has fully relied upon information derived from such experts, including information contained in:

- Etude d'impact Envrionnemental et Social (EIES) – Projet d'exploitation industrielle d'or a Lafigué, dans le département de Dabakala, dated February 2021.

The information is used in Section 22.0.

4.0 PROPERTY DESCRIPTION AND LOCATION

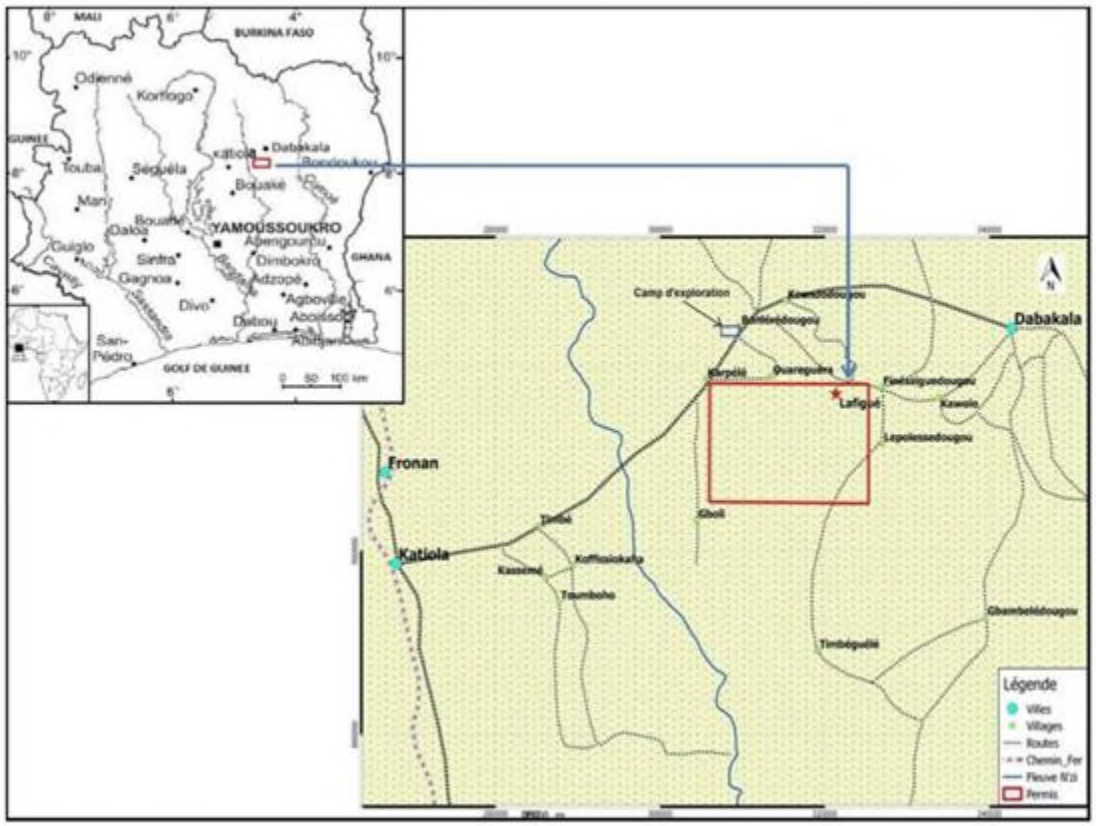
4.1 Property Location

The Fetekro Property is located in the northern part of Côte d'Ivoire, 550 km north of Abidjan and 90 km and 45 km northeast of Bouaké and Katiola towns respectively. The Fetekro exploration permit (also known as the prospecting right), number PR 329, covers an area of 249.5 km².

The Lafigué deposit sits on the eastern side of Fetekro Property. The deposit is open at depth towards the southeast and south and covers an area which is two kilometres long and one kilometre wide.

Access to the Fetekro Property is via sealed regional highway (B412), which traverses in an east/west direction approximately 15 km to the north of the project site. An unsealed track extends to Lafigué. The general proximity to regional villages and towns and the access route is shown in Figure 4.1.1.

Figure 4.1.1 General Project Location and Access



4.2 Issuer's Interest

La Mancha Côte d'Ivoire (LMCI) is 100% held by Ity Holdings, a wholly-owned subsidiary of Endeavour Mining Corporation (as noted in Section 2) and has a 100% interest in the exploration permit.

Once an Industrial Operating Permit (i.e., a mining or exploitation licence) covering the Lafigué deposit is granted, Endeavour will be entitled to an 80% stake in 'Société des Mines de Lafigué', the exploitation entity which will mine the Lafigué deposit, while SODEMI and the Government of Côte d'Ivoire will each retain 10% of the remaining 20% stake respectively. Endeavour, through its exploration subsidiary La Mancha Côte d'Ivoire, will retain the full ownership of the remaining area of the Fetekro exploration licence until such time as it is converted into mining licences, subject to the discovery of new deposits, or the permit expires.

4.3 Mineral Tenure

Fetekro's original prospecting permit (PR 57) covered an area of 2,600 km² and was valid for gold and all substances except hydrocarbons. The permit was initially awarded to SODEMI according to Decree N° 93-215 of February 3, 1993 for a period of three years

This permit was first renewed for a period of two years pursuant to Arrêté N° 044/MRMP/DMG of May 28, 1996, with a subsequent reduction in surface area to 1,300 km².

A second renewal of the permit, valid for two years from May 28, 1998, was granted to SODEMI pursuant to Arrêté N° 054/MRMP/DM of July 23, 1999, with a subsequent reduction in surface area to 614 km².

Finally, an exceptional renewal of the permit, valid for a period of three years, was granted to SODEMI pursuant to Arrêté N° 014/MME/DM of April 30, 2001, with a subsequent reduction in surface area to 307 km².

The Côte d'Ivoire civil war between 2002 and 2010 did not facilitate the SODEMI-COMINOR partnership to continue exploration activity on PR 57 as planned within the framework of the exceptional renewal. Hence, PR 57 expired and a new permit, PR 329, was awarded to SODEMI by Decree N° 2013-410 of June 6, 2013, with a surface area of 335.5 km².

The PR 329 permit was renewed by Arrêté N°090 MIM DGMG of July 11, 2017 for a period of three years from June 6, 2016. The area covered was consequently reduced by 25% to 249.5 km².

A second renewal was issued by Arrêté N°00008/MMG/DGMG of January 13, 2020 for a period of three years from June 6, 2019, with no reduction in surface area applied, thereby maintaining an area of 249.5 km².

Recently, pursuant to a sale of the exploration permit agreement, PR 329 was transferred from SODEMI to LMCI by order Ministerial Order N° 00174/MMG/DGMG of December 18, 2020.

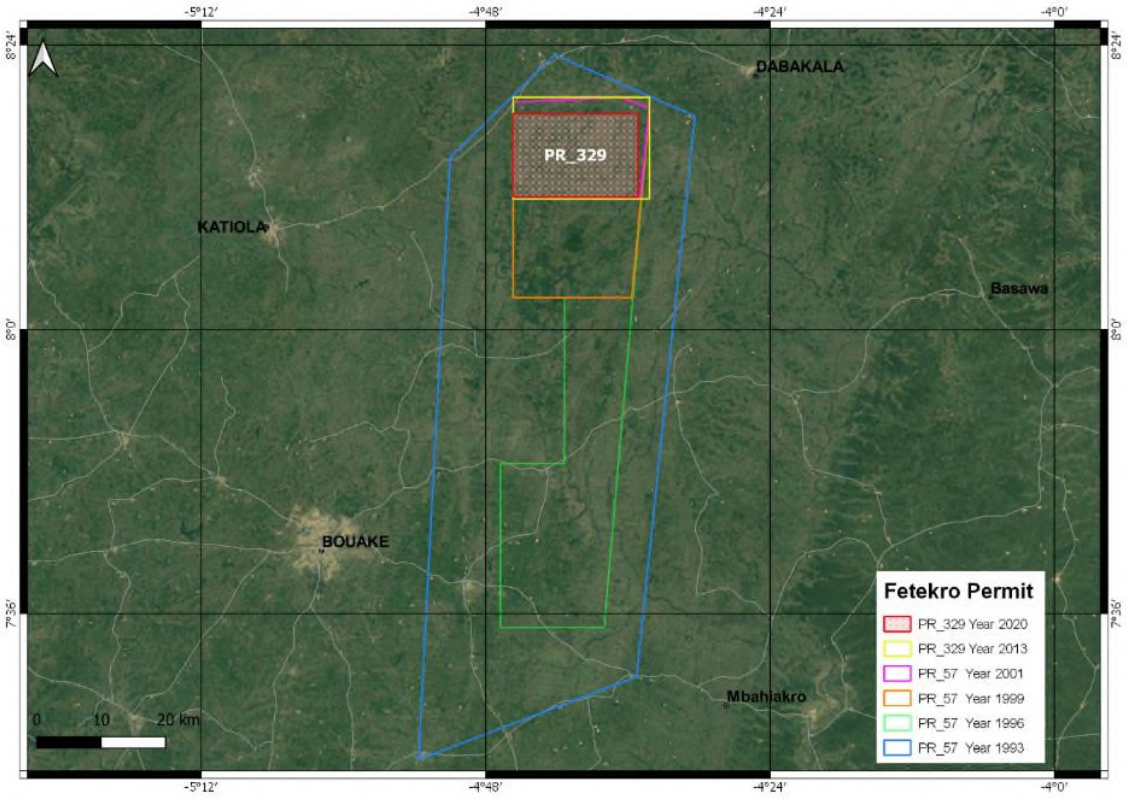
The Fetekro Permit covers an area of 249.5 km² and is defined by the coordinates shown in Table 4.3.1.

Table 4.3.1 Fetekro Exploration Permit Coordinates

Point	Latitude North	Latitude East
A	8° 18′ 12.00″	4° 45′ 40.00″
B	8° 18′ 12.00″	4° 35′ 10.00″
C	8° 11′ 12.00″	4° 35′ 10.00″
D	8° 11′ 12.00″	4° 45′ 40.00″

Figure 4.3.1 following, illustrates the evolution of the Fetekro exploration permit since 1993 to 2020.

Figure 4.3.1 Evolution of the Fetekro Exploration Permit



According to the Ivorian 2014 Mining Code, an exploration permit is granted by a Presidential Decree. The exploration permit is valid for an initial period of four years, it may be renewed for two consecutive periods of three years and a final exceptional renewal for a two-year period, provided the titleholder complies with the rights and obligations, as set out under the Mining Code. At each renewal, at least 25% of the original area must be relinquished; however, the titleholder can decide to maintain the full surface rights by paying an 'option fee'.

The Mining Code gives the exploration permit holder, the exclusive right to explore for the minerals requested, on the surface and in the subsurface, within the boundaries of the permit.

The application for a permit must be filed along with a programme of exploration works that the applicant commits to carrying out during the first four years of permit validity, along with the budget for said programme. The titleholder of a permit must start exploration works within the area of interest, not later than six months from the date of validity of the permit and must commit to work diligently over this period.

4.4 Industrial Operating Permit (Mining Permit)

The exploration permit gives the holder the exclusive right, at any time, to apply to convert the exploration permit into an Industrial Operating Permit ('Permis minier' or mining permit).

The Industrial Operating Permit grants its holder the exclusive right to exploit mineral deposits within the specified perimeter, defined by the mining permit boundary coordinates Table 4.3.1. Mining permits are valid for an initial period established on the life of mine presented in the feasibility study (with a maximum of 20 years) and are renewable for periods up to 10 years, until the reserves have been depleted.

The application for a mining permit must be filed along with a feasibility study and a plan for the development and exploitation of the deposits, which shall include, inter alia, an environmental and social impact study and a mitigation and rehabilitation plan.

4.5 Royalties and Agreements

4.5.1 Royalties

The mining permits are subject to a 10% carried ownership interest to the benefit of the Government of Côte d'Ivoire. In addition, once a Mining Convention is signed and a mining permit is awarded by the Government, a graduated royalty based on gold price is payable as follows:

- Below a gold price of US$1000/oz, a royalty of 3% is payable.
- At a gold price between US$(1000 and 1300)/oz, a royalty of 3.5% is payable.
- At a gold price between US$(1300 and 1600)/oz, a royalty of 4% is payable.
- At a gold price between US$(1600 and 2000)/oz, a royalty of 5% is payable.
- Above a gold price of US$2000/oz, a 6% royalty is payable.

In addition, a 0.5% local community levy is payable on gross revenue.

4.5.2 Agreements

In 1996, an exploration, development and operating agreement was entered into by SODEMI, the title holder, and GENCOR (through its Ivoirian company GATRO-CI) on the Fetekro property (the Exploration Agreement). According to this Exploration Agreement, the exploration campaigns were done by GENCOR through its Ivoirian company (GATRO-CI) and SODEMI/BRGM/La Source.

In 1999, the Compagnie Minière Or (COMINOR) took over La Source participation and the GATRO-CI contractual commitments under the Exploration Agreement.

In 2000, COMINOR was transferred to Compagnie Générale des Matières Atomiques (COGEMA) which was incorporated into the La Mancha Group in 2006, via a reverse takeover of La Mancha by Compagnie Française de Mines et Métaux (CFMM), a wholly owned subsidiary of the AREVA group.

In 2013, AREVA sold its gold assets in Côte d'Ivoire to a private fund.

In 2014, La Mancha Côte d'Ivoire S.a.r.l (LMCI) was incorporated in Côte d'Ivoire as a 100% subsidiary of COMINOR and took over the exploration activities of COMINOR managed so far by its Ivoirian branch, COMINOR CI, including COMINOR contractual commitments under the Exploration Agreement.

LMCI became a full subsidiary of Endeavour in November 2015. LMCI is fully owned by Ity Holdings, a wholly owned subsidiary of Endeavour.

On November 26, 2020, a sale of the property was entered into between LMCI and SODEMI. This sale agreement states that the Exploration Agreement will terminate on the date of the Ministerial Order transferring PR 329 to LMCI. SODEMI will be entitled to 10% of the future Société des Mines de Lafigué, operating entity which will operate the Lafigué Project.

4.6 Environmental Permitting

LMCI retained the services of ENVAL to undertake studies and environmental research as part of the Lafigué Project. These studies began in November 2019 and resulted in the publication of the Environmental and Social Impact Assessment (ESIA) at the end of September 2020.

The ESIA was submitted to a government committee for validation on January 20, 2021 and the study received a favourable opinion. An Arrêté will be delivered and will be part of the documents requested for the Mining Permit application.

All phases of development on the Property will be subject to environmental regulations, including but not limited to, the maintenance of air and water quality standards, land rehabilitation and waste disposal. ENVAL and Endeavour are not aware of any environmental liabilities to which the Property may be subjected to; however, ENVAL have not investigated any environmental liabilities that may arise from previous work.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Côte d'Ivoire is located along the coast of western Africa and is bordered to the east by Ghana, to the north by Burkina Faso and Mali, and to the west by Guinea and Liberia. The Property is located in the north-central part of Côte d'Ivoire, approximately 500 km from Abidjan, and close to the village of Lafigué. The Property is located within the Dabakala Department in the Hambol Region of the Vallée du Bandama District.

The Lafigué Project sits five kilometres from its namesake village, in the northeastern corner of the Fetekro Property.

Côte d'Ivoire is one of Africa's more developed countries, with comparatively excellent infrastructure, including a reliable national electricity grid and highway network, both of which will be utilised by the Project.

Access to the Property is via a sealed regional highway (B412), which traverses in an east/west direction, approximately 15 km to the north of the project area. An unsealed track extends from Koundodougou (7 km west of Boniérédougou) to Lafigué. The general location of the Property and its, proximity to local towns and other Endeavour Mining Côte d'Ivoire Projects is shown in Figure 5.1.1 and Figure 5.1.2 following.

Figure 5.1.1 General Property and Project Location

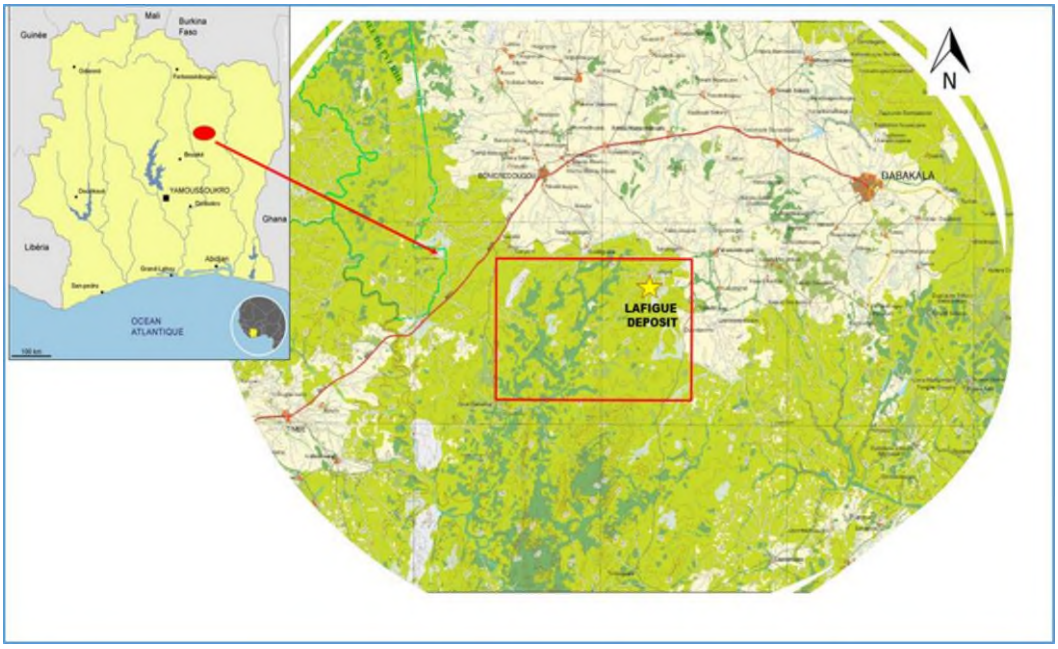
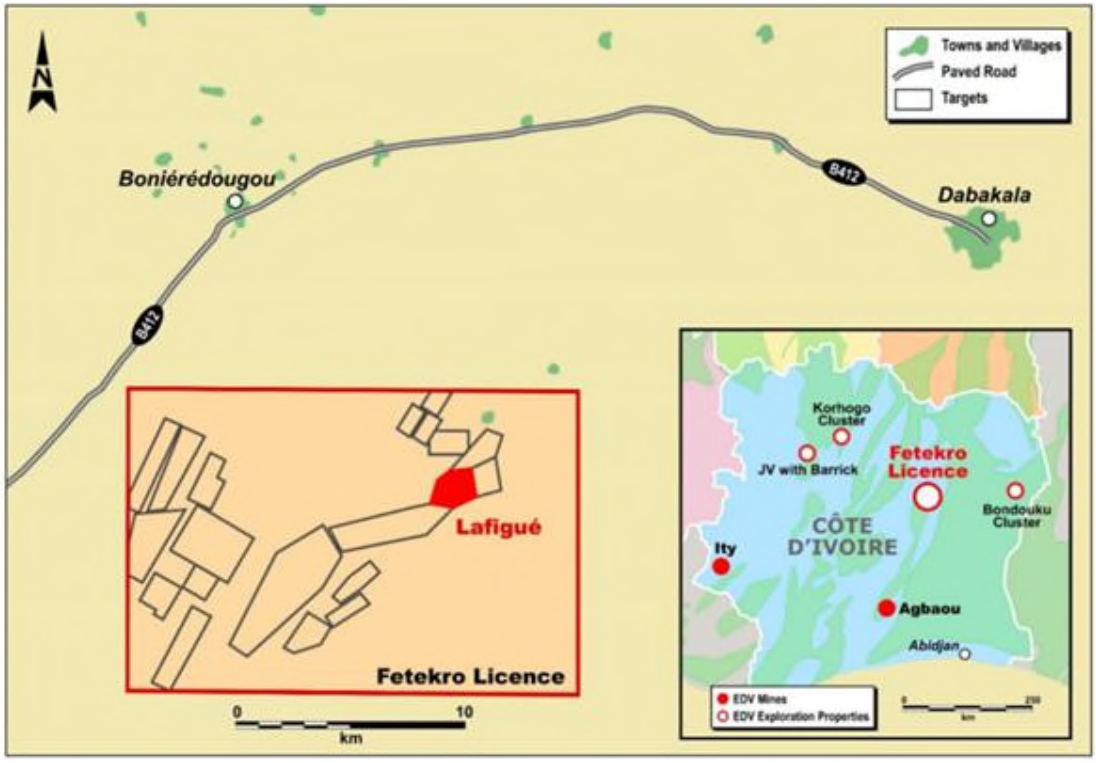


Figure 5.1.2 Property and Endeavour Côte d'Ivoire Projects



5.2 Climate

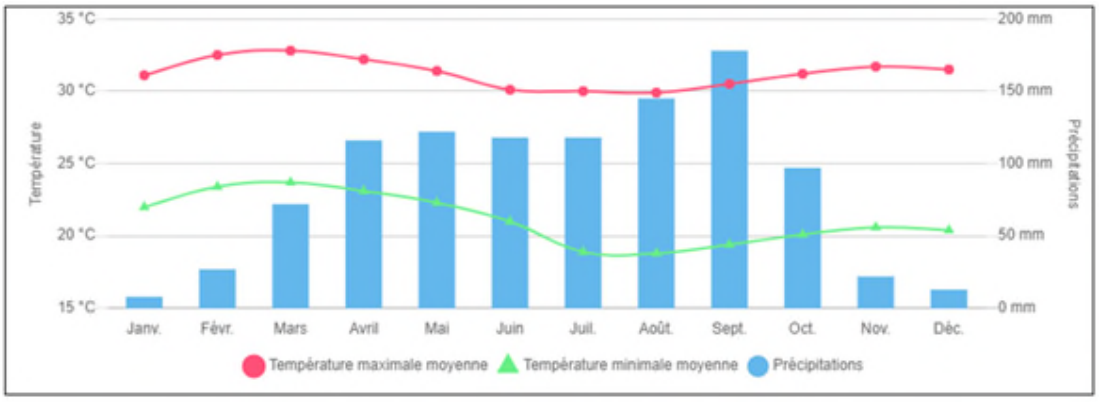
5.2.1 Temperature and Rainfall

The Property is subject to the Sudanese climate, a tropical transition regime between the semi-arid sub-Saharan zones and the humid tropical zones of the Gulf of Guinea, with maximum rainfall in July, August and September and a period of drought in December, January and February.

The climate is regulated by the displacement of the Intertropical Front (ITF). Annual rainfall is almost the same for the whole region, with an annual average of 800 mm of rainfall and a monthly average of 120 mm for the months of June to September; however it is noted that a cyclical phenomenon, about every five years, causes this annual average rainfall to vary between 800 mm and 900 mm (Geomines, 1982). The annual average temperature is 28°C.

The dry season from November to February is dominated by the harmattan, a dry and cool wind that blows from the Sahelian areas from north to south. It is characterised by low temperatures at night.

Figure 5.2.1 Average Monthly Temperature and Rainfall in the City of Katiola



Source: SODEXAM, 2018

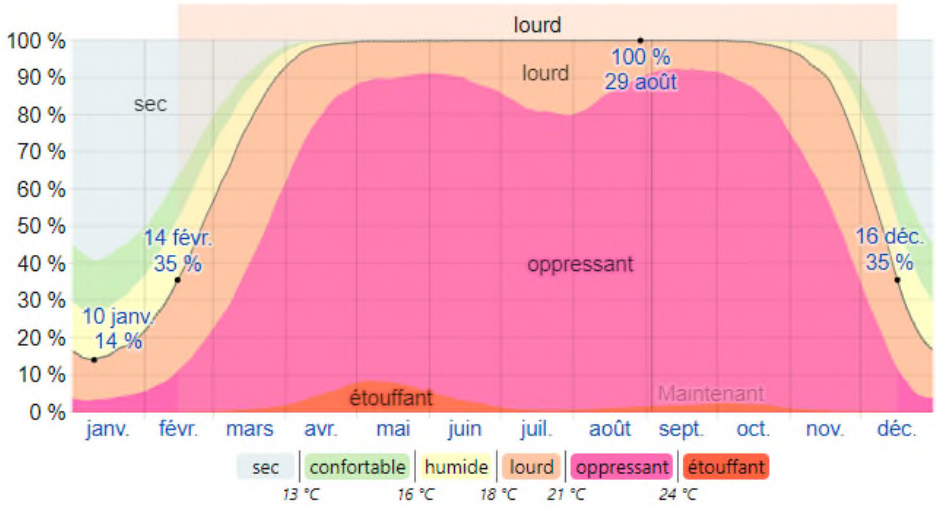
5.2.2 Humidity

The comfort level is estimated according to the humidity on the dew point, as it determines whether sweat will evaporate from the skin, causing the body to cool. Lower dew points are felt as a drier environment and higher dew points as a more humid environment.

Unlike temperature, which usually varies widely between day and night, dew points vary more slowly. So, although the temperature can drop at night, a heavy day is usually followed by a heavy night.

The Dabakala region of Côte d'Ivoire experiences extreme seasonal variation in the perceived humidity. The less comfortable time of the year lasts ten months, from mid-February to mid-December, feeling muggy, oppressive, or stuffy at least 35% of the time. Figure 5.2.2 following, illustrates the humidity levels by month, over a typical year.

Figure 5.2.2 Percentage of Time in Various Humidity Comfort Levels



5.2.3 Wind

In the Fetekro area, winds typically originate from the southwest, blowing from 220°N to 240°N, with wind run speeds varying between 1 and 4 m/s.

Table 5.2.1 Average Wind Run Speed (m/s)

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Average
2014	2.0	2.2	3.0	2.4	3.0	3.0	2.0	2.0	2.0				2.4
2015	2.0	2.0		2.0	2.0	2.0	2.0	1.0	2.0	2.0	2.0	2.0	1.9
2016	1.0	1.0	1.0	2.0		3.0	2.0	1.0	2.0	3.0	2.0	2.0	1.8
2017	2.0	2.0	2.0	2.0	2.0	3.0	2.0	2.0	2.0	3.0	3.0	3.0	2.3
2018	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	3.0	3.0	2.0	2.2

Source: SODEXAM, 2018

Table 5.2.2 Direction of Prevailing Winds (degrees)

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Average
2014	240	220	220	240	240	240	230	280	220				237
2015	240	240		240	220	220	240	220	220	220	220	220	227
2016	280	280	240	220		220	240	240	280	240	220	200	242
2017	240	240	280	240	280	240	240	280	280	220	220	220	248
2018	20	220	220	240	200	200	180	220	220	220	220	240	202

Source: SODEXAM, 2018

5.3 Local Resources and Infrastructures

The city of Dabakala is the nearest relatively important town (in size) to Lafigué where basic services and supplies are available. With a population of 56,000 (2014 census, Wikipedia), it is serviced by a hospital, hotels and several shops.

Specialised services and equipment have to be sourced in Abidjan or Yamoussoukro. Mine labour would likely be sourced from local and expatriate personnel. Currently power is envisioned to be provided by the national power grid (refer to Section 18).

Development of the Project will require an upgrade or construction of new infrastructure including an access road (from the main National Road), water and power supply, waste dumps and tailings storage areas, air strip, as well as plant and camp facilities.

5.4 Physiography, Topography, Elevation and Vegetation

The Lafigué Project site is located in the Bandama Valley, characterised by generally flat topography, with a series of small northeast striking hills in the northwest and northeast corners of the Fetekro Property and on the southern boundary. Topographic elevations range from 200 to 400 mamsl and the project area is traversed by a north-south striking valley.

The Fetekro Property vegetation is characterised by savannah forest with grass.

Land use is largely agricultural in nature and includes cotton and cashew plantations, peanuts, rice, yam and corn fields and cattle farming.

6.0 HISTORY

6.1 Historical Exploration Work

The first geological mapping campaign and exploration started in the Fetekro area in 1935 (BUMIFORM).

In 1962, the BRGM and the SODEMI did some more work in the area of interest to determine if any gold or bauxite deposits could be found. At the same time, geophysical airborne surveys were undertaken (1965-1968 BRGM/Canadian Aero Mineral Surveys Ltd and 1973-1976 CIDA/Côte d'Ivoire/Kenting Ltd).

In 1996, GATRO-CI undertook classical stream sediment and then soil geochemical sampling, pits, trenching and limited drilling (14 diamond drill (DD) holes for 1,446 m and 37 reverse circulation (RC) holes for 1,549 m) in the Fetekro area. Four main targets were identified, Lafigué A, B, C and D. The Lafigué deposit as the basis of this PFS, corresponds to the historical Lafigué A target.

In 1999, COMINOR started an exploration programme on the Fetekro Property to find new resources to replace Angovia's production after its depletion.

In 2002, COMINOR realised 1,803 m of rotary air-blast (RAB) drilling, 1,281 m of RC drilling and 461 m of DD drilling, which showed that the mineralisation is not continuous between Lafigué Centre and Lafigué Nord; and felsic dykes play a role in mineralisation control.

Field work was halted from 2002 to 2010 due to the Ivorian civil war.

In 2010, COMINOR drilled 11 RC holes for 1,109 m of and four DD holes for 396 m to check the mineralisation extension downdip in the Lafigué Centre area.

In 2014, LMCI drilled 23 DD holes (1,864 m) and 54 RC holes (4,634 m) to generate structural data and to check for an extension in the northern area of the Lafigué deposit.

All historical drill holes that could be found were re-surveyed in 2014 by the company Environnement Technologie Côte d'Ivoire (Envitech-CI) using a differential GPS; 65 RAB holes and three RC holes from 1997 (numbers R2087, R2997 and R30B97) could not be found

In 2015, LMCI sponsored a Lidar survey of the area.

All work undertaken prior to 2017 and the takeover by Endeavour in late 2016 is considered as historical; however, the work completed by LMCI between 2014 and 2016 was undertaken by the same team as used by Endeavour. On the basis that the Exploration Team at the Property has been stable over the last seven years and the procedures and protocols have been developed and refined in-house over time, regardless of ownership, the LMCI and Endeavour data is considered to be compatible, comparable and of equal stature by the Qualified Person. The 1996 GATRO-CI and 2002 and 2010 COMINOR drill holes are utilised in the Mineral Resource Estimate and were subjected to the validation and verification steps as stipulated in Section 12.

A summary of historical work is presented in the tables below.

Table 6.1.1 Fetekro Historical Drilling Summary

Year	Area	Drill Type	No. Drill Holes	Metres
1997	Lafigué	RC	37	1,549
1997	Lafigué	DD	14	1,446
2002	Lafigué	RAB	75	1,803
2002	Lafigué	RC	32	1,281
2002	Lafigué	DD	11	461
2002	Target 8	RAB	19	294
2010	Lafigué	RC	11	1,109
2010	Lafigué	DD	4	396
2014	Lafigué	RC	54	4,638
2014	Lafigué	DD	23	1,864
Total			**280**	**14,548**

Figure 6.1.1 Location of Lafigué Historical Drill Holes

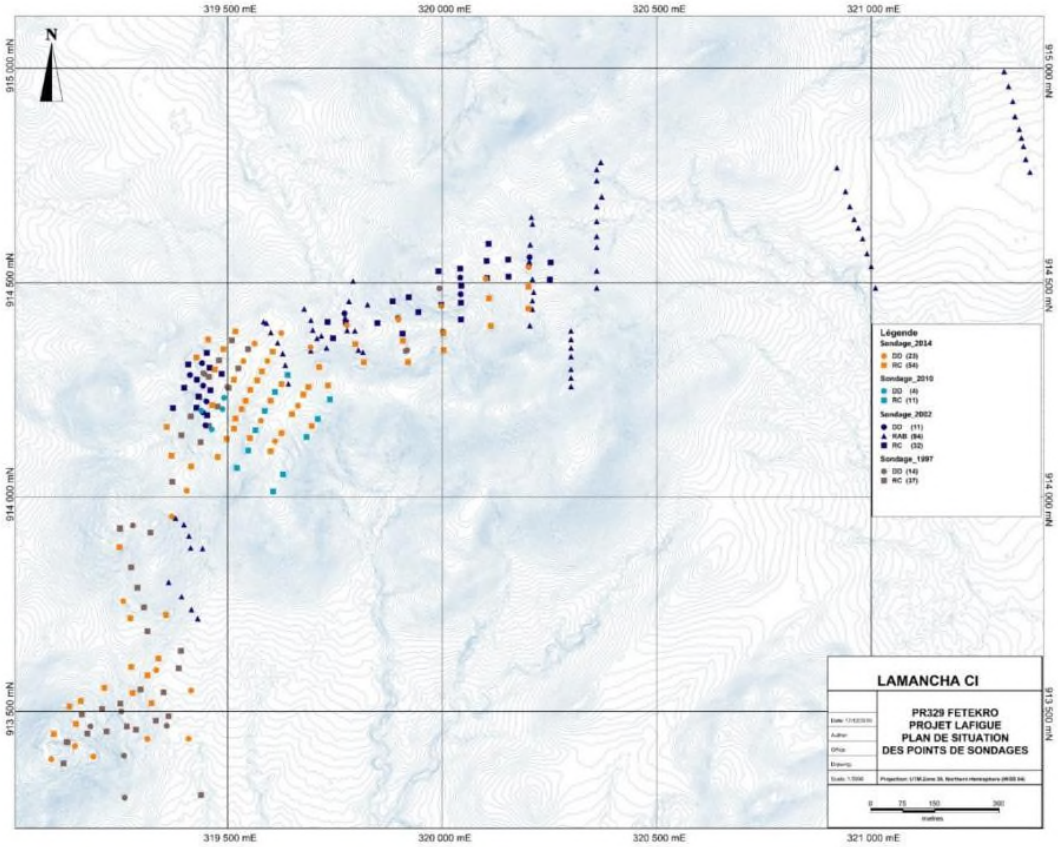


Table 6.1.2 Summary of Historical Exploration Activity

Lafigué Project			
Company	**Year**	**Work**	**Main Results**
GENCOR	**1994**	**Aéromagnétiques Magnetic airborne survey interpretation** Kenting Pty Ltd mission 1973-76	
GATRO CI	**1994**	**Stream sediment geochemistry** 2,143 samples taken (1 éch. / 1.2 km²) 1,970 samples analysed for gold 1,006 ICP analyses	4 anomalies • Sandérékro • Tibéguélé • **Lafigué** • Sarakakro No correlation between gold and other elements
GATRO CI	**1995**	**LAFIGUÉ anomaly, soil geochemistry** 200 m x 100 m spacing (locally 100 m x 50 m) 1,862 samples Au analyses (detection limit : 5 ppb) **Rock samples** 35 samples taken and Au Analysed	4 gold anomalies • **Lafigué A**: main one, 1,700 m long and 250 m wide • Lafigué B: 2 values > 2 g/t Au but not validated → work stopped on this area • Lafigué C: close to a granodiorite, 300 m long and 100 m wide • Lafigué D: on a Flat area → work stopped on this area • Lafigué A: 3.4 and 6 g/t Au on rocks coming from previous work
GATRO CI	**1996**	**LAFIGUÉ A anomaly** **Soil geophysics** **(magnetics, VLF-EM, PP)** • Mag. and VLF: 130 km lines with a 100 m x 30 m spacing • Induced polarisation: 4 profiles (7.4 km) **Trenches** 26 trenches (3,800 m) 2,154 samples taken (horizontal grooving) 264 samples (vertical grooving)	The mineralisation seems to be low grade and disseminated (0.4 to 0.8 g/t Au). The highest grade (up to 50 g/t Au) are associated with quartz tourmalines veins. The channel samples done in the overburden shows mineralisation with an average grade of 3.8 g/t and 1.3 m average length. Some mineralisation was intersected by the vertical grooving (1.3 m @ 3.8 g/t).
GATRO CI	**1997**	**Diamond drillholes campaign** 14 drillholes (1,447 m)	Gold is associated with weathered schists and mylonites from metavolcanic rocks. The rocks are locally sheared. The mineralisation is present as coarse gold in and around quartz veins with sericite and tourmaline, or, deeper, associated with oxidised sulphide.
GATRO CI	**1997**	**LAFIGUÉ "A"Anomaly** **Hole campaign:** 100 m x 100 m spacing then 50 m x 50 m 313 holes dug (depth from 0.45 m to 8.7 m) 1,018 grooved samples **RC drilling campaign** 37 RC drilled (1,549 m) **Mineral processing test** 3 oxidised samples were tested : • One from the eluvial quartz • One from quartz vein • Disseminated gold Only the samples were recovered by gravity (Knelson concentration) and the rejects were cyanide leached	**Target:** First calculation of the quartz potential close to the surface and in the saprolite **Results (with 1.57 density)** • tonnage (t): 978 797 • grade (g/t Au): 2.54 • Au content (t): 2.5 **Target:** Test mineralised quartz continuities on the Centre and South Area **Results:** 1) Gravity recovery: • (84 to 88)% • (86 to 89)% • (17 to 35)% 2) Combined cyanide leaching and gravity recovery: • eluvial quartz (96 to 99)% • quartz vein: (98 to 99)% • disseminated gold: (64 to 83)%
COMINOR	**2001**	**METALLURGICAL TESTWORK** Inventory of RC 1997 duplicate samples. 40 samples were sent to SGS for analyses. "A" and "Z" composites for insertion during the RAB and RC drilling campaign.	Mineral processing test (bottle test) of 36 samples from overburden mineralisation (12 holes, 21.35 m)
COMINOR	**2002**	GATRO CI data validation and resources estimation (A. Grove and H. Sanguinetti – COGEMA, Vélizy) LAFIGUÉ "A" anomaly:	
		Drilling campaign:	
		1) RAB: 94 holes, 1,803 m, 12 profiles	1) Target: Continuity between the Centre and the North area, search for extension, "C" anomaly checking. Results: No continuity, 200 m Eastern extension, no positive results on C. 2) Target: Centre area extension, increase the resources. Results: discontinuity on the centre zone linked with a felsic dyke N125°E, North: discontinuity also explained by a felsic dyke N160°E.
		2) RC: 32 holes, 1,281 m, 17 profiles 3) DD: 11 holes, 461 m, 8 profiles	3) Target: Main mineralisation confirmation, density measurements. Mean density: saprolite (2.0 t/m³), oxidised zone (2.1 t/m³), transition zone (2.5 t/m³), sulphide zone (2.8 t/m³)
COMINOR	**2010**	Drilling campaign: 1) RC: 11 holes, 1,109 m 2) DD: - 4 holes, 396.30 m	Target: -Check the mineralisation extension downdip on the centre area, check the extensions
LMCI	**2014**	Drilling campaign: 1) RC: 54 holes, 4,634 m 2) DD: 23 holes, 1,864 m	Recognise mineralisation intersected during previous campaigns and test their extensions.
LMCI	**2015**	DGPS survey of collars LIDAR survey done by AOC	

6.2 Historical Resource Estimates

The following historical resource estimates were not reported publicly, or within any regulatory framework. The estimates were intended for the internal use of the companies involved only. These resource estimates have not been evaluated by the QPs for this Report, are not current and are reported for background information only.

A QP has not done sufficient work to classify the following historical estimates as Mineral Resources or Mineral Reserves under an industry standard code (e.g. CIM, JORC, SAMREC, etc.). The Issuer is not treating the historical estimates as Mineral Resources or Mineral Reserves. They are historical resource estimates which have been replaced by the Mineral Resource Estimate as reported herein.

An initial preliminary polygonal resource estimation was done by GATRO-CI in February 1998 (Table 6.2.1). No top cut was applied. GATRO-CI identified 1,595 Mt at 2.87 g/t Au in the overburden and in the oxidised zone.

Table 6.2.1 GATRO-CI Preliminary Resource Estimate for Lafigué (1998) – Cut-off Grade 1 g/t Au

	Overburden	Oxidised Zone	Sulphide Zone	Total
Tonnage (t)	979,000	616,000	632,000	**2,227,000**
Grade (Au g/t)	2.54	3.40	6.7	**3.9**
Gold contents (t)	2.5	2	4.2	**8.7**
Gravity recovery	84 to 89%	17 to 35%	Not tested	
Gravity recovery and cyanide leaching	96 to 99%	64 to 83%	Not tested	

COGEMA reviewed the GATR-CI 1998 resource estimation using a geostatistical method (Grove 2002). The Lafigué deposit was split into the South, Centre and North zones (Table 6.2.2.).

Table 6.2.2 COGEMA Resource Estimate for Lafigué (2002) – Cut-off Grade 1 g/t Au

	Total			Oxidised Zone			Sulphide Zone		
	Tonnes (kt)	Au g/t	Au kg	Tonnes (kt)	Au g/t	Au kg	Tonnes (kt)	Au g/t	Au kg
North	940	1.6	1,520	590	1.6	590	350	1.7	604
Centre	1,599	3.0	4,846	559	2.5	1,377	1,040	3.3	3,469
South	268	1.5	412	268	1.5	268			
Overburden	662	2.6	1,688	662	2.6	1,688			
Total	**3,469**	**2.44**	**8,466**	**2,079**	**1.89**	**3,923**	**1,390**	**2.93**	**4,073**

A new resource estimation was done in 2003 by COGEMA with revised densities and an updated geological model (Table 6.2.3).

Table 6.2.3 COGEMA Resource Estimate for Lafigué (2003) – Cut-off Grade 1 g/t Au

	Total			Oxidised Zone			Sulphide Zone		
Zone	**Tonnes (kt)**	**Au g/t**	**Au kg**	**Tonnes (kt)**	**Au g/t**	**Au kg**	**Tonnes (kt)**	**Au g/t**	**Au kg**
North	914	1.94	1,778	655	1.81	1,218	299	1.87	560
Centre	1,157	3.23	3,732	550	2.49	1,373	606	3.89	2,359
South	315	1.50	473	315	1.50	473			
Overburden	1,288	2.30	2,956	1,288	2.30	2,956			
Total	**3,674**	**2.43**	**8,939**	**2,769**	**2.17**	**6,020**	**905**	**3.22**	**2,919**

6.3 Previous Mining Work

The Fetekro Property has not been subjected to commercial mining. Artisanal (also known as illegal orpaillage) mining has been observed within the Property, with quartz veins manually mined and hand sorted.

7.0 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Fetekro Property, which includes the Lafigué gold deposit, is located at the northern end of the Lower Proterozoic Oumé-Fetekro Greenstone Belt, a north-south elongated zone which is approximately 300 km long and 20 km wide. The greenstone belt is composed mainly of Birimian volcano-sediments, consisting mainly of mafic to intermediate metavolcanics, felsic metavolcanics, and clastic metasediments, that are bound and intruded by granitoid complexes. Known gold deposits such as Bonikro and Agbaou are hosted within the Oume-Fetekro Belt. Several large felsic intrusions also occur in the northern part of the Belt.

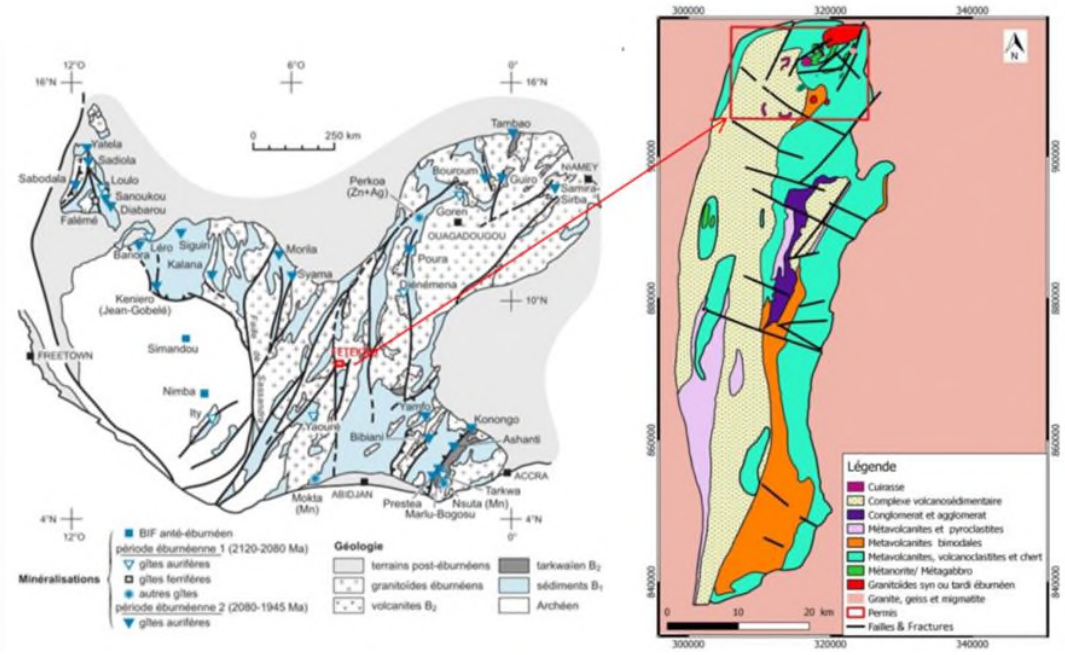
Past field studies suggest that the deformation history for the Oumé-Fetekro Greenstone Belt is poly-phased and includes (Mortimer, 1990; Leake, 1992; Houssou, 2013; Ouattara, 2015):

- D1 WNW-ESE compression resulting in the formation of NNE-trending upright to isoclinal folds (F1) with a penetrative axial-planar cleavage (S1).
- D2 WNW-ESE to NW-SE compression responsible for the regional structural grain (i.e. fold-and-thrust belt). This deformation phase correlates with the formation of inclined to upright, NNE- to NE-trending folds (F2), a penetrative axial-planar cleavage (S2), and moderate- to high-angle reverse shear zones.
- D3 NW-SE transpression marking a switch from a coaxial deformation regime to a non-coaxial regime with evolution from ductile to brittle-ductile behaviour. This deformation phase is associated with the dissection of the greenstone belt by north to NNE-trending sinistral shear zones and the formation of a NE-trending spaced crenulation cleavage (S3).
- D4 E-W shortening occurring at high crustal levels and responsible for the development of ENE-trending (dextral) and WNW-trending (sinistral) brittle strike-slip conjugated faults. This deformation episode is also associated with the formation of localised north-trending upright folds (F4) and axial-planar cleavage (S4).

Figure 7.1.1 Geological Map of the West Africa Craton (left) and Oumé-Fetekro Greenstone Belt (right) with the Location of the Fetekro Prospect

Modified after Milesi et al, 2004

The rectangular cutout is the Property outline



7.2 Local Geology

The Fetekro Property is hosted by a Birimian volcanic complex, affected by a transpressive deformation and intruded by granodioritic bodies and quartz-porphyry dykes.

The Property hosts the Lafigué deposit and 23 delineated exploration targets (14 "targets" and nine "anomalies"), Figure **7.2.1**. The Lafigué deposit was historically broken into three parts, Nord (North), Centre and Sud (South). The exploration targets are located on the eastern side of the Fetekro Property, and numbers 2, 5, 6 and 10 are contiguous to the Lafigué deposit. Eight of the nine anomalies are in the west of the Property (and referred to as the Western Anomalies (WA) (01 to 08) and one is the central area. Table 7.2.1 summarises the geology, size and exploration done and planned for the Fetekro exploration targets and anomalies.

Figure 7.2.1 Geological Map of the Fetekro Property with Deposits and Exploration Targets

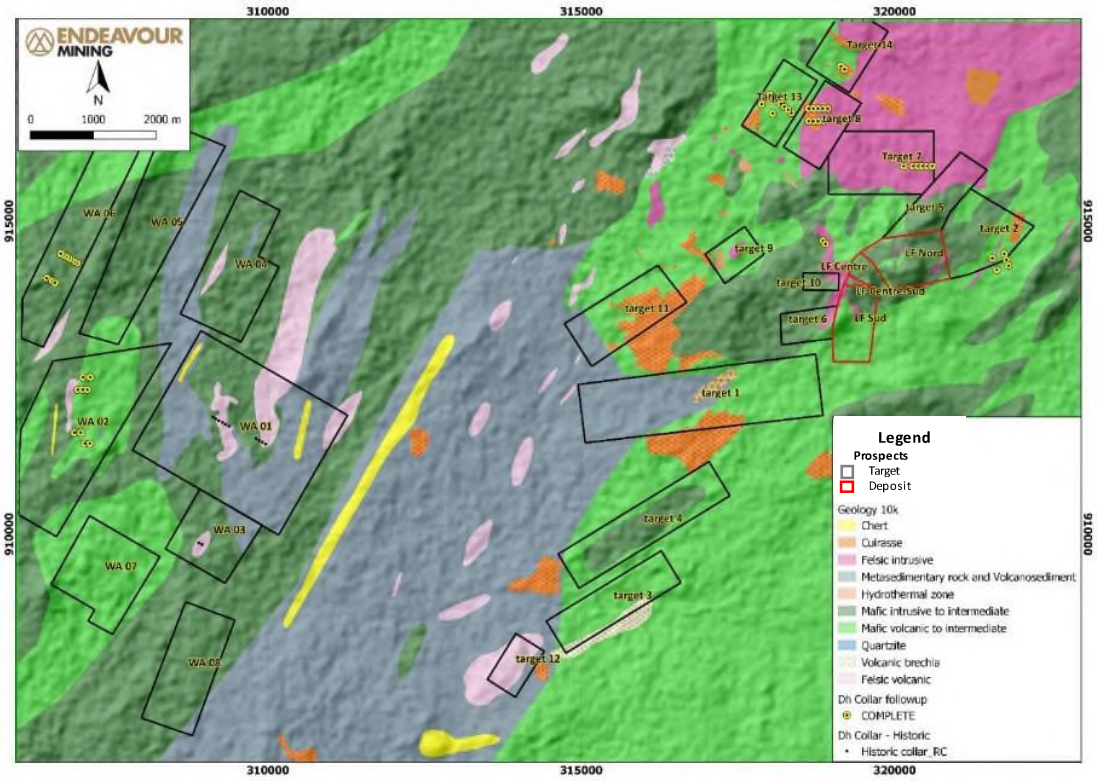


Table 7.2.1 Fetekro Deposits, Exploration Targets and Anomalies Summary

Name	Size	Host Rock	Work Completed	Comments
Lafigue	2,000 m by 1,000 m	Felsic-intermediate-mafic intrusives, felsic & mafic volcanics	Drilled	Historically broken into Nord, Centre & Sud parts.
Target 1	4, 000 m by 700 m	Intermediate to mafic intrusives, felsic and mafic volcanics, volcanosediments	Drilled	Target size based on historical soil anomaly that includes drainage from Lafigué deposit
Target 2	1,000 m by 200 m	Intermediate to mafic volcanics & Felsic and mafic intrusives	Soils, Geophysics, Drilled	IP anomaly (chargeability + resistivity) similar to Lafigué deposit but no significant mineralisation
Target 3	1,200 m by 200 m	Mafic volcanics & metasediments	Initial drill testing	
Target 4	2,400 m by 300 m	Mafic intrusives, mafic volcanics & metasediments	Soils, Geophysics, Drilled	Significant assays result confirm target potential
Target 5	1 200 m by 100 m	Mafic & felsic intrusives	Soils, Geophysics, Initial drill testing	
Target 6	700 m by 200 m	Mafic & felsic intrusives	Soils	
Target 7	500 m by 300 m	Tonalite	Initial drill testing	Artisanal site inside soil anomaly

Name	Size	Host Rock	Work Completed	Comments
Target 8	1,000 m by 200 m	Tonalite	Drilled	
Target 9	500 m by 100 m	Mafic volcanics	Initial drill testing	Soil anomaly and N050° geophysical structure
Target 10	250 m by 200 m	Mafic intrusive, tonalite and mafic volcanics	Initial drill testing	
Target 11	750 m by 200 m	Laterite duricrust	None	
Target 12	650 m by 150 m	Felsic and mafic volcanics, metasediments	None	
Target 13	750 m by 300 m	Microtonalite & intermediate-mafic volcanics	Initial drill testing	
Target 14	600 m by 100 m	Felsic intrusive & mafic volcanic	Initial drill testing	
WA01	2,000 m by 400 m & 2,000 m by 150 m	Mafic intrusives, intermediate volcanosediments, felsic volcanic	Soil, Geophysics, Initial drill testing	Two soil anomaly corridors, both with significant assays results
WA02	2 x 1,500 m by 200 m	Mafic volcanic, felsic volcanic and mafic intrusive	Soil, Initial drill testing	Significant assays result confirm target potential
WA03	2 x 800 m by 200 m	Mafic intrusives, felsic volcanics	Soil, Geophysics, Initial drill testing	Significant assays result confirm target potential
WA04	2,000 m by 100 m	Mafic intrusives, volcanosediments	Soil	
WA05	2,200 m by 150 m	Mafic intrusives, volcanosediments, mafic volcanics	Soil	
WA06	2,000 m by 500 m	Mafic intrusives, mafic volcanics, volcanosediments, felsic volcanics	Soil, Initial drill testing	Significant assays result confirm target potential
WA07	1,000 m by 300 m	Mafic intrusives, intermediate to mafic volcanics	Soil	
WA08	2,200 m by 150 m	Metasediments, mafic intrusives	Soil	
Central Area	4,000 m by 100 m	Metasediments, felsic volcanics	Soil	

7.3 Deposit Geology and Mineralisation

The Lafigué gold deposit has been traced over two kilometres along an ENE axis and over one kilometre down-dip. It is a Birimian volcanic complex predominately composed of mafic rocks, namely metagabbros / metanorites and metabasalts and felsic intrusives (granodiorite or tonalite) that lie in the western part of the Fetekro Property. This volcanic complex is affected by a transpressive deformation and intruded by granodioritic bodies and quartz-porphyry dykes. Regional foliation varies in strike from north-south to N070°, with gentle to intermediate / steep dips to the east and south (25° to 65°).

Mineralisation at Fetekro is mainly controlled by an ENE-trending brittle-ductile thrust fault dipping 15° to 45° SSE. Mineralisation is mainly hosted by a network of Qz-Cb-To-Py-Po±visible gold quartz veins within sheared and altered brittle-ductile deformation zones of various thickness (few metres to 10 m). The alteration assemblage comprises Bt-Ser±To±Chl±Cb (carbonates) and various amounts of disseminated pyrrhotite and pyrite (up to 5%).

The shear zones show a typical shear foliation (C-S) geometry, with the CS at some angle of the S (schistosity) fabric due to shearing. The veins are emplaced in the deformation corridors both along CS and S planes. The shear foliation and most of the veins dip shallowly to the south or SSE. The Qz-Cb-To veins occur in the shear zones, present crack-seal textures or have a breccia texture with strongly altered xenoliths of sheared host rock. They appear to dilate the sheared CS foliation of the host rock at various stages of the shearing history, which demonstrate their syn-deformation emplacement.

At Lafigué, a prominent deformation zone is typically located at the contact zone between a mafic intrusive (gabbro) and mafic volcanics, whereby the contact also occurs with a felsic intrusive at Lafigué Nord. The shear zones are better developed at, or near lithological contact zones, where competency contrasts favour the localisation of brittle-ductile shearing, permeability increase and enhanced hydrothermal fluid flow; however, these shear zones also appear in the core of massive intrusive or metavolcanic units.

Figure 7.3.1 Lafigué Geology and Selected Mineralised Intercepts

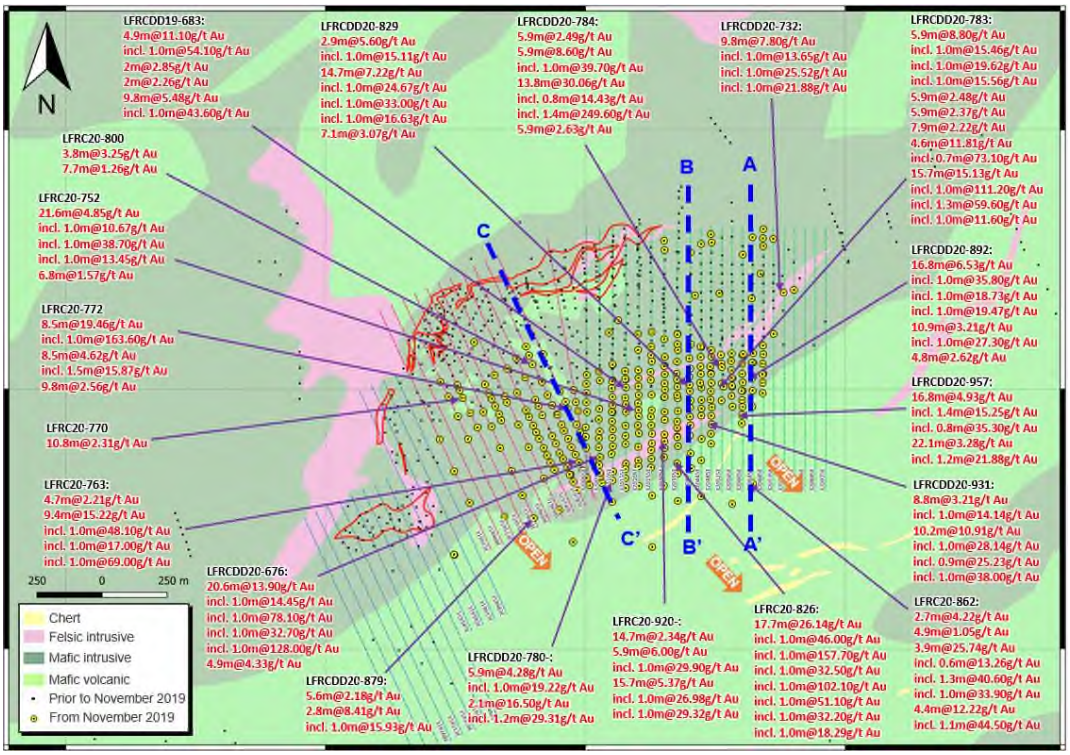


Figure 7.3.2 Lafigué Section A-A'

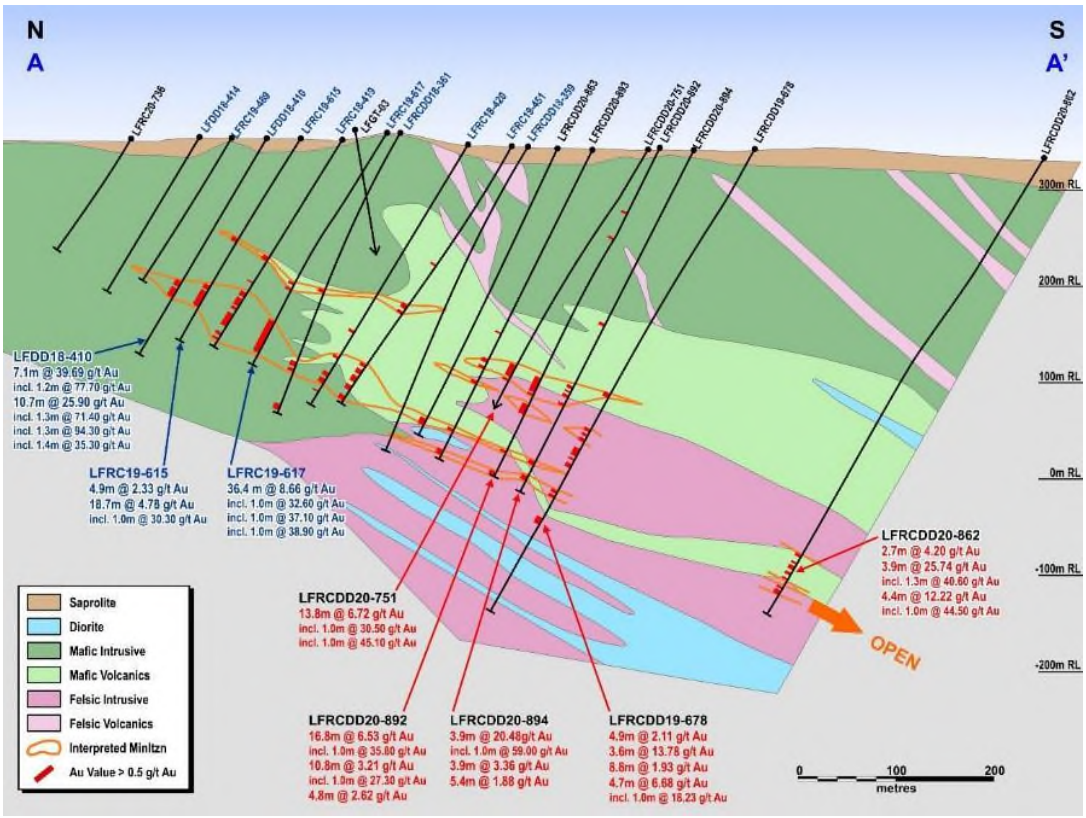


Figure 7.3.3 Lafigué Section B-B'

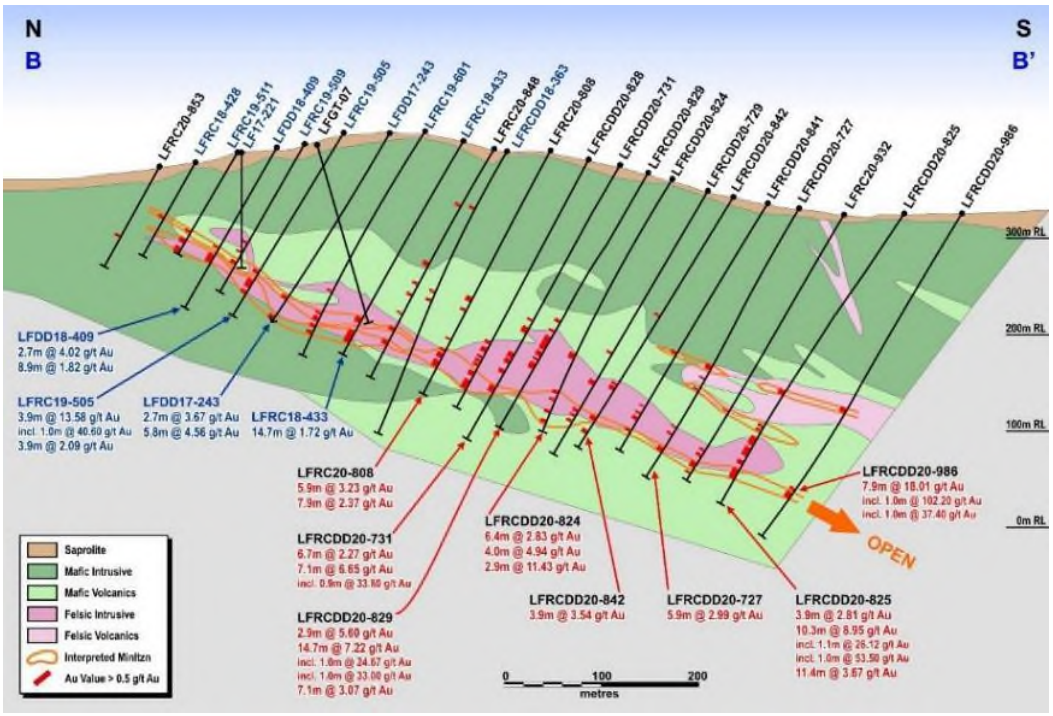
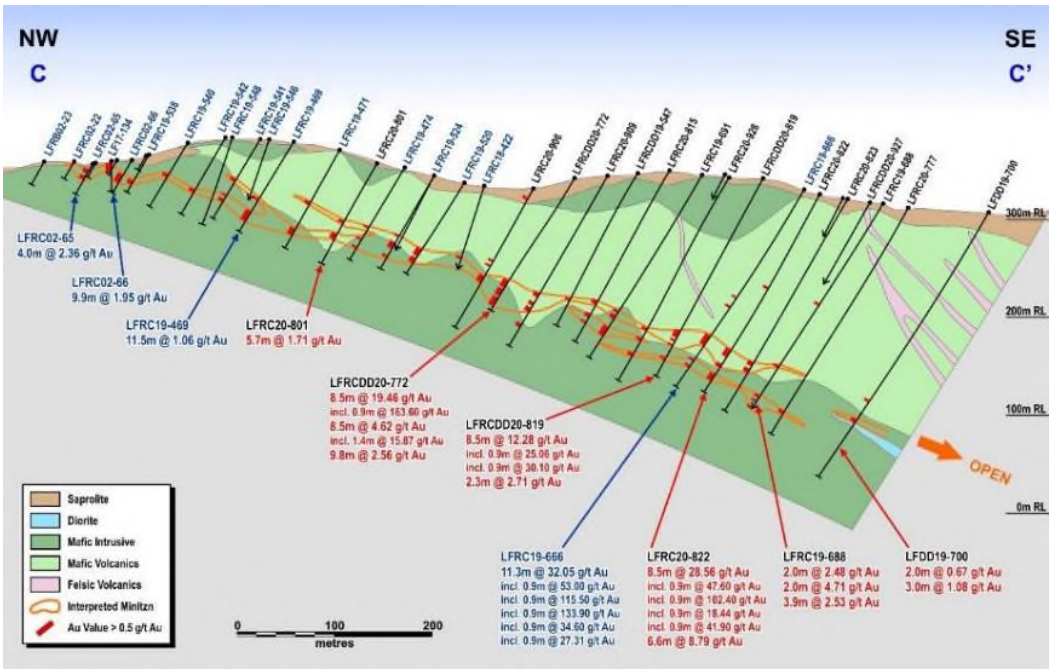


Figure 7.3.4 Lafigué Section C-C'



7.4 Mineralisation

Two types of mineralisation have been identified to date at Lafigué, namely:

- At surface - transported material composed of quartz blocs and pebbles in a clayish matrix. This type represents less than 5% of the Lafigué resource.

- Quartz vein hosted - This is the main type of mineralisation. The succession of hydrothermal events associated with C-plane fracture phases and thrusting resulted in the formation of two gold mineralisation-hosting, quartz-carbonate tourmaline-(chlorite-biotite-pyrrhotite-pyrite-gold) echelon extension vein generations. The textural and geochemical (minor elements and boron isotopy) features of the distinct tourmaline generations highlight the micro-scale record of fault-valve processes leading to the overall gold endowment of the Lafigué deposit. The lodes generally occur on lithological or structural discontinuities, typically at the granodiorite edges, on C-planes or re-opening early quartz-carbonate veins. At the deposit scale, the lodes show pinch and swell shapes, laterally and longitudinally with thicknesses of up to 40 m.

Quartz vein mineralisation may also be spatially related to the emplacement of intrusives. The gold mineralisation is mesothermal in origin and occurs as free gold in quartz vein stockworks and zones of silicification, associated with tourmaline, calcite, ankerite and pyrite. The gold mineralisation is found in linear zones in or near the contacts between two different rock types (metabasalt and metagabbro or metabasalt and intrusive). This contact shows evidence of shearing. Alteration is weak to severe depending on the development of the system.

8.0 DEPOSIT TYPE

Lafigué is a typical shear zone deposit of the West African granite-greenstone terrane. Lafigué gold mineralisation can be associated with the low-sulphide quartz gold deposit model of Laurence J. Drew (Drew, L., 2003, Low-Sulfide Quartz Gold Deposit Model. U.S. Geological Survey, p1-24).

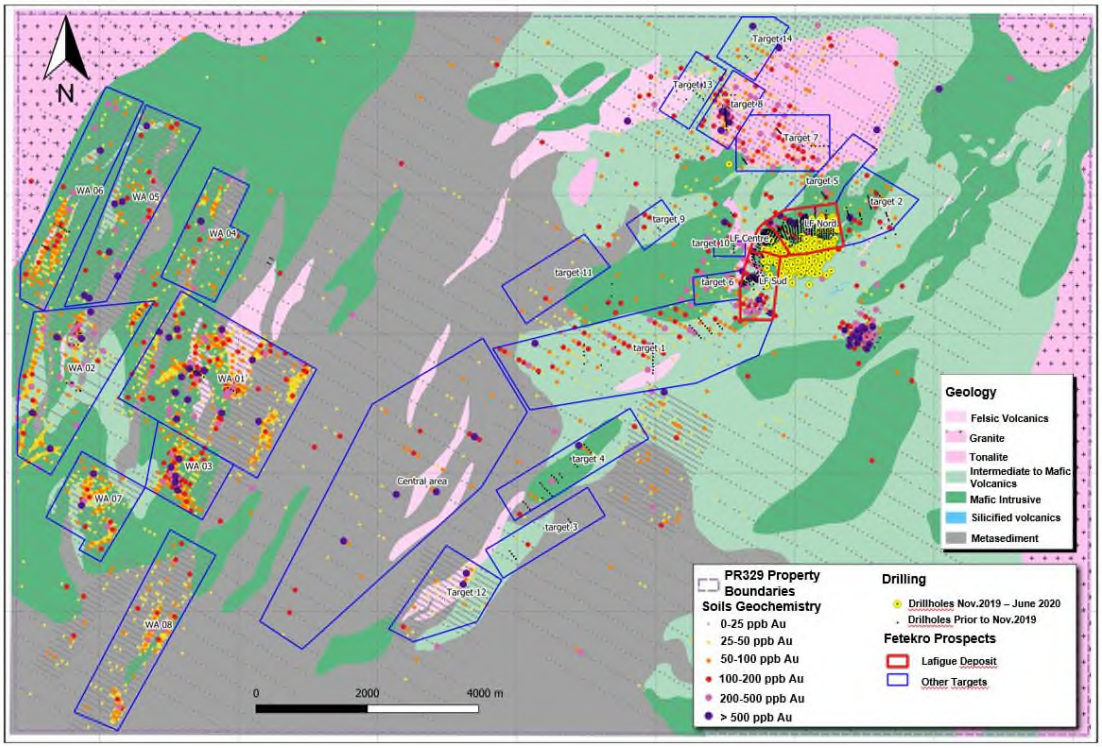
The Lafigué deposit is associated with a major regional north-south shear zone hosted in variable lithologies (sediment, volcano-sediment or igneous), with the main feature being a shear zone between the two contrasting lithologies (metabasalt and metagabbro or metabasalt and intrusive).

9.0 EXPLORATION

Endeavour began exploration on the Fetekro Property in March 2017 following a strategic assessment of its exploration tenements, which ranked the property as a top priority target. Exploration has included geophysics, geochemistry and surface mapping to facilitate the definition of the structural context of the property and has identified a number of gold prospects and anomalies.

Approximately twenty prospects, predominately spread around the Lafigué deposit and on the western and central parts of the Property were identified by a geochemistry gold in-soil (> 50 ppb) sampling campaign and VTEM geophysical aerial survey, as schematically presented in Figure 9.0.1.

Figure 9.0.1 Fetekro Geochemical Sample Grid and Exploration Targets



9.1 Geochemical Surveys

Between 2017 and 2020, Endeavour conducted regional and infill soil geochemical surveys on the Fetekro Property, with 11,523 soil samples and 60 termite mound samples collected. The sampling grids ranged from 400 m by 100 m (regional) to 100 m by 50 m (infill).

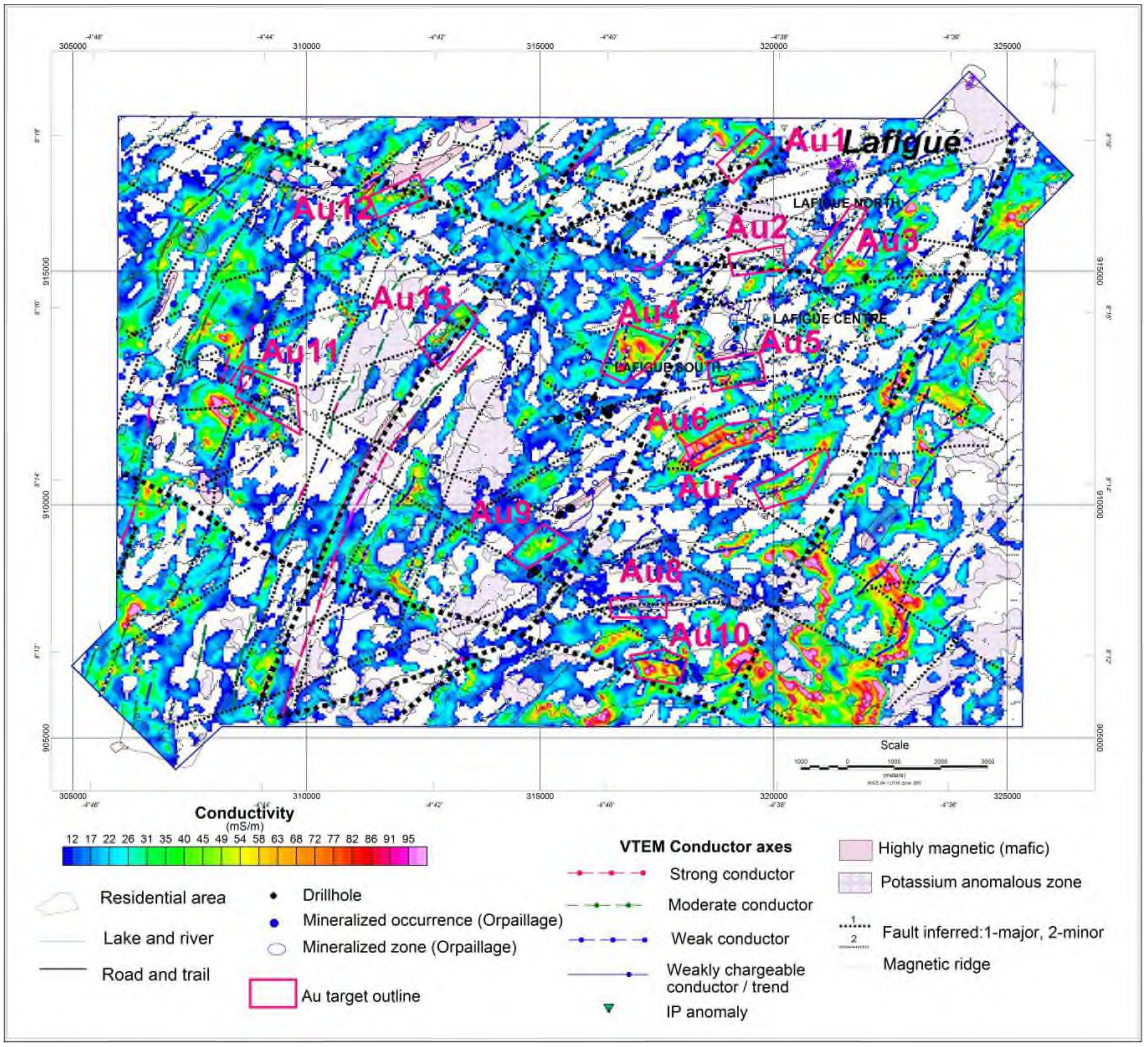
In 2019, Endeavour conducted a regional soil geochemical survey on the Central Area of the Property and a detailed soil geochemical survey on anomalies >50 Au ppb previously highlighted in the western part of the Property with a total of 3,469 soil samples taken. Five new targets were identified on significant soil anomalies, which were several hundred metres long and probably correspond to N10° to N25° shear zones.

9.2 Geophysical Surveys

A VTEM aerial geophysical survey was flown in 2017 to better define the structural context of the permit. The survey area was flown in a northwest to southeast (N135°) direction with a traverse line spacing of 150 m, at a mean altitude of 84 m above the ground. A total of 1,858-line kilometres of geophysical data was acquired during the survey over an area of 257 km^2.

Thirteen potential targets for gold mineralisation and two potential targets for VMS and Cu-sulphide mineralisation were highlighted for further exploration.

Figure 9.2.1 Fetekro Integrated Geophysical Interpretation Including Selected Au Targets

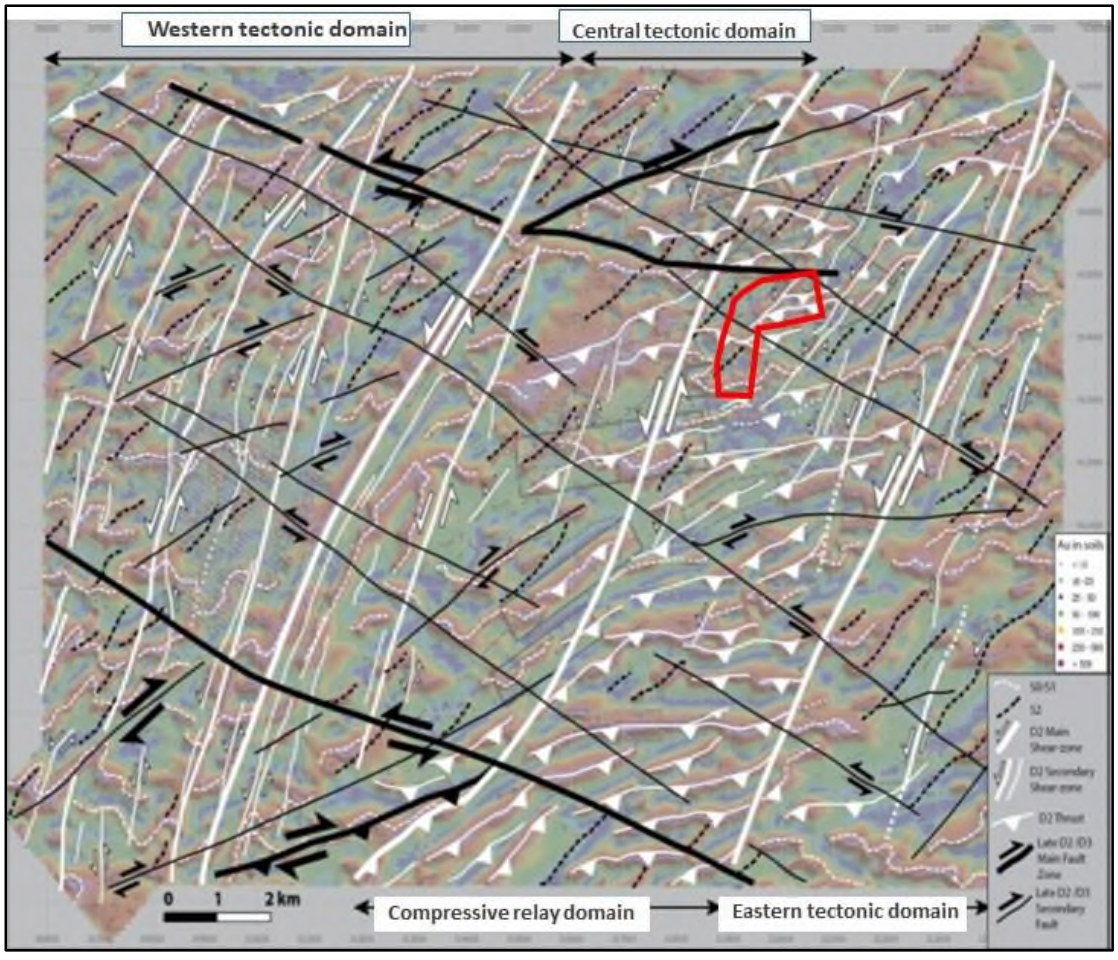


A structural interpretation of the VTEM survey data highlighted four tectonic domains, namely the:

- Western tectonic domain marked by N020 senestral shear zone and N040 regional foliation.
- Central tectonic domain, a transitional domain.
- Compressive relay domain marked by ENE trust.
- Eastern tectonic domain, similar to western domain (senestral N020 shear zone).

The Property is transected by senestral N120° and dextral N060° faults. The structural framework of Lafigué is inferred to be formed in response to NW-SE directed shortening during the D2 and D3 regional tectonic events under a compressive/transpressive regime of deformation.

Figure 9.2.2 Fetekro VTEM Structural Interpretation



Induced polarisation (IP) pole-dipole and gradient surveys were carried out on Lafigué North, Target 2 and Target 5 to better understand the mineralised structures and to find their replicas or extensions. An IP anomaly showing the same signature as the one observed at Lafigué was seen to occur on Target 2. This anomaly was tested by drilling, but no significant mineralisation was found.

Following a ground Induced Polarisation (IP) gradient survey test carried out in January 2018 by SAGAX Afrique on Lafigué known mineralisation, IP pole-dipole and gradient surveys were conducted on Lafigué North, Target 2, Target 5 and Target 4 in order to better understand the mineralized structure and to find its replicas or extensions. A gradient IP survey was also conducted on West Anomalies WA01 and WA03 to find other geophysical signatures of a similar style.

A total of 123 linear kilometres of gradient surveys and 40 linear kilometres of pole-dipole surveys have been performed by SAGAX Afrique. Measurements were taken every 25 m along lines spaced 100 m on Lafigué North, Target 2, Target 5, WA01 and WA03. On Target 4, measurements were taken every 50 m along 200 m spaced lines.

Figure 9.2.3 Ground Geophysics (IP) Surveys

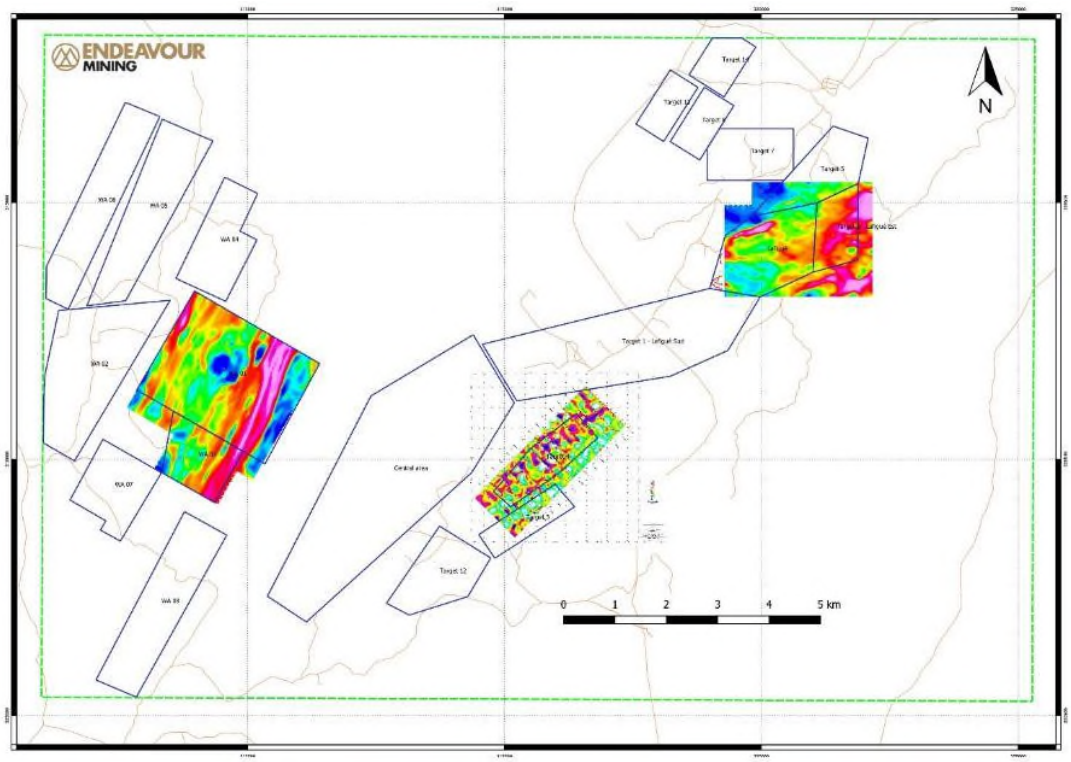
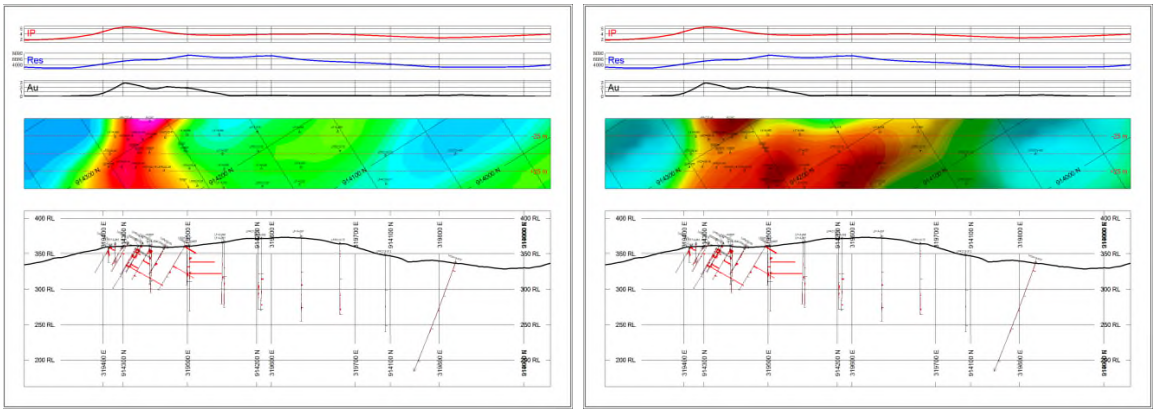


Figure 9.2.4 Lafigué IP Cross-section IP Profiles



Section 3 – Chargeability

Section 3 - Resistivity

9.3 Surface Mapping

Detailed mapping campaigns have been undertaken to refine the existing geological map, to classify soil geochemical anomalies in a regolith regime, i.e. to clarify their 'in-situ' or 'transported' character, to update the cartographic contours of artisanal work areas and to establish correlations between airborne radiometric survey data and geological field observations. Over three hundred grab samples were taken and analysed during this programme.

10.0 DRILLING

10.1 Introduction

Since 2017, drilling by Endeavour on the Fetekro Property has focused on developing the Lafigué deposit and testing priority targets in the vicinity of Lafigué and in the western area of the property. Infill drilling on the Lafigué deposit was also undertaken for resource extensions, metallurgical and comminution samples and for geotechnical data.

Between 2017 and 2020, 39 diamond drill (DD) holes for 6,598 m, 50 reverse circulation (RC) holes for 80,151 m and 164 RC-DD holes for 51,322 m were drilled on the Lafigué deposit, Table 10.1.1 and Figure 10.1.1. In addition, 14 DD holes for 2,244 m and 182 holes for 22,301 m were drilled on the exploration targets (excluding Lafigué) within the Fetekro Property, Table 10.1.2.

Figure 10.1.1 Lafigué Drill Holes Location

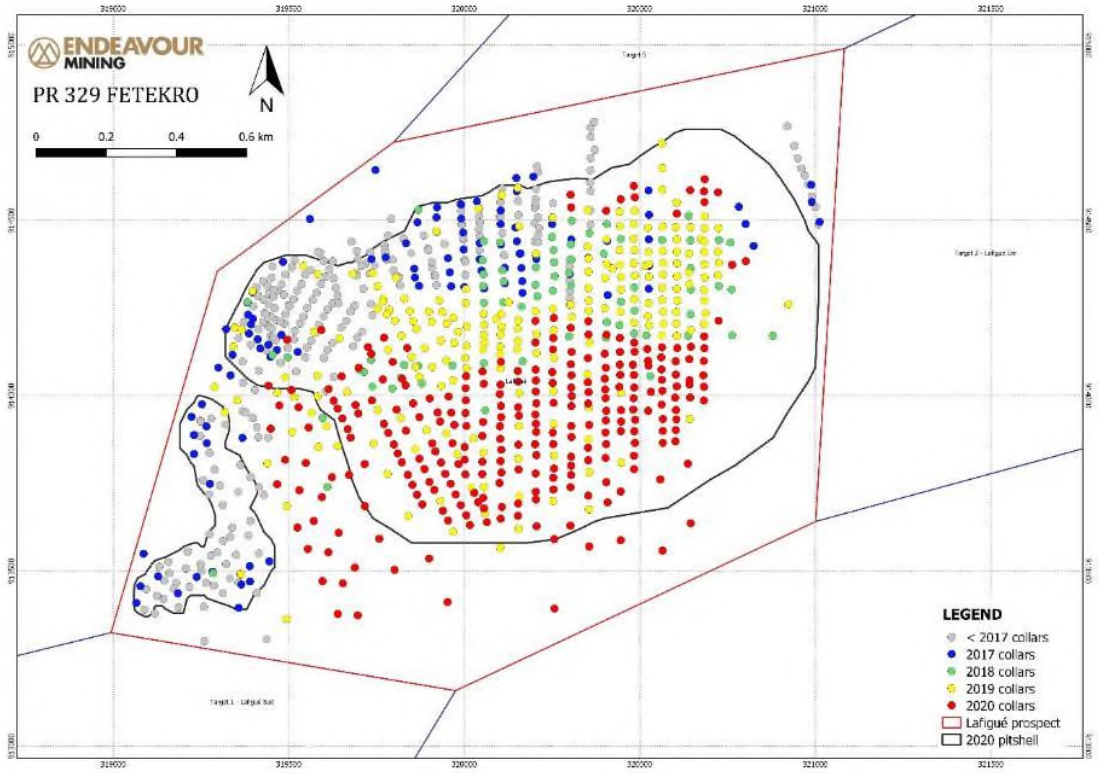


Table 10.1.1 Lafigué Deposit Drilling Summary 2017-2020

Year	Drill Type	No. Drill Holes	Metres	Drill Company
2017	RC	119	6,582	FORACO
	DD	6	762	FTE
2018	RC	44	7,610	GEODRILL
	RC-DD	8	2,662	GEODRILL
	DD	18	3,293	FORACO-GEODRILL
2019	RC	172	30,628	GEODRILL
	RC-DD	27	7,804	GEODRILL
	DD	15	2,543	FORACO-GEODRILL
2020	RC	165	35,331	GEODRILL
	RC-DD	129	40,856	FORACO-GEODRILL
Total		**703**	**138,071**	

Table 10.1.2 Fetekro Property (excl. Lafigué) Drilling Summary 2017-2020

Year	Area	Drill Type	No. Drill Holes	Metres
2017	Target 1	RC	10	1,224
	Target 1	DD	7	897
	Target 2	RC	15	1,506
	Target 2	DD	2	290
	Target 3	RC	3	715
	Target 3	DD	1	237
	Target 4	RC	10	1,211
	Target 4	DD	1	252
	Target 5	RC	5	378
	Target 6	RC	7	461
	Target 8	RC	2	590
	Regional	RC	8	643
	Subtotal		**71**	**8,404**
2018	Target 1	RC	16	2,013
	Target 2	DD	1	327
	Target 3	RC	3	369
	Target 4	RC	14	1,759
	Target 7	RC	10	726
	Target 8	DD	2	241
	Target 9	RC	2	248
	Target 10	RC	4	484
	WA01	RC	10	1,203
	WA03	RC	2	235
	Subtotal		**64**	**7,605**
2019	Target 2	RC	5	984
	Target 7	RC	6	720
	Target 8	RC	9	1,093
	WA02	RC	12	1,641
	WA06	RC	11	917
	WA10	RC	2	202
	WA13	RC	9	1,118
	WA14	RC	2	330
	Subtotal		**56**	**7,005**
2020	Target 1	RC	2	568
	Target 2	RC-DD	2	640
	Target 7	RC	1	323
	Subtotal		**5**	**1,531**
Total			**196**	**24,545**

The main drilling contractor employed since 2018 has been GEODRILL. Drilling was carried out in two 12-hour shifts per machine, six days a week. Endeavour geologists supervised the drilling and geological technicians monitored the machines throughout the shift. The latter were assisted by a team of six labourers for the RC rig and two for the DD rig per shift.

The Exploration Team at Fetekro has been stable over the last seven years and therefore the procedures and protocols have been developed and refined in-house over time, regardless of Project ownership (Endeavour or La Mancha). The drilling data has been collected with the same methods, supervision, and overall quality, and is considered to be compatible, comparable and of equal stature by the Qualified Person.

10.2 Drilling Procedures

Comprehensive procedures documents have been developed for drilling activities by Endeavour geologists. The documents establish the standard procedures for both diamond drilling (DD) and reverse circulation (RC) drilling 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.

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 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 assay laboratory.

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

10.2.2 Reverse Circulation Drilling

At the RC drill rig, the RC chips are collected in one metre intervals from the cyclone 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 assay laboratory.

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 Drill Hole Surveying

10.3.1 Collars

All drill hole collars since 2017 were surveyed by an independent company, Société Nationale de Topographie (SNT), using a differential GPS.

Due to the COVID-19 crisis and difficulties for SNT in reaching the site, the 2020 drill hole collars have been surveyed by another company, Cabinet Kouamelan, also using a differential GPS.

10.3.2 Downhole Survey

Downhole survey measurement are recorded in each hole by the drilling company. Geographic north is used as a reference (magnetic declination was -3.4° in 2020).

Table 10.3.1 below summarises the DTH survey tools used for each company and drill campaign.

Table 10.3.1 DTH Survey Summary

Year	Drill Contractor	Downhole Survey Instrument	Comments
2017	FTE	Gyro	
2017	FORACO	Reflex-EZ track	
2017	GEODRILL	Reflex-EZ track	
2018	FORACO	Reflex-EZ track	
2018	GEODRILL	Reflex-EZ track + EZ-Gyro	
2019	FORACO	Reflex-EZ track	
2019	GEODRILL	EZ-Gyro + SPRINT Gyro	TN14 AZI ALIGNER
2020	FORACO	Reflex-EZ track	
2020	GEODRILL	EZ-Gyro + SPRINT Gyro	TN14 AZI ALIGNER

To better control downhole deviation in the deep holes, stabiliser rods have used since November 2019.

Note, the drilling grid azimuths have changed according to the campaign and drilled area as the project evolved to:

- Southern area N300°.
- Central area N300° and N335°.
- Northern area N180°.

The main azimuths used are listed in Table 10.3.2.

Table 10.3.2 Fetekro Drilling Azimuth Measurements

Azimuth	No. of Drillholes
(0-5)	619
90	12
120	1
130	1
135	2
140	3
180	1
210	1
270	18
(290-295)	2
(295-300)	143
(300-305)	4
310	11
(315-320)	83
324	1
330	2
335	118
(335-340)	64
(340-345)	9
(345-350)	24
(350-355)	37
(355-360)	17

The planned drill hole dips used are -60 or -50°.

The first DTH survey measurement is usually taken at 12 m downhole to verify the inclination and orientation of the drillhole. Subsequent measurements are taken at 50 m downhole and then at 50 m intervals or a shorter interval if the hole starts to swing. The last measurement is taken at the bottom of the hole.

Orientation of the core is undertaken in fresh rock using the Reflex ACTIII system with measurements taken after every run.

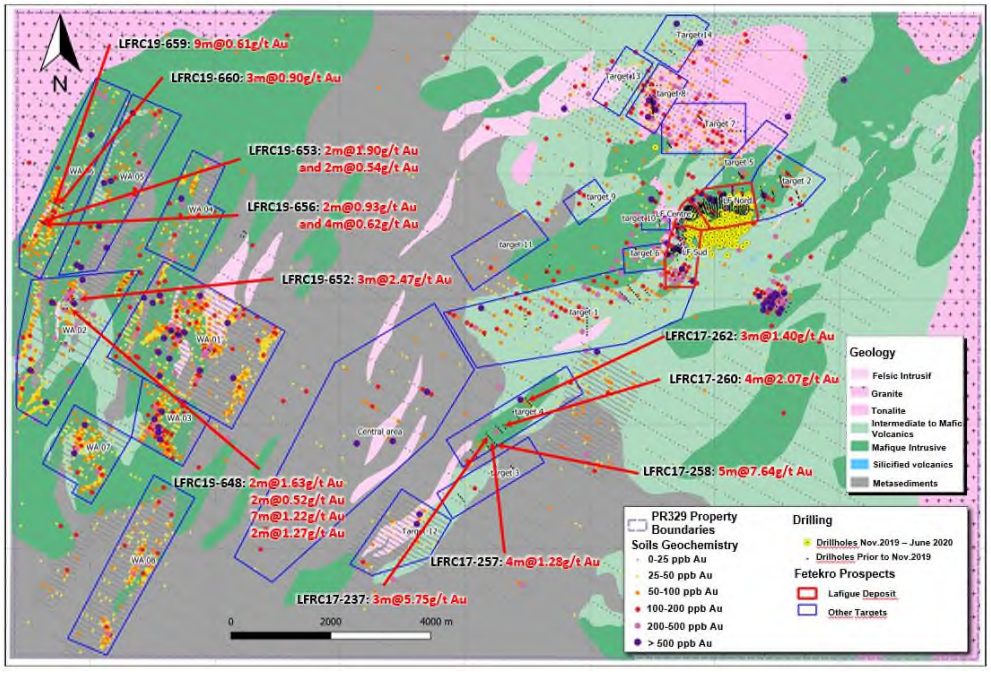
10.4 Exploration Drilling

Several exploration targets have been identified within a 10 km radius of the Lafigué deposit but, until now and due to the high priority given to Lafigué deposit, limited exploration drilling has been done on them, Figure 10.4.1. Drill results over large gold-in-soil anomalies have successfully identified mineralisation associated with a network of subparallel N020° directed steep shear zones hosting both sub-vertical and shallow quartz veins of different natures and with strong hydrothermal alteration.

Table 10.1.2 summaries the exploration drilling undertaken on the Fetekro Prospect by target area, excluding Lafigué, since 2017. Targets with significant drill results include:

- Target 1
- Target 4
- WA01
- WA02
- WA03
- WA06

Figure 10.4.1 Fetekro Drill Targets and Best Drilling Intercepts



11.0 SAMPLING PREPARATION AND SECURITY

11.1 Introduction

The following section includes discussion and comment on the collection and analysis of samples which inform the geological modelling and resource estimates as reported herein. Specifically, this section focuses on; sampling methods, sample submission, sample preparation and analysis, quality assurance and quality control, density analysis, chain of custody and sample security, risks and opportunities, and interpretation, conclusions and recommendations.

The processes described below applies to all drilling campaigns since 2017 for the Fetekro Property. Endeavour has procedural manuals which document all sampling protocols in detail. In addition, regular monitoring of sampling procedures is undertaken to ensure compliance and introduce improvements where appropriate. Sample preparation, analysis and security are under the supervision of Endeavour geologists and were under the supervision of merger partner LMCI from 2013 up until the time at which Endeavour acquired the project in 2016. Information pertaining to historical drilling and sampling prior to 2013 is limited.

Sample collection by Endeavour and LMCI (or by third parties contracted by these companies), were subject to quality control procedures that ensured that industry best practice was utilised for the handling, sampling, transport, analysis, storage and documentation of sample material and analytical results.

11.2 Sampling Methods and Submission

Endeavour has comprehensive procedures manuals that describe the standard procedures for both diamond drill (DD) and reverse circulation (RC) drilling at the rig site. The manuals suggest site layouts and give procedures and examples of the handling of samples, making of boxes, cores, chip bags and boxes, etc.

11.2.1 Reverse Circulation Drill Sampling

RC samples are typically collected in one metre sample lengths in bags directly from the cyclone discharge at the drill rig. The samples are then riffle split into a numbered sample bag with a corresponding sample tag to produce a sample of approximately two kilograms to four kilograms in size. The riffle splitters, plates, tubs and working areas are cleaned with brushes and compressed air after each sample is processed. The remaining reject sample is returned to the bulk plastic bag and either remains at the drill site or is transported to the Exploration Core Yard.

A second two to four kilogram representative backup sample is riffle split at the drill site and stored in large plastic drums in addition to a small portion of drill chips collected from the reject and catalogued in chip trays for future reference.

Approximately 30 assay samples are placed in large polyweave bags and transported to the Bureau Veritas laboratory in Abidjan, Côte d'Ivoire.

Figure 11.2.1 Splitting Methodology using a Riffle Splitter

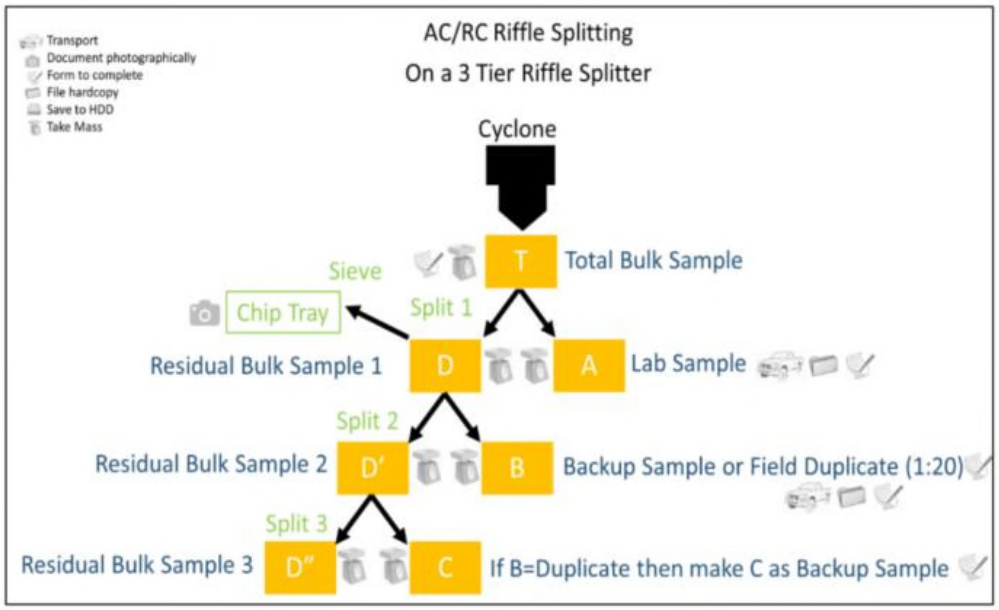
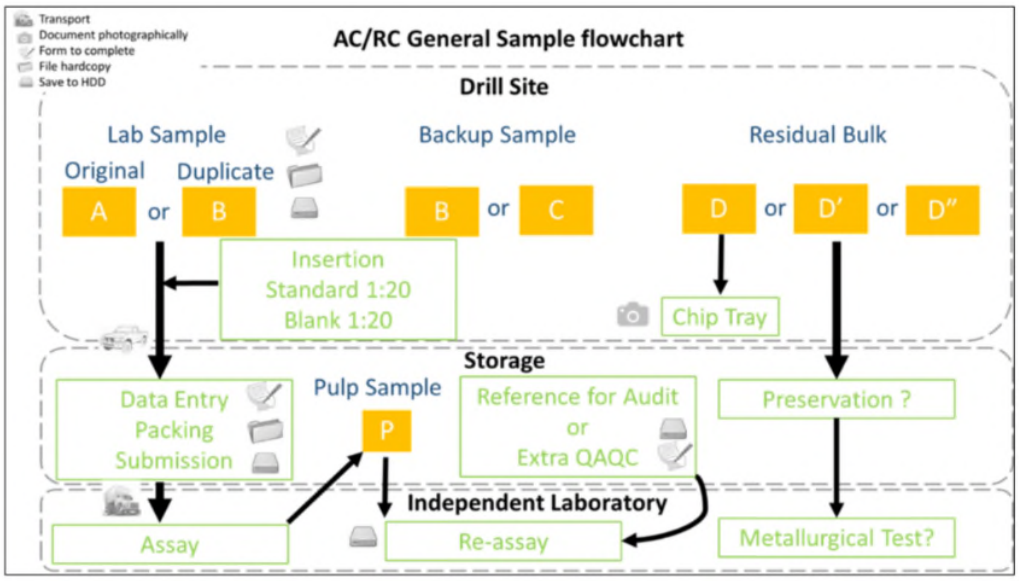


Figure 11.2.2 RC Sample Flowsheet



11.2.2 Diamond Drilling Core Sampling

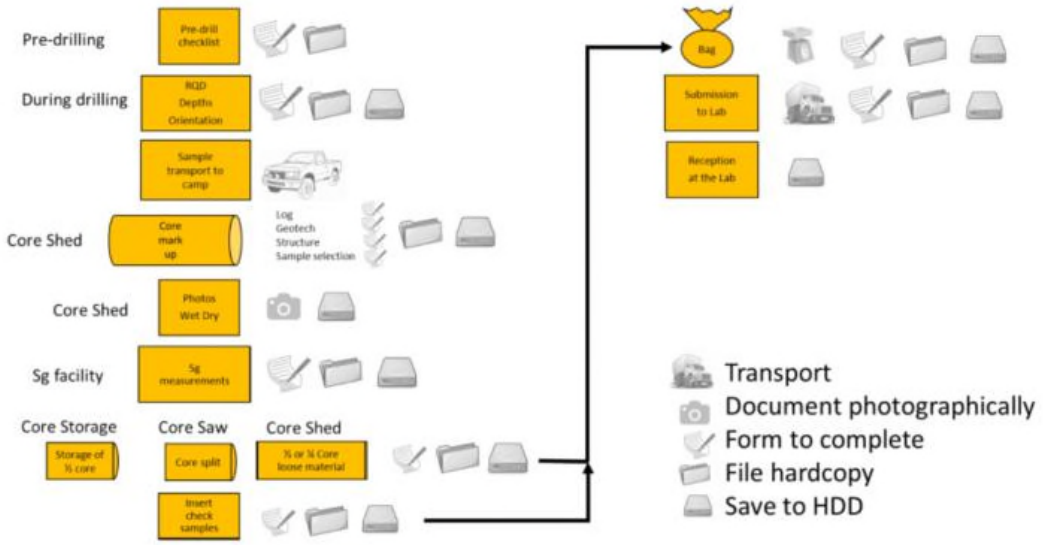
Core from diamond drilling (DD) is placed in either steel or timber core boxes. Core trays are marked with the drill hole number and downhole distance for the start and finish of the core in the tray. Core orientation measurements are taken at the drill site soon after the core is removed from the core barrel in competent fresh rock. The core is geotechnically logged and photographed at the drill site, before being transported to the Core Yard where the core is laid out for geological logging and other data collection.

Specific intervals for sampling based on lithology, generally around one metre in length, are identified during the logging process.

The core is cut in half along its longitudinal axis with a purpose-built diamond-blade core saw. After the core is cut, the right-hand side of the cut core (looking down hole) is placed in a sample bag and sealed, with the remaining core left in the core box. The samples are placed in large polyweave bags and transported to the Bureau Veritas laboratory.

Half of the core is retained at the Core Yard for reference purposes.

Figure 11.2.3 DD Sample Flowsheet



11.2.3 Sample Submission

Sample submission forms detailing the sample number sequences and instructions accompany every sample dispatched to the assay (Bureau Veritas) laboratory. Upon arrival at the laboratory, samples are checked against the submission to ensure all samples have been received. On completion of assaying, company geological staff confirm that all the samples submitted are analysed in accordance with the methods requested.

11.3 Sample Preparation and Analysis

All samples from the Fetekro Property have been prepared and analysed at the Bureau Veritas laboratory in Côte d'Ivorie. Whilst the laboratory is not ISO17025 accredited, it is in the process of achieving this accreditation. The Bureau Veritas laboratory does however, work under the accreditation of the global Bureau Veritas group of laboratories (including those in Australia and Canada) and are covered by the group's ISO9001, ISO14001, ISO18001 and IFIA certificates.

11.3.1 Sample Preparation

When the samples are received at the Bureau Veritas laboratory, they are logged into the laboratory information management system (LIMS).

Sample preparation undertaken at the Bureau Veritas Laboratory includes:

- Oven drying at 105°C to 110°C
- Crushing using a jaw crusher such that 75% passes 2 mm diameter and sub-sampling with a riffle splitter, pulverisation of approximately 0.5 kg with an LM2 pulveriser such that 90% passes a 75 µm mesh. A final homogenised 250 g pulp is split out for transfer to the fire assay circuit.
- Systematic checks to verify particle size during crushing and pulverisation.

11.3.2 Assay Analysis

Fire Assay with an Atomic Absorption Finish is the primary technique applied for gold analysis, with three alternate assay methods also used depending on the gold grade. Bureau Veritas Fire Assay methods include:

- Fire Assay (BV code FA450) - Nominal 50 g charge analysed. Silver used as secondary collector and Au is determined with AAS Finish.
- Fire Assay with Gravimetric Finish (BV FA550 or FAGRA01) - carried out on all samples with a grade greater than 10 g/t Au.

11.4 Quality Assurance and Quality Control

Quality Assurance / Quality Control (QA/QC) programmes are designed to detect contamination or bias and allow analytical precision and accuracy to be quantified in order to provide confidence in their use for Mineral Resource estimations.

QC samples are implemented to assess presence of contamination, accuracy and precision of the sample preparation and analytical procedures. Geochemically barren 'blank' samples are inserted to detect contamination during the sample preparation and analytical stages.

Accuracy is assessed by the assaying of commercially available Certified Reference Materials (CRM) or in-house developed "standards" and by check assays ("reference" or "umpire" samples) at externally accredited laboratories. QC samples can also help identify possible mix-ups or mislabels during sample preparation. The precision of sample results is the measure of how closely the results can be repeated and is assessed by the processing of duplicate samples.

The Bureau Veritas laboratory have their own internal QC 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 blanks, CRMs and duplicates and repeat assaying to monitor the quality of the preparation and analytical processes of the laboratory.

11.4.1 QA/QC Summary

Endeavour has implemented rigorous QA/QC protocols aligned with industry best practice, including:

- Blanks - material returning less than five times the detection limit of the assay method,
- Certified Reference Materials (CRM) - independently certified commercial material from Geostats of Australia,
- Field Duplicates (FDU) - a second sample collected in the field from the same source at the same interval.

QC samples are assessed in an Endeavour corporate standard QA/QC analysis report. This report is auto generated within the 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.

Since 2017, the QA/QC procedures followed during the Fetekro drilling programmes include the insertion of:

- One blank sample per 30 samples.
- One CRM sample per 20 samples.
- One duplicate sample per 20 samples.

Table 11.4.1 summarises the QA/QC insertion rates of the Fetekro drilling programmes.

Table 11.4.1 Summary of QA/QC Insertion Rates for the 2017 to 2020 Fetekro Drilling Programmes

No. Blanks Inserted	Blanks Insertion Rate	No. CRMs Inserted	CRM Insertion Rate	No. Field Duplicates Inserted	Field Duplicate Insertion Rate
5,672	4%	7,089	5%	7,089	5%

11.4.2 Certified Reference Materials (CRM)

Results of the regular submission of Certified Reference Materials (CRM) are used to identify problems of accuracy with specific sample batches and long-term biases associated with the primary assay laboratory. CRM from Geostats were used in the 2017 to 2020 drilling programmes. Specific pass/fail criteria were determined from the certified values provided for the CRM by the manufacturer. The approach to setting acceptance limits was to use the mean assay (also known as the Certified Reference Value (CRV)) plus/minus two standard deviations (±2SD) as a warning limit and plus / minus three standard deviations (±3SD), as a failure limit. Results falling outside of the failure limit of ±3SD deviations must be investigated to determine the source of the erroneous result, either from analytical sources, sample dispatch, CRM insertion errors or data entry errors. An overall CRM failure rate of less than 2% is considered acceptable by Endeavour.

Endeavour inserted 7,089 CRMs into sample sequences in the 2017 to 2020 drilling programmes, as summarised in Table 11.4.2 following.

Table 11.4.2 CRM Insertion Summary

CRM Type	Number Inserted	Certified Value	Certified Standard Deviation	Mean – 3 SD	Mean + 3 SD
Std-G300-8	124	0.06	0.06	-0.06	0.18
Std-G310-6	241	0.04	0.04	-0.04	0.12
Std-G316-2	168	0.04	0.04	-0.04	0.12
Std-G910-8	124	0.04	0.04	-0.04	0.12
Std-G913-9	2,367	0.17	0.17	-0.17	0.51
Std-G914-2	2,053	0.08	0.08	-0.08	0.24
Std-G915-6	2,012	0.04	0.04	-0.04	0.12
Total	**7,089**				

In summary, of the 7,089 CRMs inserted in drill programmes, 29 or 0.41% were outside ±3SD and were considered failures.

Table 11.4.3 summarises the CRM performance statistics.

Table 11.4.3 CRM Performance Summary

CRM Statistic - Fire Assay											
CRM	No. CRM	Min Au_ppm	Max Au_ppm	Average Au_ppm	CRM Source	Certified Value	Certified SD	CRV - 3 SD	CRV + 3 SD	No. Failures	% Failures
Std-G300-8	124	0.61	1.13	1.07	Geostats	1.07	0.06	0.89	1.25	1	0.01%
Std-G310-6	241	0.58	5.05	0.65	Geostats	0.65	0.04	0.53	0.77	2	0.03%
Std-G316-2	168	0.6	1.11	1.03	Geostats	1.04	0.04	0.92	1.16	6	0.08%
Std-G910-8	124	0.58	1.07	0.61	Geostats	0.63	0.04	0.51	0.75	1	0.01%
Std-G913-9	2,367	0.61	5.27	4.96	Geostats	4.91	0.17	4.40	5.42	10	0.14%
Std-G914-2	2,053	0.63	2.7	2.49	Geostats	2.48	0.08	2.24	2.72	3	0.04%
Std-G915-6	2,012	0.59	5.05	0.67	Geostats	0.67	0.04	0.55	0.79	6	0.08%
Totals	**7,089**									**29**	**0.41%**

11.4.3 Blanks

The regular submission of blank samples is used to assess contamination during sample preparation, fusion and cupellation, and in AAS solutions. Blanks can also identify sample numbering and swapping errors. The QA/QC protocol for the Fetekro Property, stipulates that blanks be inserted in the sample stream at a rate of at least one in every 30 samples. For the 2017 to 2020 programmes, two types of blank material were used, a coarse crushed granite sourced from a quarry near Abidjan to test the crusher and a fine river sand sourced from a river in Abidjan to test the pulveriser. These materials were classed as blanks after analysis at several laboratories in Côte d'Ivoire.

A blank assay is considered a failure by Endeavour if it returns a value greater than five times the detection limit of the assay method. Some 5,672 blank samples were inserted over the course of the 2017 to 2020 drilling programmes.

The overall performance of the blanks samples was satisfactory with zero failures (0.0% failure rate).

Table 11.4.4 following, provides a summary of blanks used in the 2017 to 2020 drilling programmes.

Table 11.4.4 Blank Performance Summary

Parameter	Value
Number of Blanks	5,672
Blanks > Detection Limit (0.005 ppm)	811
Max Blank Grade (ppm Au)	0.03
Blank Failures (>0.05 ppm Au)	0
Blank Failures Percentage	0.00%

11.4.4 Duplicates

Duplicates are generated to assess the precision of samples.

An original sample and a field duplicate sample are taken at the same interval down the drill hole but are bagged separately. Each sample is given a unique sample number in order to be blind to the laboratory team. Field duplicates assess variability when two samples are collected at the same interval. Field duplicates can also help detect sample number mix-ups and assess the natural local-scale grade variation or nugget effect (i.e. inherent grade variation).

Finally, field duplicates are a check on possible sample over-selection or bias. Bias may arise where the sampler has either purposely, or inadvertently, preferentially sampled geological material with visible mineralisation.

A set of calculations have been integrated into the corporate standard QA/QC report based on procedures used by some assay laboratories to evaluate internal duplicate and repeat samples. 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 (ARD) is then calculated for each individual pair. If the ARD result exceeds the calculated tolerance result the duplicate pair is considered to have failed, as illustrated in Table 11.4.5.

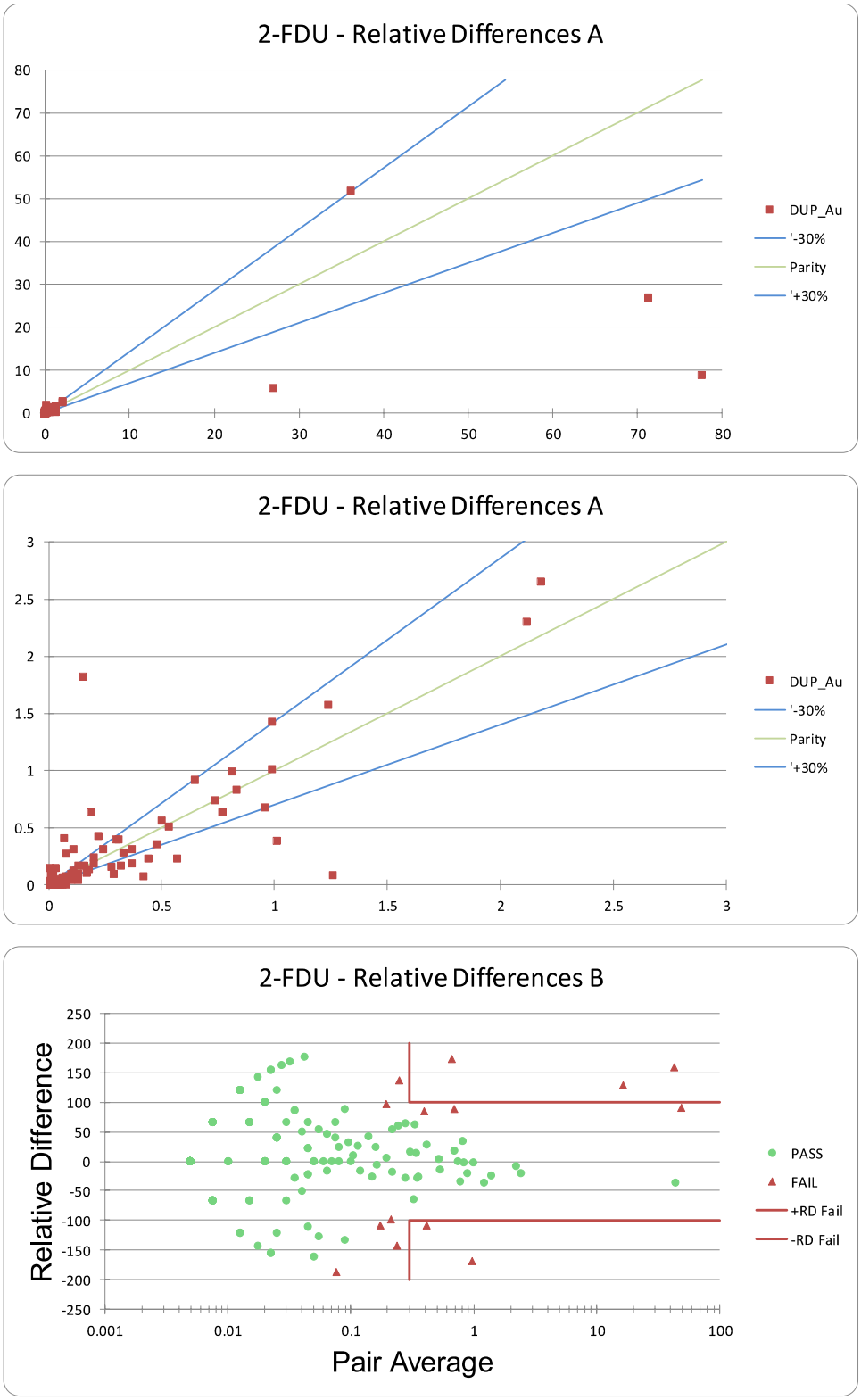
Endeavour does not submit pulp or laboratory duplicates.

Table 11.4.5 Example of Field Duplicate Analysis Report

ORIG_SampleID	ORIG_Au	DUP_SampleID	DUP_Au	PairAvg	RelDiff	pctTolerance	pctDifference	CSA_Pass
329097930	0.080	329097931	0.005	0.043	1.765	315.294	249.567	PASS
329097950	0.005	329097951	0.005	0.005	0.000	2080.000	0.000	PASS
329097970	0.005	329097971	0.005	0.005	0.000	2080.000	0.000	PASS
329097990	0.070	329097991	0.040	0.055	0.545	261.818	77.139	PASS
329098010	0.020	329098011	0.020	0.020	0.000	580.000	0.000	PASS
329098030	0.090	329098031	0.060	0.075	0.400	213.333	56.569	PASS
329098050	0.130	329098051	0.050	0.090	0.889	191.111	125.708	PASS
329098070	0.290	329098071	0.100	0.195	0.974	131.282	137.795	FAIL
329098090	0.005	329098091	0.005	0.005	0.000	2080.000	0.000	PASS
329098110	0.005	329098111	0.005	0.005	0.000	2080.000	0.000	PASS
329098130	0.030	329098131	0.010	0.020	1.000	580.000	141.421	PASS
329098150	0.005	329098151	0.005	0.005	0.000	2080.000	0.000	PASS
329098170	0.005	329098171	0.010	0.008	0.667	1413.333	94.281	PASS
329098190	0.005	329098191	0.020	0.013	1.200	880.000	169.706	PASS

Graphical evaluation of field duplicate performance is depicted in XY scatter plots and relative difference against mean grade plots (Figure 11.4.1).

Figure 11.4.1 Example of Field Duplicate Plots



The performance of the field duplicates during the 2017 to 2020 drilling programmes are well within acceptable limits and, no significant bias is present in the duplicate assays.

Most failures are due to due to coarse gold in the system. The Correlation Coefficient statistic was lower than expected, as seen in Table 11.4.6. This was in contrast to good results presented in the Variance, Coefficient of Variance and % Difference between Means; however, it is apparent that a skewing was introduced to the population by the presence of very high grade coarse gold samples.

High grade coarse gold samples invariably present bias concentration in samples and often fail QC tests for duplicates.

When these very few very high-grade coarse gold samples are removed from the population, the Correlation Coefficient improves dramatically as presented Table 11.4.7.

Table 11.4.6 Summary of Duplicate Statistics - All Duplicate Pairs Included

Field Duplicates	**Fire Assay**	
No. Duplicates	7,099	
No. Duplicate Pairs > D.L.	3,325	
No. Duplicate Failures	82	1.16%
	Original	**Duplicate**
Mean Grade	0.412	0.423
Minimum Grade	0.005	0.005
Maximum Grade	78.890	77.910
Median Grade	0.005	0.005
Variance	12.78	13.06
Standard Deviation	3.58	3.61
Coefficient of Variation	8.68	8.54
Correlation Coefficient	**0.88**	
% Difference Between Means	2.75%	

Table 11.4.7 Summary of Duplicate Statistics - Very High Grade Coarse Gold Duplicate Pairs Removed

Field Duplicates	Fire Assay	
No. Duplicates	7,090	
No. Duplicate Pairs > D.L.	3,316	
No. Duplicate Failures	73	1.03%
	Original	**Duplicate**
Mean Grade	0.375	0.390
Minimum Grade	0.005	0.005
Maximum Grade	73.140	77.910
Median Grade	0.005	0.005
Variance	10.27	11.92
Standard Deviation	3.20	3.45
Coefficient of Variation	8.56	8.86
Correlation Coefficient	**0.98**	
% Difference Between Means	3.97%	

11.4.5 Failure Procedures

When a blank, duplicate or CRM failure is encountered, the initial step is to determine if the failure is genuine or a data entry problem. Data entry problems are identified by cross-checking the relevant original paper sample log against the database. CRMs are additionally checked against photographs taken of each CRM sachet directly prior to insertion into the sample bag. Sample weights are checked to verify that the sample providing the failed values are, in fact, QA/QC samples. Genuine failure procedures are summarised as:

- **Duplicate Failure**: The weight of the duplicate pairs is checked and should be similar. If the gold values in the duplicate pair are high and there is a large contrast between the original and duplicate assay grades, the samples are assumed to be in a zone of coarse gold and no re-assay is requested. If the difference between samples is within normal failure ranges, a re-assay is requested.
- **Blank Failure**: Always sent for re-assay.
- **CRM Failure**: The weight of the failed CRM is checked. The weight of CRM is always much less than a down-the-hole sample, e.g. 0.5 kg to 0.6 kg for a CRM versus over 2 kg for a field drill sample.
  - If a CRM failure occurs in a zone devoid of gold mineralisation, the laboratory is informed of the failure but a re-assay request is not made.
  - If the failed CRM occurs in a zone of gold mineralisation, the laboratory is informed and a re-assay request made.

For all failed QA/QC samples, an interval is requested for re-assay including the failed QA/QC sample and all samples above and below to which the QA/QC sample are linked. Typically, five samples before and five samples after the failed QA/QC sample are chosen. The re-assay sample is generated by taking an additional (50 g to 60 g) sample from the original 250 g pulp sample. If the re-assay sequence includes CRMs, replacement CRMs are provided.

If there is insufficient original pulp material for a re-assay, material from the coarse reject is sourced and a new pulp is prepared and provided for re-assay after pulverisation.

11.5 Density Analysis

For the 2017 to 2020 drilling programmes, there were 1,991 density measurements taken by Endeavour.

Drill core samples are prepared and measured on site at the Fetekro Core Yard by technicians under the supervision of geologists. The rock type and weathering profile (e.g. oxide, transition, fresh) are recorded.

Competent core is sawn into pieces of approximately 160 g to 1,000 g. In cases of highly oxidised rock, pieces of appropriate length are chosen. The samples are sun dried for two days before preparation. The density is then determined in accordance with the following procedure:

- The dry sample is weighed and the mass recorded in a spreadsheet.
- The sample is covered with a thin wax film or plastic film wrap.
- The wax covered/film wrap sample is weighed and the mass recorded.
- The sample is put in water, weighed and the mass recorded.
- Sample density is calculated using the following formula:

  Density=A/(B-C((B-A)/D))

  where: A= Mass of dry sample; B=Mass of waxed/film wrapped sample out of water; C=Mass of waxed sample in water; D=Density of paraffin (0.9 g/cm^3). Units are g/cm^3 or kg/m^3.

Table 11.5.1 summarises densities by rock type. This summary includes all three weathering types, with each of the major lithologies logged within the weathering type.

Table 11.5.1 Densities by Rock Type

Weathering	No. Density Measurements	Average Density Measurement (g/cm³)	Min. Density Measurement (g/cm³)	Max. Density Measurement (g/cm³)
BDRK	**1,979**	**2.80**	**1.73**	**4.25**
DIOT	168	2.78	2.11	3.53
QRTZ	2	2.70	2.65	2.76
QZVN	52	2.70	2.50	3.59
IMAF	506	2.88	2.03	4.22
VMAF	557	2.84	1.73	4.05
IFEL	578	2.72	2.12	4.21
QZTM	41	2.81	2.64	3.06
VFEL	36	2.72	2.68	2.83
DYKE	6	2.85	2.82	2.93
IMM	1	2.93	2.93	2.93
CHRT	6	2.70	2.67	2.73
QZON	25	2.82	2.65	4.25
NRCV	1	2.92	2.92	2.92
SAPR	**2**	**2.03**	**1.42**	**2.63**
ARGL	2	2.03	1.42	2.63
SPRK	**10**	**2.65**	**2.25**	**3.05**
IMAF	4	2.65	2.25	3.05
VMAF	4	2.65	2.45	2.82
QZTM	1	2.65	2.65	2.65
VFEL	1	2.70	2.70	2.70
Grand Total	**1,991**	**2.80**	**1.42**	**4.25**

11.6 Chain of Custody and Sample Security

All DD and RC samples are transported to Endeavour's secure (walled and lockable) Core Yard (i.e. sample management facility) on the Fetekro Property.

QA/QC samples are inserted into the sample sequence for DD and RC drill holes. Sample batches are then placed in sealed and numbered poly-weave or plastic bags for transport. The sample shipments are verified by Endeavour personnel at the Core Yard and a shipment list is sent to the laboratory. Upon reception at the laboratory, lab sample preparation personnel verify the shipment samples against the shipment list and sign a confirmation of receipt. All aspects of sample collection and dispatch is undertaken by/or managed by Endeavour personnel.

11.7 Risks and Opportunities

To date, no significant risks or opportunities have been identified with respect to sample preparation, quality control and security as practiced on the Fetekro Property.

11.8 Interpretation, Conclusions and Recommendations

It is the Qualified Person's opinion that, the information as presented herein, is suitable for the estimation of Mineral Resources. It is noted that:

- All aspects relating to the collection, preparation and dispatch of drill samples (DD and RC), is carried out by Endeavour personnel.
- Sample collection and preparation, analytical techniques, security and QA/QC protocols are consistent with standard industry practice.
- The sampling procedures are appropriate and consistent with the style of gold mineralisation under consideration.
- Results of the QA/QC programme ensure the samples are accurate, precise and reliable.

12.0 DATA VERIFICATION

12.1 Introduction

In accordance with 'Item 12' of NI Form 43-101F1 and the Qualified Persons' responsibilities defined in Table 2.5.1, Sections 12.2 to 12.6 summarise where relevant:

- The data verification procedures applied by the Qualified Person(s);
- Any limitations on or failure to conduct such verification, and the reasons for any such limitations or failure; and
- The Qualified Person's opinion on the adequacy of the data for the purposes used in the Technical Report.

12.2 Geological Data

12.2.1 Introduction

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

The key areas addressed comprise the data verification processes applied for the Fetekro property, site visits completed, risks and opportunities, and interpretation, conclusions and recommendations.

Endeavour, under the direction and oversight of the Qualified Persons, implements a comprehensive geological data management system to assure the appropriate level of quality control and data verification is undertaken to support the geological models and current Mineral Resources as reported herein.

12.2.2 Historical Data Validation and Verification

No independent Qualified Person has evaluated the historical Fetekro data (pre-2017). For the historical holes, most of the paper logs and sampling plans have been located and checked against the electronic database. The historical data, according to the archives and the end of drilling campaign reports, the procedures applied appear to be similar to the ones used by Endeavour since 2017.

In 2013, La Mancha implemented an in-house SQL based Database Management System (DBMS).

12.2.3 Database Checks in Drilling Programmes Managed by La Mancha and Endeavour Exploration

After acquisition of the Property in 2016, Endeavour implemented their own SQL based DBMS. All historical data generated in the Fetekro Property was audited during importation into the Endeavour DBMS, so as to adhere to the strict data integrity requirements of an industry standard SQL database.

Standard error mitigation measures built into the Endeavour DBMS capture errors including inconsistent collar coordinates, incorrect or missing down hole survey records, missing sample-assay records and missing or overlap errors in down the hole interval data. Once these errors are corrected, the data is imported.

Since 2013, all data has been managed with the strict built-in data integrity requirements of an industry standard SQL based DBMS with database checks including:

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

All inconsistencies are reported to the database management team and actioned accordingly.

The Endeavour DBMS is managed by Geoff Day P.Geo., Exploration Group Database and Quality Control Manager, a Qualified Person.

12.2.4 Twinned Hole Comparison

There have been no twinned holes drilled on the Fetekro Property between January 2017 and 31 July 2020.

12.2.5 Paired Statistics

Given that the Fetekro Property is at the PFS level of development, no grade control (GC) drilling has been undertaken and therefore a comparison of GC and exploration/resource drilling results and models cannot be made.

12.2.6 Site Visits

Ms Silvia Bottero (Pr. Nat. Sci.), a Qualified Person, was La Mancha's Exploration Manager for the Project from 2014 to 2016 and Endeavour's Exploration Manager from 2017 to August 2020 and over this period, was at the Site extensively. In September 2020, Ms Bottero was appointed Endeavour's VP for Exploration for Côte d'Ivoire and has continued her involvement with the Project, with her last site visit being December 14 to 17, 2020.

To ensure adherence to Endeavour procedures and protocols, regular site visits were made by Gérard de Hert, EurGeol, Senior VP Exploration, a Qualified Person, since 2016.

CSA-Global visited the Fetekro Property in May 2017 and May 2019, and subsequently undertook an independent, comprehensive review of all field and QA/QC procedures in the sample management facility / Core Yard.

No other independent Qualified Persons have visited the Fetekro Property between January 2017 and July 31, 2020.

12.2.7 Risks and Opportunities

To date, no significant risks and opportunities have been identified with respect to geological data verification.

12.2.8 Interpretation, Conclusions and Recommendations

The geological data verification processes completed to date have been done 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 and resource modelling that inform the Mineral Resources as reported in this Technical Report. The Endeavour Qualified Person considers the sample data to be robust and of a high standard, and appropriate for the use in a Mineral Resource Estimate.

12.3 Mineral Resource

Before the Mineral Resource Estimate was compiled, Kevin Harris, the QP for the Mineral Resource, ran an audit of the drill hole database as supplied by the central database team from the DBMS. The audit employed the in-built functions in GEOVIA Surpac™ and Geosoft Target. It also applied visual checks of the data in three-dimensions and in Microsoft Access. Audit reports were sent to the central database team for investigation, and correction if required in the master database. All errors were corrected with an audit trail.

The Qualified Person considers that the internal processes applied by Endeavour are appropriate to support the geological model and current Mineral Resources as reported herein.

12.4 Mineral Reserves and Mine Planning

This subsection has been written by Mr Allan Earl, the QP for the Mineral Reserve and Mine Planning components of this report.

Geological wireframes, oxidisation surfaces and Mineral Resource block models for Lafigué were provided by Endeavour. The block model attributes and grade tonnage curves were reviewed and re-reported. No discrepancies with the Mineral Resource block model report were identified.

The metallurgical recoveries and processing costs for the oxide, saprock, and fresh rock used for the pit optimisation process were provided by Endeavour and Lycopodium. The optimisation results and mine schedules were provided to Endeavour and Lycopodium to confirm the correct values had been used.

Pit slope angles used in the optimisation process and final mine designs were based on Golder's geotechnical study. The final pit slope angles were compared with the recommendations in the Golder report. No material issues were identified.

Mining costs used in the pit optimisation process were derived from typical industry costs. Snowden provided bench inventory and haul profiles to Endeavour, who calculated the mining capital and operating costs. These costs were used in the PFS financial model. A comparison of the optimisation costs and the calculated PFS costs did not show a material difference.

12.5 Metallurgy

This subsection has been written by Mr David Gordon, the QP for metallurgy.

12.5.1 Sample Selection

Samples for the metallurgical testwork programme were selected from drill core available at the Lafigué site. The QP, Mr David Gordon, did not visit the site for sample selection; however, a process representative of Lycopodium (Mr Mark Giddy, MAusIMM) did visit the site to assist geologists with sample selection and to verify samples to be sent back to the nominated metallurgical test laboratory (ALS in Perth, Western Australia).

Samples did not require any special storage requirements (i.e. cold storage or purged with nitrogen) due to the proposed flowsheets been metallurgically tested (oxide ore processing scheme).

Samples selected for the metallurgical testwork programme were bagged, labelled and placed in drums for transport to ALS in Perth. The inventory of samples selected on site was compared to the received samples at ALS in Perth and no anomalies were reported.

Adequate steps were taken to ensure samples suitable for metallurgical testwork were selected.

12.5.2 Metallurgical Sample Assays

In general, the Lafigué deposit contains a high proportion of gravity recoverable gold i.e. coarse gold. Multiple gold head grade assays on the same sample, as well as screen fire assays conducted on head samples, confirmed the presence of coarse gold. As a consequence, the assayed head grade of samples sometimes differed from the calculated head grade during metallurgical testwork. To ensure coarse gold contained in samples was not biasing calculations of metallurgical performance in leach tests, the calculated head grade was used to determine overall metallurgical performance (i.e. gold extraction). This is standard metallurgical practice for ores containing coarse gold and adequately states the expected metallurgical performance.

12.5.3 Metallurgical Testwork

Adequate testwork data was generated during the metallurgical testwork programme for the purposes of this technical report i.e. for determining overall flowsheet, determining a plant design basis and predicting gold extractions and reagent consumption.

Metallurgical testwork data was developed using industry standard test procedures at an industry recognised metallurgical laboratory. The data generated from testwork is adequate for preliminary process design and overall project assessment.

Some minor additional metallurgical testwork has been recommended during the next phase of study to provide confirmatory data for plant design and predictions of metallurgical performance and reagent consumption.

12.6 Environmental and Permitting

The information related to Section 20 of this Report was reviewed by the QP, Mr Patrick Pérez, and is based on recent environmental studies developed for Endeavour by reputable local environmental consultants (Cabinet ENVAL). The Environmental and Social Impact Assessment (ESIA) study developed as part of the PFS was done as per International Standards and met all the criteria in order for the government to deliver the Environmental permit for the project.

12.7 Marketing and Contracts

The information related to the Market Study and Contracts section (Section 19) was reviewed by the QP (Mr Patrick Pérez) and is based on recent values and procedures used by Endeavour to market the products from its operating sites.

12.8 Financial Modelling

The information related to Section 22 of this Report was reviewed by the QP (Mr Patrick Pérez) and is based on the values and procedures used by Endeavour to evaluate its own projects

12.9 Project/Engineering Development

12.9.1 Plant Geotechnical Data

No plant site geotechnical data was available for the Technical Report. Thus, Lycopodium made adequate allowances at this level of study (PFS) for preliminary engineering design and capital cost estimation.

Site plant site geotechnical data will be developed at the next stage of project development to refine the assumptions made.

12.9.2 Site Geotechnical Data

For the site geotechnical data, Knight Piésold used their experience from other projects in the region to assume the geotechnical conditions at the site for preliminary design purposes. It is envisaged that these assumptions will be evaluated by conducting a site investigation at the next stage of project study.

12.9.3 Climate Data

Rainfall data for the site was obtained from Système d'Informations Environnementales sur les Ressources en Eau et leur Modélisations (SIEREM). Evaporation data for the site was obtained from the journal article "Sugarcane Yield Variations in Northern and Central Ivory Coast as Influenced by Soil Water Balance over Two Critical Growth Stages" by C. B. Péné and S. G. Koulbaly issued in the Journal of Agricultural Science and Technology February 2011, Volume 5, Mo. 2 (Serial No. 33).

12.9.4 Site Seismicity

For the site seismic evaluation, data were obtained from the Global Earthquake Model which were input into the computer program Openquake to determine site design criteria.

12.9.5 Site Topography

For the site topography dxf files were provided by Endeavour Mining. The files were compared with Shuttle Radar Topography Mission (SRTM) data to confirm it was representative. SRTM data were used for larger catchment areas (beyond the Endeavour Mining .dxf files).

13.0 MINERALS PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Two programmes of metallurgical and comminution testwork have been undertaken for the Lafigué Gold Project on representative samples sourced from the expected minable pit based on the July 2019 Mineral Resource Estimate. Scouting metallurgical testwork was conducted in 2018 and in 2019 a metallurgical testwork programme built on the previous work and tested a wide range of metallurgical and comminution samples.

The testwork programmes were carried out by ALS (ALS Metallurgy, Perth, Western Australia) under the direction of Lycopodium Minerals Pty Ltd (Lycopodium). Comminution SMC (SAG mill comminution) test results analysis was completed by JKTech (JK Tech Pty Ltd, Queensland). Thickening testwork was carried out by GBL (GBL Process Pty Ltd, Perth, Western Australia) on slurry samples prepared by ALS.

The 2018 testwork programme was conducted on ten variability samples and two comminution samples. The following provides a summary of the testwork undertaken:

- Comminution testing to investigate the comminution parameters.
- Variability composite gravity-cyanidation leach testwork at a fixed grind size to determine gravity gold recovery and gold leach extractions.
- Three master composites were made up from the variability composites and gravity-cyanidation leach testwork was conducted at three grind sizes to determine the effect of grind size on gravity and leach gold extraction.
- Rheology testwork was conducted on selected master composites.

The 2019 testwork programme involved a detailed metallurgical and comminution testwork programme on 36 variability samples and 12 comminution samples representative of the 2019 Mineral Resource. The testwork programme included the following stages:

- Selection of samples for comminution, gold and silver extraction and physical characterisation testwork that were representative of the range of ore types, oxidation states, locations and head grades.
- Comminution variability testwork was carried out on the comminution composites with parameters required for comminution circuit design determined.

- Ore type master composite optimisation testwork programme. Formation of ore type master composites, by combining selected drill core intervals representing the main ore types, on which testwork to determine the optimum processing route was completed.
  - Mineralogy and gold extraction testwork to assess the key elements present and to identify potential impacts on leaching characteristics.
  - Testwork to develop the optimal process conditions (grind size, cyanide addition, slurry % solids, aeration) for the maximum economic gold recovery and minimum reagent consumption.
  - Determine the engineering parameters necessary for leach/adsorption and cyanide detoxification circuit design and determine slurry rheology and thickening characteristics.
- Variability testwork. Evaluate gold extraction, using the optimum processing conditions, on individual samples representing the range of ore types, grades and spatial distributions within resource.

13.2 Sample Selection Methodology

All samples were selected by site geologists for both programmes to be representative of the mineable resource of that time with a range of ore types, oxidation states, locations and head grades.

A total of 163.5 kg of drill core was delivered to ALS, Perth, in June 2018 for the 2018 scouting testwork programme. The quarter drill core was contained in 14 individual sample bags within two drums. Each sample bag contained a client sample identification number, drill hole number and interval depth. From the samples received, two comminution composites, and 10 initial variability composites from which three master composites were generated.

A total of 779 kg of drill core was delivered to ALS, Perth, in September 2019 for the 2019 metallurgical testwork programme. The broken diamond drill core was contained in 461 sample bags within nine drums. Each sample bag contained a client sample identification number, drill hole number and interval depth. From the samples received, 12 fresh ore comminution composites, 35 initial variability composites from which 29 final variability composites, and three master composites were generated.

13.3 2018 Scouting Testwork Programme

13.3.1 Comminution Testwork

Comminution testwork was completed on the two comminution composites with results for this testwork (presented along with the 2019 results) in Table 13.5.1.

Head Assay and Gravity / Extraction Testwork

Testwork was completed on the individual variability composites and on three master composites made up from the variability composites.

Variability and master composite gold and silver head assays are summarised in Table 13.3.1. The variability in the triplicate gold head assays confirmed the presence of significant coarse gold. Silver head grades were generally below the 2 ppm detection limit. Detailed head assays on the master composites showed few deleterious elements for gold leaching with low levels of base metals and arsenic.

Gravity / extraction testwork was conducted with results summarised in Table 13.3.1and Figure 13.3.1. For the variability samples at a grind size of 80% passing (P_{80}) 75 µm, gravity recoverable gold (GRG) was 20% to 85% and overall gold extractions were greater than 96% with very consistent results. After removal of the gravity gold, leach kinetics were very fast with the bulk of gold leached within four to eight hours. Silver head and leach residue assays indicated >70% silver extraction. Reagent requirements were low for the fresh samples and low to moderate for the oxide samples.

The effect of grind size on gold extraction was evaluated on the master composites with results summarised in Table 13.3.1 and Figure 13.3.1. Generally, gold extraction increased as grind size decreased; however, extractions were only slightly higher at a grind size of P_{80} 75 µm than at P_{80} 106 µm. The economic optimum grind appeared to be P_{80} 106 µm with similar gold recoveries and lower operating costs when compared with P_{80} 75 µm.

2018 Rheology Testwork

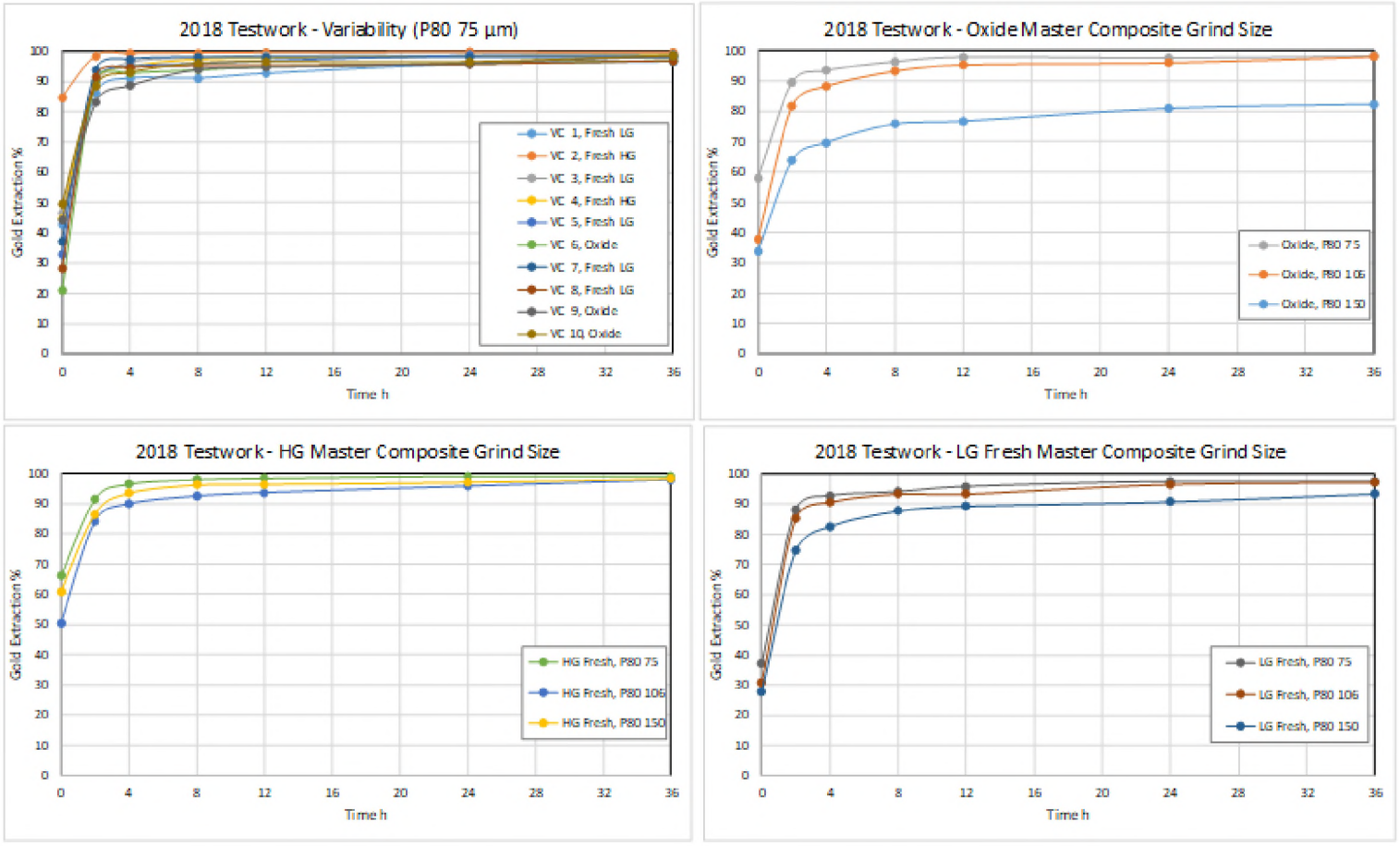
Rheology testwork was completed on the master composite slurries at a grind size of P_{80} 106 µm, pH 10.5 and slurry densities between 40 to 60% w/w solids.

The fresh composite had low to moderate viscosities and conventional agitators and pumps will be suitable. The oxide composite had extreme viscosity and would require dilution to less than 30% w/w solids or blending with fresh ore for processing. Rheology results for this testwork are presented with the 2019 results in Table 13.9.2.

Table 13.3.1 2018 Testwork - Head Assay and Gravity / Leach Results

Test No.	Sample ID	Facies	Grind Size	Head Assay		Calc. Head	Gold								Reagent Cons. kg/t		Silver		
				Triplicate	Avg		Residue	% GRG	Overall % Extraction @ Time hrs								Head Assay	Res. Grade	Ind. Ext'n*
			P80 µm	Au g/t	Au g/t	Au g/t	Au g/t		2	4	8	12	24	36	NaCN	Lime	Ag g/t	Ag g/t	% Ag
BF1171	VC 1	Fresh LG	75	0.64 / 0.66 / 0.53	0.61	0.88	0.03	43	86	91	91	93	96	96.6	0.25	0.25	0.6	<0.3	75
BF1172	VC 2	Fresh HG	75	7.35 / 7.54 / 6.23	7.04	6.51	0.03	85	98	99	99	100	100	99.5	0.15	0.25	8.1	<0.3	98
BF1173	VC 3	Fresh LG	75	0.86 / 1.30 / 1.22	1.13	0.95	0.02	47	89	96	98	99	99	97.9	0.22	0.25	0.6	0.2	67
BF1174	VC 4	Fresh HG	75	3.63 / 3.66 / 5.53	4.27	3.50	0.04	45	90	95	97	98	98	98.9	0.22	0.25	1.8	<0.3	92
BF1175	VC 5	Fresh LG	75	2.24 / 2.13 / 1.69	2.02	1.52	0.02	33	90	95	96	97	99	98.7	0.40	1.25	0.3	<0.3	50
BF1176	VC 6	Oxide	75	0.58 / 0.57 / 0.65	0.60	0.67	0.02	21	91	93	94	95	96	97.0	0.44	2.20	1.2	<0.3	88
BF1177	VC 7	Fresh LG	75	1.63 / 1.17 / 1.10	1.30	1.03	0.02	37	94	97	98	98	98	98.1	0.38	0.30	<0.3	<0.3	0
BF1178	VC 8	Fresh LG	75	1.29 / 1.11 / 3.02	1.81	1.23	0.04	28	92	95	95	95	96	96.8	0.39	0.30	0.3	<0.3	50
BF1179	VC 9	Oxide	75	2.03 / 2.12 / 3.90	2.68	2.01	0.03	44	83	89	94	95	96	98.5	0.58	1.80	22	3.0	86
BF1180	VC 10	Oxide	75	3.80 / 5.17 / 3.32	4.10	3.74	0.05	49	89	93	96	96	96	98.7	0.51	3.00	3.9	<0.3	96
BF1231	MC 1	Oxide	150			2.47	0.44	34	64	70	76	77	81	82.2	0.44	3.10		0.3	96
BF1232	MC 1	Oxide	106	1.73 / 1.26 / 1.92	1.64	2.03	0.04	38	82	88	93	95	96	98.0	0.40	3.05	8	0.3	96
BF1233	MC 1	Oxide	75			2.25	0.04	58	90	93	96	98	98	98.2	0.40	3.15		<0.3	98
BF1234	MC 2	HG Fresh	150			3.91	0.07	61	86	93	96	96	97	98.2	0.16	0.35		<0.3	85
BF1235	MC 2	HG Fresh	106	5.87 / 4.03 / 4.36	4.75	5.06	0.11	51	84	90	92	94	96	97.8	0.22	0.35	<2	<0.3	85
BF1236	MC 2	HG Fresh	75			5.76	0.05	66	91	96	98	98	99	99.1	0.17	0.35		<0.3	85
BF1237	MC 3	LG Fresh	150			1.82	0.12	28	75	83	88	89	91	93.4	0.39	0.75		<0.3	85
BF1238	MC 3	LG Fresh	106	1.54 / 1.86 / 1.90	1.77	1.07	0.03	31	85	91	93	93	97	97.2	0.36	0.80	<2	<0.3	85
BF1239	MC 3	LG Fresh	75			1.62	0.04	37	88	93	94	96	98	97.5	0.39	0.75		<0.3	85

Figure 13.3.1 2018 Testwork - Variability Gold Extractions and Effect on Master Composite Gold Extractions



13.4 2019 Programme Sample Selection

The comminution samples are listed in Table 13.4.1 and the metallurgical test samples in Table 13.4.2. The drill holes sampled provided good geographical coverage of the 2019 Mineral Resource with examples from the Lafigué North, Center and South zones. Location of the drills holes are shown in Figure 13.4.1, Figure 13.4.2, Figure 13.4.3 and Figure 13.4.4.

Samples selected were contiguous lengths of either quarter or half diamond core with drilling diameters varying with hole depth. Met Comp #17 was made up by combining two near contiguous intercepts to increase mass reducing the total number of metallurgical test composites to 35.

The samples covered the range of gold mineralisation styles with high grade coarse gold intercepts as well as disseminated low grade examples. The samples also provided examples of the oxide and fresh weathered states and different host lithologies. No oxide samples were suitable for comminution testing as the material is too fine for breakage or work index testing. Comm Comp #3 was a low mass intercept selected as a BWI variability sample. Meterage representing a single lithology was limited in the mineralised zones being at or near the contact between lithological hosts.

Table 13.4.1 2019 Testwork Comminution Samples

Composite ID	Drill Hole ID	From (m)	To (m)	Mass (kg)	Lithology
Comm #1	LFDD18-250	25	52.7	27.3	IMAF with intervals VMAF + QZ
Comm #2	LFDD18-400	57.3	80.25	21.6	VMAF
Comm #3	LFDD18-400	87.4	97.9	10.45	IMAF
Comm #4	LFDD18-400	118.9	125.95	20.3	VMAF
Comm #5	LFDD18-403	110.2	140.95	29.85	VMAF + IFEL +QZVN
Comm #6	LFDD18-407	168.5	189.8	24.5	contact VMAF/IMAF
Comm #7	LFDD18-410	35.3	70.5	32.1	Sheared Gabbro interleaved with QV
Comm #8	LFRCDD18-358	198.4	220	25.1	IMAF + VMAF + QZ + IFEL
Comm #9	LFRCDD18-358	279.2	308.95	38.3	VMAF + IFEL + IMAF
Comm #10	LFRCDD19-547	159.45	176.8	21.9	IMAF + VMAF intervals
Comm #11	LFRCDD19-667	66	77	44.3	IFEL
Comm #12	LFRCDD19-667	110	117	28.7	IMAF

Table 13.4.2 2019 Testwork Metallurgical Samples

Composite ID	Drill Hole ID	From (m)	To (m)	Mass (kg)	Site Assay Avg Grade (Au g/t)
Met#1	LFDD17-233	152	162.6	9.44	10.3
Met#2	LFDD17-243	199	205.4	6.60	1.8
Met#3	LFDD17-243	224	233.25	8.94	3.0
Met#4	LFDD18-252	82.15	90	6.82	1.4
Met#5	LFDD18-400	80.25	87.4	6.60	0.9
Met#6	LFDD18-400	97.9	106.95	8.70	1.2
Met#7	LFDD18-404	100.8	107.65	6.40	3.8
Met#8	LFDD18-404	111.2	115.7	4.28	0.9
Met#9	LFDD18-405	164	169.7	5.84	3.9
Met#10	LFDD18-409	115.1	126.85	11.48	3.1
Met#11	LFDD18-409	126.9	138.05	10.94	2.2
Met#12 [HG	LFDD18-410	173.8	182.1	7.54	29.8
Met#13 [HG]	LFDD18-410	182.1	193	10.26	24.4
Met#14	LFDD18-412	342.6	351	8.46	1.0
Met#15	LFRCDD18-359	300.4	304.75	4.56	7.2
Met#16	LFRCDD18-361	258.8	270.6	12.50	3.8
Met#17*	LFRCDD18-361 LFRCDD18-361	270.6 277.7	272.7 282.95	8.46	1.1
Met#18	LFRCDD18-361	305.4	315.4	11.52	1.7
Met#19	LFRCDD18-362	266.6	273.2	7.48	0.8
Met#20	LFRCDD19-441	245	252.72	6.52	5.2
Met#21	LFRCDD19-459	181.4	190	9.90	3.2
Met#22 [Oxide]	LFDD19-670	0	7.3	17.30	22.2
Met#23 [Oxide]	LFDD19-670	7.3	10.4	12.58	2.5
Met#24 [Oxide]	LFDD19-671	0.9	7.7	16.22	1.5
Met#25 [Oxide]	LFDD19-668	5.45	11.3	17.22	2.9
Met#26 [Oxide]	LFDD19-669	3.4	6.7	15.12	0.9
Met#27 [Oxide]	LFDD19-669	6.7	12.3	14.50	3.5
Met#28 [Oxide]	LFDD19-669	13.75	17.2	16.52	0.6
Met#29	LFRCDD19-667	62	66	15.78	0.8
Met#30	LFRCDD19-667	77	83.45	26.42	0.9
Met#31	LFRCDD19-667	83.45	90	26.04	1.9
Met#32	LFRCDD19-667	90	95	19.96	1.8
Met#33	LFRCDD19-667	95	99	15.56	8.9
Met#34	LFRCDD19-667	99	103.5	16.98	1.9
Met#35	LFRCDD19-667	103.5	109.25	22.78	3.7

*Met#17 consists of 2 samples in order to make up sufficient composite mass.

Figure 13.4.1 Lafigué Prospect Showing Geological Interpretation and Selected Intercepts per Domain

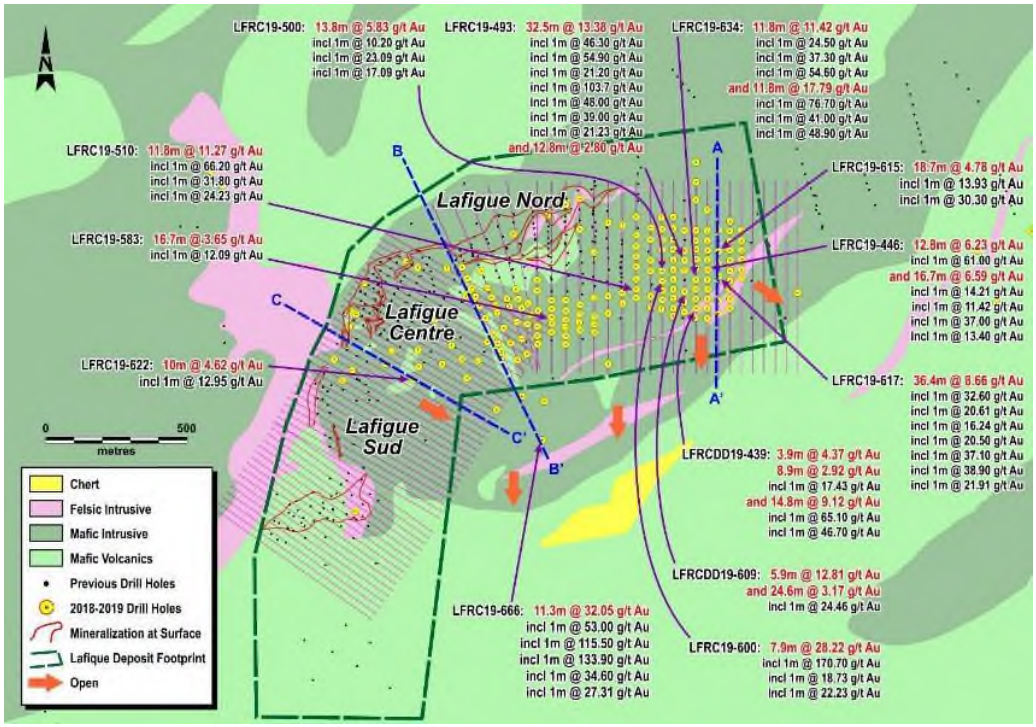


Figure 13.4.2 Metallurgical Sample Hole Locations with the Pit Outline

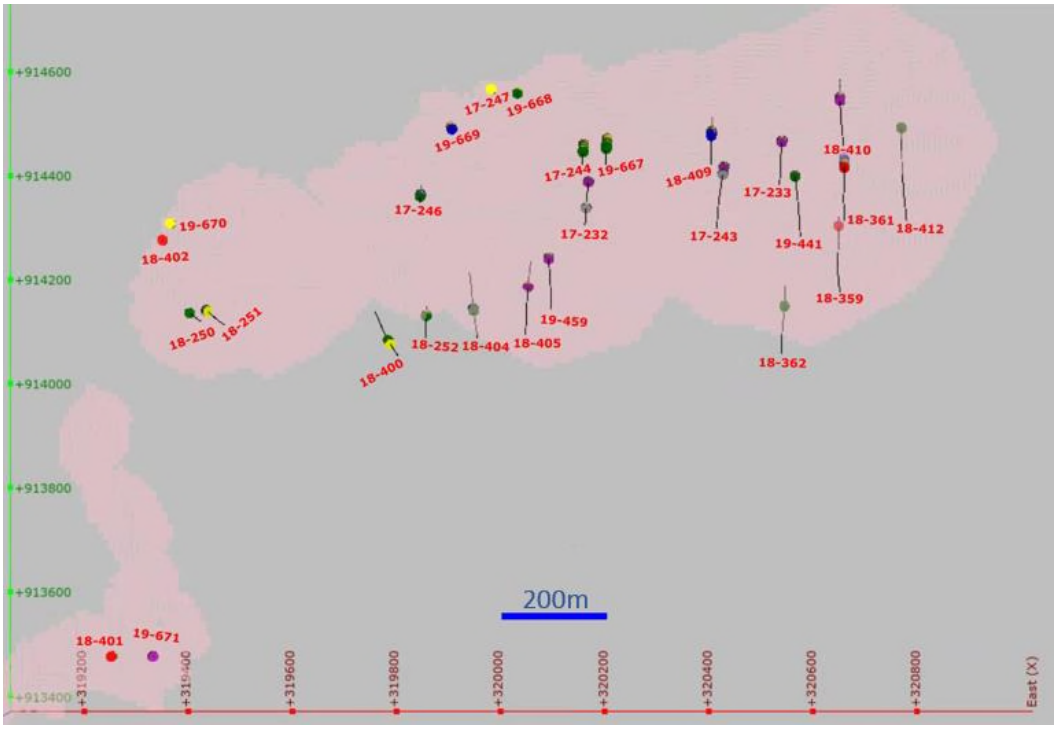


Figure 13.4.3 Oblique View of Lafigué from the Northeast Showing the Metallurgical Sample Hole Locations, Preliminary Pit Shell and Mineralisation

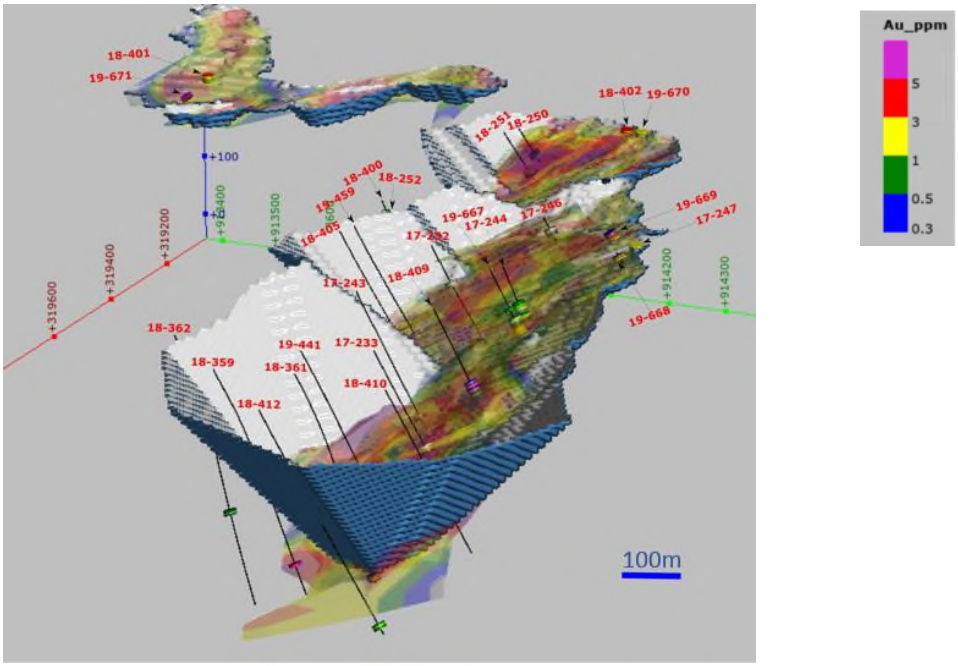
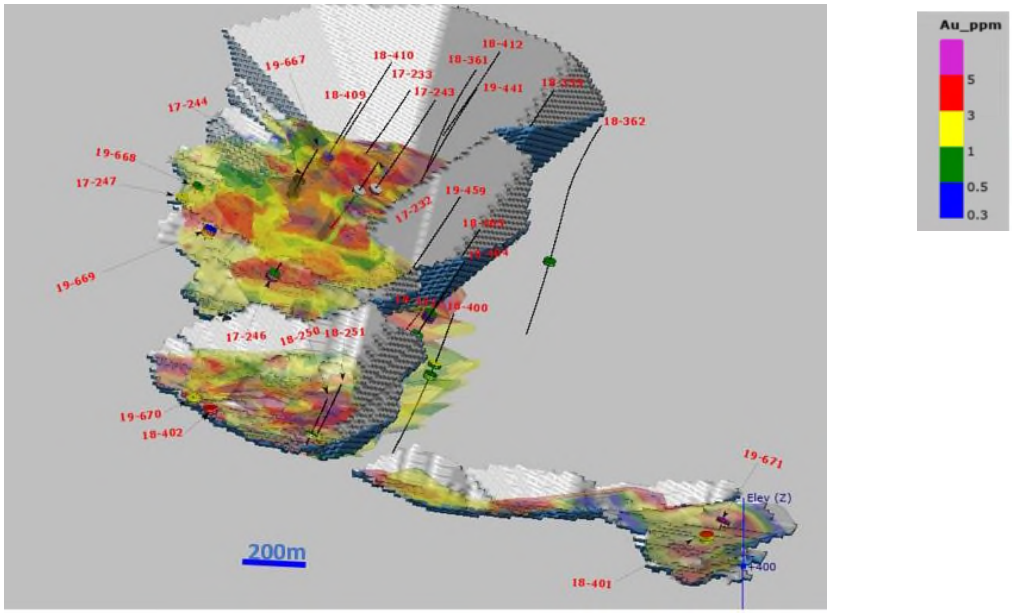


Figure 13.4.4 Oblique View of Lafigué from the Southwest Showing the Metallurgical Sample Hole Locations, Preliminary Pit Shell and Mineralisation



13.5 2019 Programme Comminution Testwork

The purpose of the 2019 comminution testwork programme was to determine the comminution parameters to allow design of a crushing and milling circuit appropriate for the plant throughput and feed type. The following comminution testwork was completed:

- SMC tests (@ -22.4+19.0 mm fraction).
- Bond Abrasion Index (Ai) determination.
- Bond Rod Mill Work Index (BWi) determination (@106 µm closing screen to yield P_{80} of 1,180 µm).
- Bond Ball Mill Work Index (BWi) determination (@106 µm closing screen to yield P_{80} of 75 µm).

The comminution test results for the 2019 samples are summarised in Table 13.5.1, along with the results for the 2018 samples. Composite 3 is a transition sample and was too fine for SMC testwork. No oxide samples were selected for comminution testing as the oxide ore is too fine for breakage or work index testing.

Table 13.5.1 2018 and 2019 Comminution Results

Sample Designation		SMC Parameters				Ai	RWi kWh/t	BWi kWh/t
		SG	DWi kWh/m³	Axb	ta			
2019 Composites								
Comp 1	IMAF+VMAF/QZ	2.87	7.9	36.4	0.3	0.0409	-	13.6
Comp 2	VMAF	2.82	9.3	30.7	0.3	0.0831	-	15.8
Comp 3	IMAF (Transition)					-	-	16.2
Comp 4	VMAF	2.88	11.0	25.9	0.2	0.0717	-	16.6
Comp 5	VMAF+IFEL+QV	2.78	9.5	29.3	0.3	0.228	-	19.6
Comp 6	VMAF/IMAF contact	2.79	8.9	31.2	0.3	0.228	-	15.7
Comp 7	Sheared Gabbro+QV	2.75	10.4	26.1	0.3	0.2746	18.8	17.2
Comp 8	IMAF+VMAF+QZ+IFEL	2.81	8.1	34.7	0.3	0.0259	-	14.4
Comp 9	VMAF+IFEL+IMAF	2.84	8.0	35.7	0.3	0.142	20.7	16.0
Comp 10	IMAF+VMAF	2.97	10.1	29.7	0.3	0.0584	-	13.5
Comp 11	IFEL	2.72	10.6	26.0	0.3	0.3976	18.4	16.4
Comp 12	IMAF	2.83	6.9	41.2	0.4	0.035	15.7	12.4
2018 Composites								
Comp. #1	Mafic / QV	2.66	4.7	56.9	0.55	0.089		13.9
Comp. #2	Felsic	2.63	8.2	32.0	0.32	0.349		16.8

As shown in Table 13.5.1, the comminution results are reasonably consistent. The Axb values are low and the DWi values are high (90th percentile in the JKTech database) indicating that the fresh ore is competent with a high breakage energy requirement. The RWi and BWi are not as extreme but are still moderately high. The abrasion index is low to moderate.

The comminution results were provided to Orway Mineral Consultants (OMC) for comminution circuit selection and equipment sizing. The selection of the comminution circuit includes the key design criteria used in modelling of the comminution circuit, the equipment for the selected circuit and the comminution power and consumables estimates for use in the operating cost estimate.

13.6 2019 Programme Master Composite Samples

13.6.1 Formation of Ore Type Master Composites

The consistent metallurgical response from the variability samples in the 2018 scouting testwork testing phase showed that all samples from each weathered state could be considered equally representative and combined to form the master composites.

The fresh master composite was the main composite on which the optimisation testwork would be completed given that it represents the majority of the ore (85% for the 2019 Mineral Resource) and is the driver for the process flowsheet design. The fresh master composite was made up from equal sub-sample masses (3 kg per intercept composite) from intercept Composites #1-4, #6, #10 and 11, #14, #16-19, #21 and #29-35. This composite had an expected gold grade of 2.75 g/t based on the averaged site assays. Several low mass composites were not included in the fresh master composite and were set aside for variability testing only. Two high grade composites were excluded from the fresh master composite in order to achieve the target gold grade.

The high-grade master composite was made up from equal sub-sample masses (6 kg per intercept composite) from intercept Composites #12-13. This composite had an expected gold grade of 2.71 g/t based on the averaged site assays.

The oxide master composite was made up of equal sub-sample masses (7 kg per intercept composite) from intercept Composites #22-28. This composite had an expected gold grade of 4.9 g/t based on the averaged site assays.

The remaining portion of the composites and unused composites were available for variability testwork.

13.6.2 Master Composite Head Assay

Multi-element head assays were determined for each of the composite samples and a mineralogical investigation was conducted on the fresh ore composites.

Triplicate gold analyses were performed by standard fire-assay on the master composite with the balance of the elements being determined by ICP scan or standard assay techniques. Master composite key elements are summarised in Table 13.6.1.

The variance in the triplicate gold assays suggests the presence of significant coarse gold or highly localised fine gold concentrations. This presence of coarse gold was confirmed in subsequent gravity testwork.

The fresh and oxide master composite average gold grades are typical of the expected annual gold grade ranges. The high-grade fresh composite did not have the expected elevated sulphide and other metal grades typically associated with high-grade gold; this composite became an additional variability sample.

Silver assay head grades are below the 2 g/t detection limit. Silver extraction rates were not generally monitored in the testwork given the low head grades.

There are no deleterious elements for gold leaching present with low levels of base metals, antimony and tellurium. Mercury and arsenic levels are low and should not present an environmental or occupational health risk in the elution or electrowinning circuits.

Organic carbon levels were low for all composites (<0.03% C_{org}) and preg-robbing due to the presence of organic carbon is not expected to occur.

Table 13.6.1 2019 Testwork Master Composite Head Assay

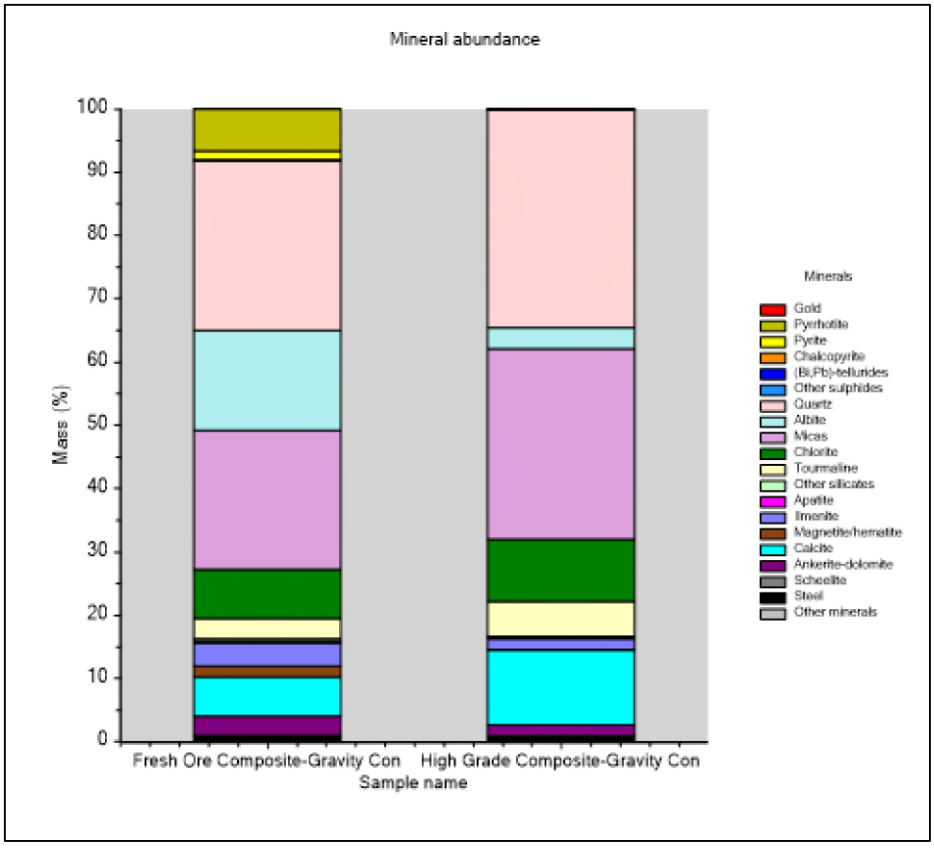
Element	Master Composite		
	Fresh	High-Grade Fresh	Oxide
Au 1 (g/t)	2.53	56.5	3.57
Au 2 (g/t)	1.78	22.4	2.41
Au 3 (g/t)	1.83	19.6	3.19
Avg Au (g/t)	2.05	32.8	3.06
Ag (ppm)	<2	<2	<2
As (ppm)	<10	<10	<10
Corg (%)	0.03	<0.03	0.06
Cu (ppm)	66	44	90
Fe (%)	4.98	6.58	8.36
Hg (ppm)	<0.1	0.5	<0.1
S (%)	0.42	<0.02	0.04
S^{2-} (%)	0.26	<0.02	0.02
Sb (ppm)	0.2	0.2	0.2
SiO_2 (%)	56.6	56.8	61.8
Te (ppm)	0.4	4.8	0.4
Zn (ppm)	116	114	112
True SG	2.79	2.81	2.79

13.6.3 Mineralogical Analysis – Fresh and High-Grade Fresh Master Composites

Fresh and high-grade fresh master composite sub-samples were ground to P_{80} 75 µm and submitted for quantitative mineralogical analysis by QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy) and XRD (X-ray diffraction). Each sample was separated into gravity concentrate and gravity tail fractions using centrifugal concentration and hand-panning before being prepared for mineralogical investigation.

Minor amounts of pyrrhotite and pyrite were detected in the fresh master composite gravity concentrate fraction as shown in Figure 13.6.1. These sulphides occur as moderately well liberated particles. Hardly any sulphide minerals (pyrrhotite, pyrite, etc.) were detected in the gravity concentrate fraction of the high-grade fresh composite.

Figure 13.6.1 2019 Testwork Gravity Concentrate Mineralisation



Silicates and carbonates make-up most of the remaining sample mass for each gravity concentrate fraction. The gravity tail fractions are similarly mainly silicates and carbonate minerals. Biotite, quartz, chlorite, muscovite and albite are the main silicates. Calcite is the main carbonate mineral.

Optical microscopy and QEMSCAN analysis were used to detect gold grains in the concentrates. Some coarse free/liberated grains were found in both the concentrates (Figure 13.6.2), but a number of the grains were partially or totally encapsulated. Eleven grains were detected in the master composite and 56 grains in the high-grade composite. The largest free gold grain is approximately 1.5 mm in size (detected in the high-grade composite) and the remaining grains ranged in size from 2 µm to 500 µm.

The smaller grains in the master composite were exclusively pyrite/pyrrhotite hosted (frequently encapsulated), but those in the high-grade composite were mainly associated with the bismuth tellurides (Bi, Pb Te). Examples are presented in Figure 13.6.3.

Figure 13.6.2 2019 Testwork Coarse Gold Mineralisation

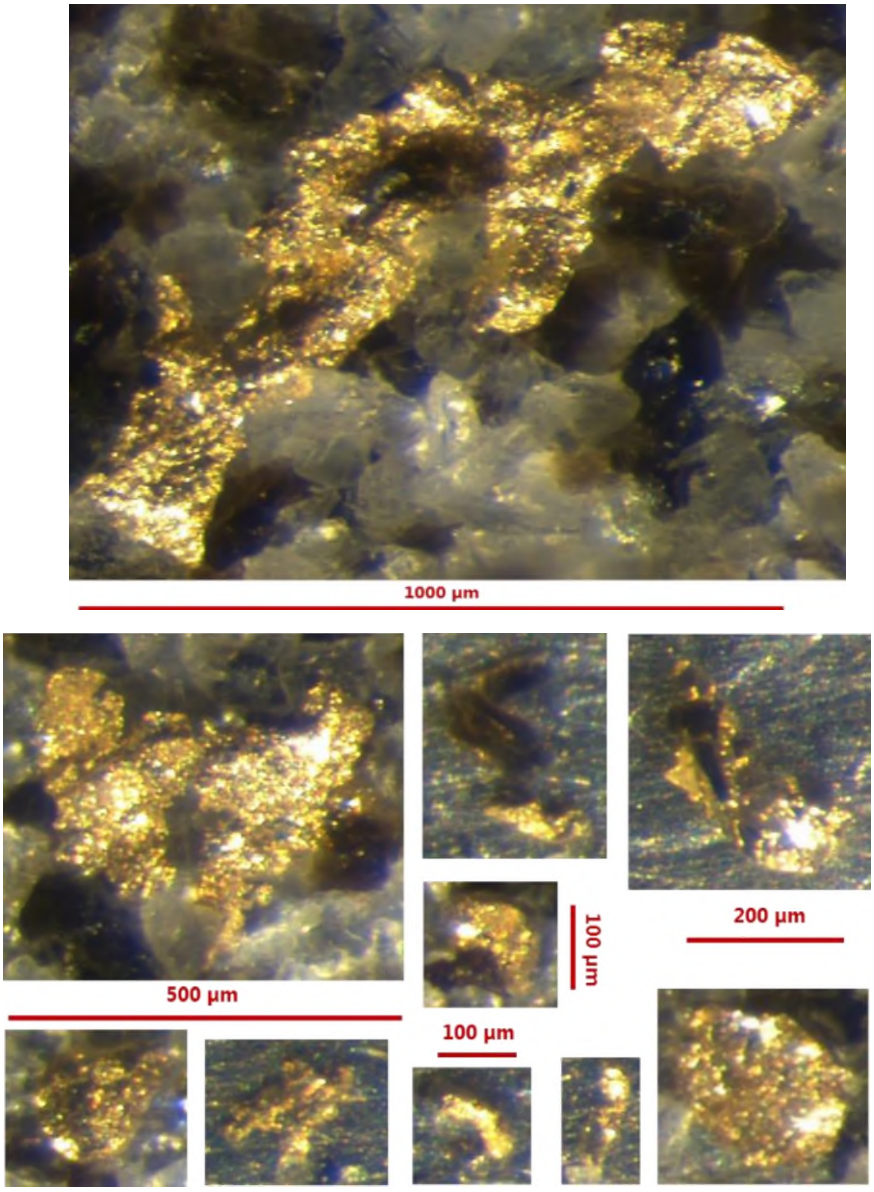
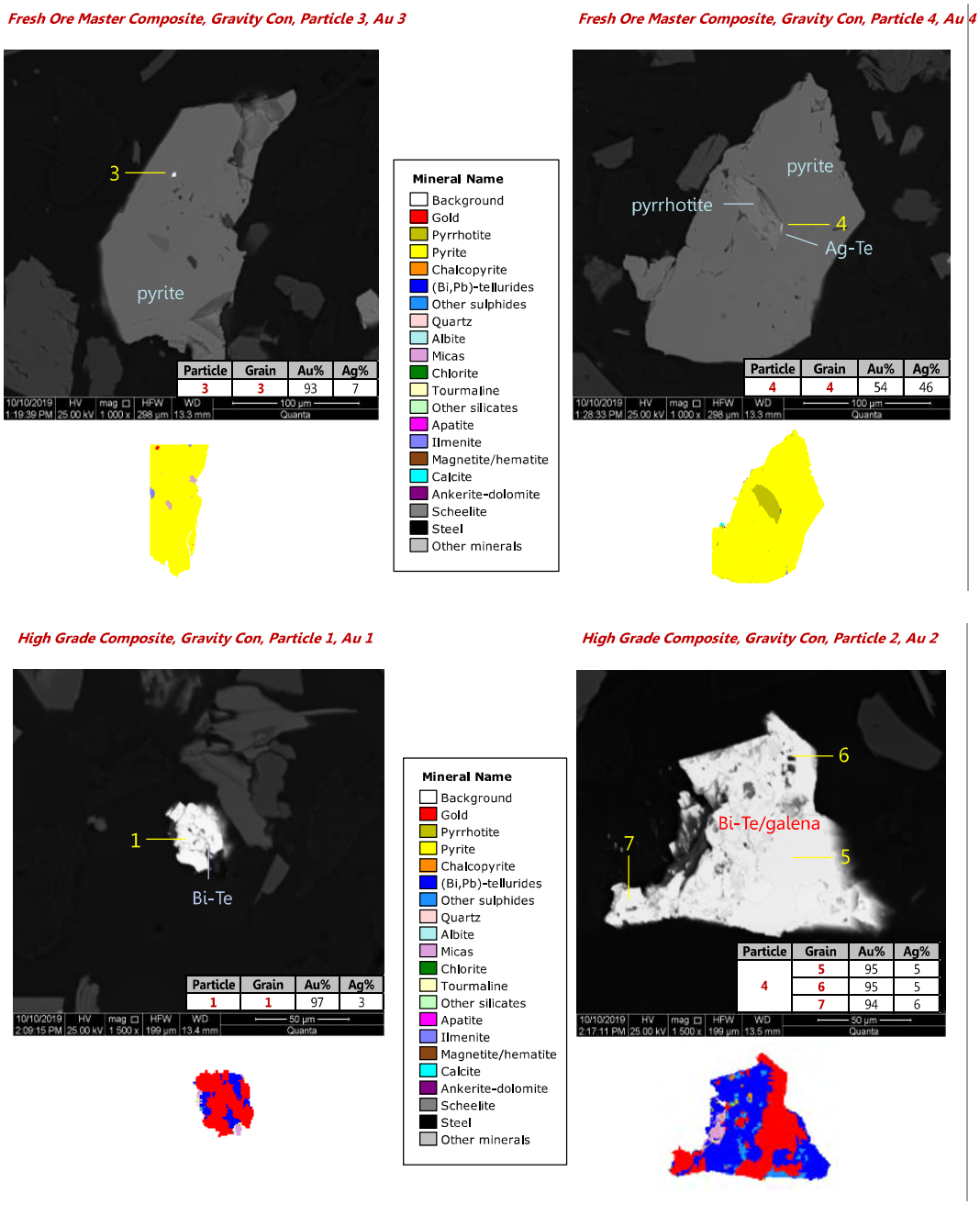


Figure 13.6.3 2019 Testwork Locked Gold Mineralisation



13.7 2019 Programme Optimisation Testwork

13.7.1 Master Composite Grind Optimisation Testing

Testwork to determine the optimum grind size was carried out on the fresh and oxide master composites. Grind / extraction tests with gravity prior to leaching were completed at grind sizes of P_{80} 125, 106, 90 and 75 µm to determine the effect of grind size on gold extraction.

The tests on the fresh master composite were completed at two cyanide concentrations with the test outcomes relatively unaffected by the cyanide concentration. The grind / gravity / cyanidation gold and silver extraction results are summarised in Table 13.7.1 and Figure 13.7.1. The fresh composite results typically indicate:

- Gravity gold content is high (60 - 70%) with significant variation between the assayed head and calculated head gold grades. Gravity recovery appeared to be lower for the coarser grind (125 µm) suggesting that more free gold was liberated at the finer grinds.
- The fresh composite achieved high leach extractions (>97%) across the size range tested with some increase in leach kinetics and gold extraction as grind size decreased. Gold extraction was essentially complete within eight to 12 hours.
- Reagent consumption was low and similar across the grind size range tested.
- Silver head grades were very low (0.6 g/t Ag) and silver extractions average 75% and were not impacted by grind size. Silver extraction was not tracked in subsequent testing due to the low head grades and low added value to the project.

Testwork on the oxide composite was completed at grind sizes of P_{80} 75 and 106 µm only as oxide, being a minor component of the feed blend, will not be grind determining. The results indicate:

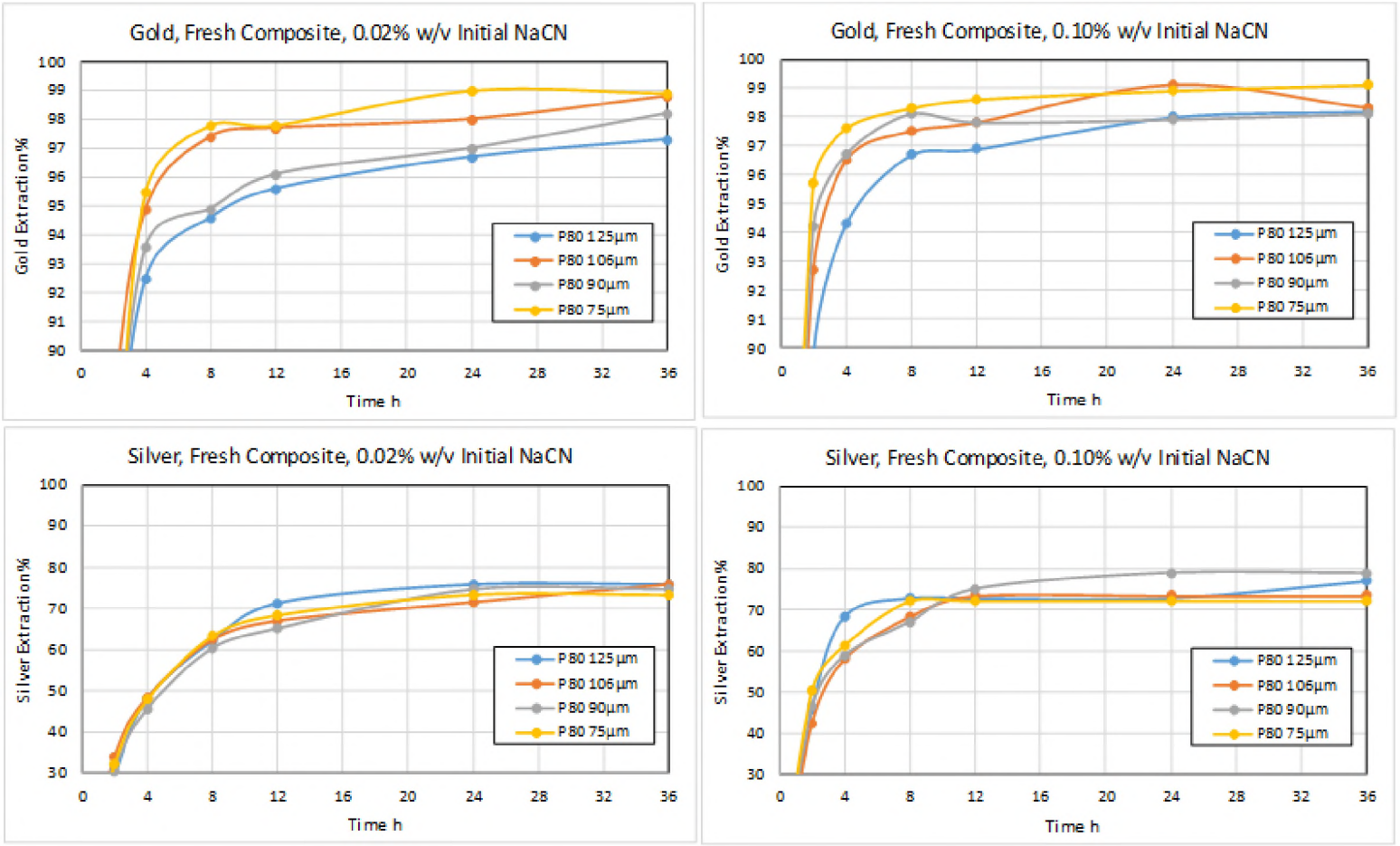
- Oxide leach kinetics were faster than the fresh ore with slightly higher overall gold extraction (99%). This composite is considered insensitive to grind size on the basis of these test results. The oxide tests were also conducted at two cyanide concentrations with similar results.

A grind optimisation economic evaluation was conducted and concluded that the optimum grind size is P_{80} 75 to 106 µm. A grind size of P_{80} 106 µm was selected for further testwork.

Table 13.7.1 2019 Testwork Master Composite Gravity Leach Grind Optimisation Tests

Master Composite	Testwork	Test No.	Grind Size P_{80} µm	Initial NaCN % w/v	Leach % Solids (w/w)	Sparge Gas	Assay Head Au g/t	Calc Head Au g/t	Residue Au g/t	% GRG	Overall % Gold Extraction @ Time hours						Reagent Cons (kg/t)	
											2	4	8	12	24	36	NaCN	Lime
Fresh	Grind Opt Low NaCN	JS4483	125	0.02	40	O_2	2.05	2.04	0.06	61	83	93	95	96	96.7	97.3	0.22	0.25
		JS4484	106	0.02	40	O_2	2.05	2.51	0.03	68	88	95	97	98	98.0	98.8	0.18	0.25
		JS4485	90	0.02	40	O_2	2.05	1.66	0.03	58	83	94	95	96	97.0	98.2	0.12	0.35
		JS4486	75	0.02	40	O_2	2.05	1.87	0.02	64	83	96	98	98	99.0	98.9	0.20	0.30
	Grind Opt Higher NaCN	JS4483R	125	0.10	40	O_2	2.05	2.74	0.05	60	90	94	97	97	98.0	98.2	0.28	0.30
		JS4484R	106	0.10	40	O_2	2.05	2.71	0.05	70	93	97	98	98	99.1	98.3	0.31	0.25
		JS4485R	90	0.10	40	O_2	2.05	2.62	0.05	69	94	97	98	98	97.9	98.1	0.28	0.26
		JS4486R	75	0.10	40	O_2	2.05	2.33	0.02	70	96	98	98	99	98.9	99.1	0.31	0.27
Oxide	Grind Opt	JS4489R	106	0.10	40	O_2	3.06	3.54	0.04	74	94	97	99	99	98.9	98.9	0.20	3.35
		JS4490R	75	0.10	40	O_2	3.06	4.62	0.03	81	97	99	99	99	99.6	99.4	0.18	3.15
							Ag g/t	Ag g/t	Ag g/t		Overall % Silver Extraction @ Time hours							
Fresh	Grind Opt Low NaCN	JS4483	125				<2	0.6	0.15	0	29	48	62	71	75.8	75.7		
		JS4484	106				<2	0.6	0.15	0	34	48	62	67	71.4	75.7		
		JS4485	90				<2	0.6	0.15	0	31	46	60	65	74.5	74.5		
		JS4486	75				<2	0.6	0.15	0	32	48	63	68	73.3	73.2		
	Grind Opt Higher NaCN	JS4483R	125				<2	0.7	0.15	0	46	68	73	73	72.9	77.0		
		JS4484R	106				<2	0.6	0.15	0	43	58	68	73	73.5	73.4		
		JS4485R	90				<2	0.7	0.15	0	46	59	67	75	78.9	78.9		
		JS4486R	75				<2	0.5	0.15	0	50	61	72	72	72.1	72.0		

Figure 13.7.1 2019 Testwork – Effect of Grind Size on Gold and Silver Extraction for Fresh Master Composite



13.7.2 Bulk Gravity Gold Separation and Preparation of Bulk Gravity Tails

Bulk gravity separation testwork was performed on the master composites to evaluate whether a larger sample mass impacted on gravity gold recovery and to generate gravity tailings samples with a more consistent head grade for downstream leach optimisation testwork.

The bulk gravity tests were conducted on 30 kg sub-samples of each ore composite at a grind size of P_{80} 106 µm using the laboratory Knelson concentrator for gravity separation. The Knelson concentrate (0.2% of feed mass) was then subjected to intensive cyanidation for 24 hours with 5.0% w/v starting NaCN, pH 12 and Leachwell addition. This approach is more aligned with what will occur in plant practice. The intensive leach solution was submitted for gold assay, whilst the intensive leach residue was recombined with the bulk gravity tailings. The bulk gravity tailings were thoroughly homogenised and split into sub-samples for the downstream testwork program.

The high gravity gold recoveries achieved in the previous work also occurred in these larger mass tests. The gravity gold recoveries from this stage (Table 13.7.2) were incorporated into the metallurgical balances for the subsequent gravity tails leach test series.

Table 13.7.2 2019 Testwork Master Composite Bulk Gravity Test Work

Sample ID	Test No.	Head Assay, Au (g/t)	Calc'd Head, Au (g/t)	Gravity Tails Assay, Au (g/t)	% GRG Au Extr'n
Fresh Ore Master Comp	JS4491	2.53/1.78/1.83	3.03	0.73/0.59/0.71	77.7
Fresh Ore High-Grade Comp	JS4492	56.5/22.4/19.6	15.7*	Not assayed*	91.5
Oxide Ore Master Comp	JS4493	3.57/2.41/3.19	4.66	1.17/1.23/1.27	73.8

*Gravity tails sample not assayed as expected to be high grade; sample calculated head determined from concentrate assay and subsequent leach testwork solution and tails assays.

13.7.3 Master Composite Leach Optimisation Testwork

Gold extraction versus cyanide concentration, slurry density and air/oxygen tests were conducted on the fresh and oxide bulk gravity tailings at the selected grind size of P_{80} 106 µm. Optimisation testwork was focussed on the fresh ore composite as it represents the majority of the Mineral Resource. The optimisation test series results are included in Table 13.7.3.

The leach optimisation results are discussed in more detail in the following sections with graphical presentation of the key results for comparative purposes.

Table 13.7.3 2019 Testwork Master Composite Leach Optimisation Testwork

Master Composite	Testwork Series	Test No.	Grind Size	NaCN Conc		Leach Solids	Sparge Gas	Assay Head	Calc Head	Residue	% GRG	Overall % Gold Extraction @ Time hrs						Reagent Cons kg/t	
				Initial	Maint.														
			P_{80} µm	% w/v	% w/v	% w/w		Au g/t	Au g/t	Au g/t	0	2	4	8	12	24	36	NaCN	Lime
Fresh	Grind Opt Repeat	JS4483R	125	0.10	0.05	40%	O_2	2.05	2.74	0.05	60	90	94	97	97	98.0	98.2	0.28	0.3
		JS4484R	106	0.10	0.05	40%	O_2	2.05	2.71	0.05	70	93	97	98	98	99.1	98.3	0.31	0.25
		JS4485R	90	0.10	0.05	40%	O_2	2.05	2.62	0.05	69	94	97	98	98	97.9	98.1	0.28	0.26
		JS4486R	75	0.10	0.05	40%	O_2	2.05	2.33	0.02	70	96	98	98	99	98.9	99.1	0.31	0.27
	NaCN Conc Opt.	JS4493	106	0.04	0.02	40%	O_2	2.05	2.98	0.04	79	94	97	98	98	98.4	98.7	0.14	0.25
		JS4494	106	0.02	0.02	40%	O_2	2.05	3.00	0.06	79	91	95	97	97	98.0	98.0	0.11	0.25
		JS4495	106	0.02	0.015	40%	O_2	2.05	2.97	0.05	79	91	95	97	98	98.5	98.3	0.07	0.25
	Slurry Density Opt	JS4496	106	0.10	0.05	50%	O_2	2.05	2.99	0.04	79	95	97	98	98	98.4	98.7	0.2	0.2
		JS4497	106	0.10	0.05	55%	O_2	2.05	2.99	0.04	79	95	98	98	98	98.8	98.7	0.19	0.15
		JS4498	106	0.10	0.05	60%	O_2	2.05	3.03	0.06	78	94	96	97	97	97.5	98.0	0.21	0.13
	Air vs. O2	JS4499	106	0.10	0.05	40%	Air	2.05	2.99	0.05	79	97	97	98	98	98.3	98.3	0.29	0.25
	Bulk Leach	JS4501	106	0.025	0.02	55%	Air	2.05	3.00	0.06	78	88	93	96	97	98.0		0.08	0.23
	No Gravity	JS4588	106	0.025	0.02	55%	Air	2.05	1.50	0.10	0	18	29	51	66	86.2	93.3	0.09	0.3
Oxide	Grind Opt Repeat	JS4489R	106	0.10	0.05	40%	O_2	3.06	3.54	0.04	74	94	97	99	99	98.9	98.9	0.2	3.35
		JS4490R	75	0.10	0.05	40%	O_2	3.06	4.62	0.03	81	97	99	99	99	99.6	99.4	0.18	3.15
	NaCN Conc Opt.	JS4505	106	0.04	0.02	40%	O_2	3.06	4.73	0.04	73	95	97	99	98.7	99.2	99.2	0.10	2.90
		JS4506	106	0.02	0.02	40%	O_2	3.06	4.75	0.05	72	92	96	97	97.9	98.8	98.9	0.10	2.70
		JS4507	106	0.02	0.015	40%	O_2	3.06	4.71	0.05	73	93	96	98	98.4	98.2	98.9	0.07	2.70
	Slurry Density Opt	JS4508	106	0.10	0.05	45%	O_2	3.06	4.70	0.04	73	95	97	98	98.5	98.8	99.1	0.24	3.00
		JS4509	106	0.10	0.05	35%	O_2	3.06	4.71	0.03	73	96	98	99	99.0	99.4	99.4	0.32	2.70
		JS4510	106	0.10	0.05	32%	O_2	3.06	4.84	0.04	71	94	98	99	98.6	99.2	99.2	0.43	3.00
	Air vs. O_2	JS4511	106	0.10	0.05	40%	Air	3.06	4.72	0.06	73	92	96	97	98	98.4	98.7	0.30	3.15
	Bulk Leach	JS4513	106	0.025	0.02	40%	Air	3.06	4.78	0.13	72	92	94	95	96	97.3		0.25	2.46
High Grade Fresh	Air vs. O2	JS4502	106	0.025	0.02	55%	Air	32.8	15.70	0.05	92	98	99	99	100	99.7		0.05	0.20
		JS4503	106	0.025	0.02	55%	O_2	32.8	15.7	0.06	92	99	99	99	99	99.6		0.01	0.20

Cyanide Concentration Test Series

As cyanide usage was low in all tests and the initial grind series with low starting cyanide showed that gold extraction was similar, the following cyanide test series was completed on the fresh and oxide master composite bulk gravity tails:

- Initial cyanide strength of 0.040% w/v NaCN maintained at 0.020% at sampling intervals (400/200 ppm).
- Initial cyanide strength of 0.020% w/v NaCN maintained at 0.020% at sampling intervals (200/200 ppm).
- Initial cyanide strength of 0.020% w/v NaCN maintained at 0.015% at sampling intervals (200/150 ppm).

All tests were conducted at 40% solids with oxygen sparging. The tests indicated that gold extraction was not sensitive to cyanide concentration over the range tested. Overall gold extractions were between 98% and 99% irrespective of initial cyanide concentration. For the fresh ore, cyanide consumption decreased from 0.31 kg/t to 0.07 kg/t as the initial cyanide concentration was decreased showing that increased NaCN concentrations drive the consumption higher with more side reactions.

Based on the results from the test series, a slightly conservative initial cyanide concentration of 0.025% w/v NaCN was selected as the optimum addition rate.

Slurry Density Test Series

The impact of leach slurry density on gold leach kinetics and overall extraction for the fresh and oxide bulk gravity tails was tested. Slurry densities trialled were:

- Fresh - 40% (from grind optimisation series), 50%, 55% and 60% w/w solids.
- Oxide - 32%, 35%, 40% (from grind optimisation series) and 45% w/w solids.

Higher percent solids were tested for the fresh ore since it was planned to thicken the leach feed. The oxide ore was tested over a lower percent solids range in case there were slurry viscosity issues as slurry density increased. All tests were conducted at 0.1% initial NaCN concentration with oxygen sparging.

Overall gold extractions were very similar for all testwork suggesting that slurry density had little impact on leaching efficiency.

For the fresh ore, a slurry density of 55% solids was selected as the preferred leach operating point. For the oxide ore, as slurry density of 40% solids was maintained, given the possible high viscosity for some of the oxide ore types (as noted for one sample in the 2018 rheology testwork).

Air vs. Oxygen Test Series

All cyanidation tests to this point were conducted with oxygen sparging. To evaluate the benefits of oxygen (O_2) sparging, air sparging tests were carried out at 40% solids and an initial 0.1% NaCN addition, to allow comparison with the grind optimisation testwork results.

While oxygen sparging marginally improved the initial leach kinetics, final gold extraction was essentially similar and complete after 24 hours of leaching in both cases. This testwork confirmed that air sparging of the leach slurry would be satisfactory.

Gravity Tails Leach Optimisation Conclusions

Recommended leaching conditions were thus:

- A cyanide concentration of 0.025% w/v NaCN (allowed to decay to 0.02%)
- A leach feed slurry density of 55% w/w solids for fresh ore and 40% w/w solids for oxide ore. In practice, if oxide is blended with low viscosity oxide or fresh ore, higher leach densities may be suitable
- Air sparged leaching with a residence time of 24 hours.

13.7.4 Master Composite Bulk Leach Testwork

Demonstration tests using the optimised leach conditions were run as bulk leaches on the fresh and oxide bulk gravity tails to prepare leach tails samples for engineering design parameter testwork. The bulk leach testing results are summarised in Table 13.7.3. The results indicate that:

- For both the fresh and oxide samples, the bulk leach kinetics were slower and the final gold extraction (24 hours) was up to 1% lower.
- For the fresh ore, a higher cyanide concentration (as per the air sparging and density optimisation tests) would have improved leach kinetics and final gold extraction.
- For the oxide ore, a higher cyanide concentration or oxygen sparging (as per the air sparging and cyanide optimisation tests) would have improved leach kinetics and final gold extraction.

The bulk leach tails were utilised for downstream testwork including sequential triple contact carbon loading testwork, viscosity testwork, cyanide detoxification testwork and vendor thickener testwork on the fresh master composite.

A blended master composite slurry (40% oxide/60% fresh) was made-up from the fresh and oxide bulk leach tails. This composite was utilised for downstream viscosity testwork, cyanide detoxification testwork and vendor thickener testwork.

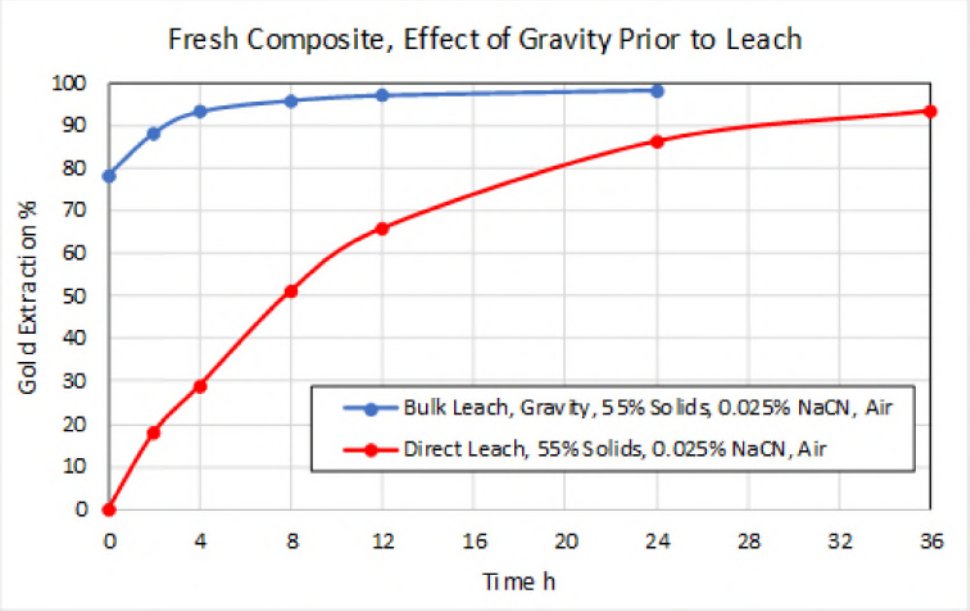
Bulk solution assays for the fresh and blended leach tails samples are presented in Table 13.9.4, being the feed to the cyanide detoxification testwork.

13.7.5 Master Composite Direct Leach (No Gravity) Testwork

The effect of a gravity concentration stage on overall gold extraction was evaluated to confirm that this was an essential addition to the flowsheet.

A whole of ore leach or direct leach test (without removal of the gravity gold prior to leaching) was conducted on the fresh master composite. The test results are included in Table 13.7.3 and shown in Figure 13.7.2. The bulk gravity leach test is included for comparison.

Figure 13.7.2 2019 Testwork Fresh Master Composite Whole of Ore Leach Testwork



Without removal of the coarse gravity gold, the leach profile is very slow with a lower overall recovery after 36 hours than was achieved in 12 hours with gravity gold removal. Cyanide and lime consumptions were comparable with and without gravity gold recovery.

This result strongly supports the inclusion of a robust gravity recovery stage and operation of this circuit at all times.

13.7.6 High-Grade Fresh Master Composite Gravity Leach Testwork

A demonstration test utilising the optimised leach conditions was conducted on the high grade fresh composite. This composite was selected assuming that the very high-grade samples would prove more challenging to leach than the average ores because the mineralogy and associated elements might require more intensive treatment. In practice as demonstrated by the mineralogical examination, this sample is a clean, free milling gold sample with no deleterious elements and is likely typical of the Lafigué high grade mineralisation.

The high-grade fresh composite gravity / leach testing results are summarised in Table 13.7.3.

The high-grade fresh composite leach test results indicate that the gold content comprised almost all free grains (92% gravity gold recovery) with little gold locked in background mineralisation and almost identical leach residues. Air or oxygen sparging provided similar very high (>99%) gold extractions.

A reasonable fraction of the gold mineralisation in the Lafigué Mineral Resource occurs as high-grade intercepts. This result suggests that very high recoveries and low reagent consumption is likely for this style of mineralisation.

13.8 2019 Programme Variability Testwork

Confirmatory testing using the optimised leach conditions established for the master composites was conducted on the 23 fresh and six oxide variability composites which had been selected to represent the various ore domains, weathered states, grade ranges and mineralisation styles of the 2019 Resource.

Variability composite samples for testing included most of the intercept composites selected on site for metallurgical testing. The majority of the composites were used to make up the master composites as described in Section 13.6.1 with a number of selected samples being set aside for variability testing only. Intercept Composites #2 and #4 were combined to make variability Composite #2A. Intercept Composites #12 and #13 made up the HG (high grade) fresh composite. Intercept Composites #27, #30 and #31 were excluded as these came from holes already represented by multiple samples.

13.8.1 Variability Composite Head Assays

Triplicate gold analyses were performed by standard fire-assay with the balance of the elements being determined by ICP scan or standard assay techniques. Variability composite details and key element assays are summarised in Table 13.8.1.

The fresh variability samples cover a gold grade range from 0.55 g/t Au to 7.8 g/t Au (excluding the HG composite). Some samples had reasonably consistent triplicate gold assays suggesting more disseminated fine gold but most of the samples had significant variation in the triplicate assay due to the presence of coarse gold. Deleterious element levels are generally low with very low base metal content. Silver grades were all below the minimum detection limit of 2 ppm.

The oxide variability samples selected have two high grade intercepts at 22.8 g/t Au (#22) and 7.1 g/t Au (#25) with the rest of the sample grades ranging from 1.1 g/t Au to 1.7 g/t Au. Deleterious element levels are generally low. Silver grades were below the minimum detection limit of 2 ppm with the exception of high-grade sample #22 with 6 g/t Ag.

Table 13.8.1 2019 Testwork Variability Composite Detailed Head Assay

Variability ID	Drill Hole ID	From	To	Head Assay											
				Au1-3	Au Ave	As	C org	Cu	Fe	Hg	Ni	S^{2-}	Sb	Te	Zn
		m	m	ppm	ppm	ppm	%	ppm	%	ppm	ppm	%	ppm	ppm	ppm
Var #1	LFDD17-233	152	162.6	3.52/3.72/2.66	3.3	<10	0.03	60	6.3	0.2	100	<0.02	0.2	2	120
Var #2a	LFDD17-243 LFDD18-252	199 82.1	205.4 90	3.37/1.93/2.28	2.53	20	<0.03	70	7.28	<0.1	25	0.46	0.3	0.4	102
Var #3	LFDD17-243	223.9	233.2	6.31/5.22/10.4	7.31	<10	<0.03	54	6.76	0.1	90	<0.02	0.2	0.2	118
Var #5	LFDD18-400	80.2	87.4	0.8/0.76/0.64	0.73	30	<0.03	82	8.7	<0.1	55	0.38	0.2	0.2	94
Var #6	LFDD18-400	97.9	106.9	1.14/1.6/1.15	1.3	20	0.06	76	8.94	<0.1	35	0.44	0.5	0.2	86
Var #7	LFDD18-404	100.8	107.6	2.18/0.59/0.55	1.11	<10	0.03	96	8.6	<0.1	50	0.24	0.1	<0.2	102
Var #8	LFDD18-404	111.2	115.7	0.75/0.63/0.65	0.68	<10	<0.03	118	13	0.4	25	0.68	0.3	<0.2	122
Var #9	LFDD18-405	164	169.7	2.68/1.35/2.27	2.1	<10	0.03	228	8.38	<0.1	35	0.62	0.1	0.4	98
Var #10	LFDD18-409	115.1	126.8	2.03/5.27/5.52	4.27	<10	<0.03	46	6.66	<0.1	65	0.02	0.2	0.6	108
Var #11	LFDD18-409	126.8	138	2.99/3.78/3.56	3.44	<10	<0.03	42	2.84	0.4	10	0.08	0.2	0.4	96
HG Comp	LFDD18-410 LFDD18-410	173.75 182.1	182.1 193	56.5/22.4/19.6	32.8	<10	<0.03	44	6.58	0.5	100	<0.02	0.2	4.8	114
Var #14	LFDD18-412	342.6	351	0.92/0.61/0.44	0.66	<10	<0.03	48	2.2	<0.1	5	0.18	<0.1	<0.2	44
Var #15	LFRCDD18-359	300.4	304.7	4.29/4.8/2.66	3.92	<10	<0.03	10	2.88	<0.1	20	<0.02	0.1	1.6	64
Var #16	LFRCDD18-361	258.8	270.6	0.75/1.41/10.1	4.09	<10	<0.03	28	5.32	0.1	70	<0.02	<0.1	0.4	122
Var #17	LFRCDD18-361 LFRCDD18-361	270.6 277.7	272.7 282.95	0.89/0.82/1.11	0.94	<10	<0.03	12	9.88	<0.1	145	<0.02	0.1	<0.2	330
Var #18	LFRCDD18-361	305.4	315.4	2/2.71/2.02	2.24	30	<0.03	52	8.28	<0.1	100	<0.02	0.2	1	128
Var #19	LFRCDD18-362	266.55	273.2	1.15/1.27/0.6	1.01	<10	<0.03	40	2.76	<0.1	5	0.16	<0.1	0.2	52
Var #20	LFRCDD19-441	245	252.7	1.74/3.02/2.8	2.52	<10	<0.03	4	1	1	5	<0.02	<0.1	4.8	24
Var #21	LFRCDD19-459	181.4	190	0.97/1.5/1.38	1.28	<10	<0.03	150	9.42	<0.1	40	0.34	0.1	0.4	136
Var #22 [Ox]	LFDD19-670	0	7.3	17.1/23/27.9	22.67	<10	0.09	92	9.34	<0.1	105	<0.02	0.3	0.4	168
Var #23 [Ox]	LFDD19-670	7.3	10.4	0.68/1.21/1.45	1.11	<10	0.03	84	8.88	<0.1	135	<0.02	0.2	0.4	124
Var #24 [Ox]	LFDD19-671	0.9	7.7	1.31/1.39/1.34	1.35	40	0.06	82	8.36	<0.1	50	<0.02	0.5	0.4	118
Var #25 [Ox]	LFDD19-668	5.4	11.3	10.3/3.03/7.92	7.08	<10	0.03	148	2.4	<0.1	15	<0.02	<0.1	0.2	114
Var #26 [Ox]	LFDD19-669	3.4	6.7	1.07/0.74/1.62	1.14	10	0.06	28	9.02	<0.1	45	<0.02	0.2	<0.2	58
Var #28 [Ox]	LFDD19-669	13.7	17.2	0.33/0.48/4.15	1.65	<10	0.03	46	14.9	<0.1	5	<0.02	0.1	<0.2	154
Var #29	LFRCDD19-667	62	66	0.45/0.42/0.77	0.55	<10	<0.03	128	2.7	0.2	50	0.34	<0.1	<0.2	222
Var #32	LFRCDD19-667	90	95	8.9/8.17/6.26	7.78	<10	<0.03	86	2.5	0.3	5	0.32	0.2	1	86
Var #33	LFRCDD19-667	95	99	1.64/1.4/1.95	1.66	10	0.03	20	2.42	<0.1	5	0.32	0.2	0.6	72
Var #34	LFRCDD19-667	99	103.5	1.67/1.48/2.14	1.76	<10	0.03	24	2.78	0.2	5	0.42	0.1	0.8	86
Var #35	LFRCDD19-667	103.5	109.2	1.24/1.28/2.25	1.59	<10	<0.03	78	7.46	0.2	70	0.16	<0.1	<0.2	122

Note: Ag grades are all below the detection limit of 2 ppm except for #22 which recorded 6 ppm.

13.8.2 Variability Composite Gravity Leach Test at Standard Conditions

Gravity/leach tests using the optimised conditions were conducted on the variability composites; grind size of P_{80} 106 µm, gravity concentration using amalgamation, 55% solids (fresh) or 40% solids (oxide), air sparging, initial 0.025% w/v NaCN maintained above 200 ppm NaCN and 24-hour leach. The results of the variability tests are presented in Table 13.8.2 and Figure 13.8.1.

Overall gold extraction from the variability samples was fairly consistent and moderately well aligned with the master composite results. Calculated head gold grades were generally higher than assayed heads due to the high amount of coarse gold. Cyanide and lime consumption were similar to the master composite usage rates. Lime usage was notably higher for the oxide ores than the fresh ores suggesting the presence of clays with some buffering effects.

Average fresh ore residue grades are only slightly higher than the bulk leach result and are significantly lower for the oxide samples. Some samples displayed slower leach kinetics with leaching continuing through to the end of the test (24 hours) but final tails grades were generally acceptable.

The oxide gold ores are generally free milling with high extractions, particularly when there is high gravity recoverable gold.

13.8.3 2019 Testwork - Repeat Variability Composite Testwork

Additional testwork was completed to investigate the slower leach kinetics in some of the variability composites. A series of repeat tests was conducted trialling:

- Pre-oxidation conditioning (using air) with lime.
- Iincreased cyanide addition (0.035% w/v NaCN maintained above 0.03% w/v NaCN).
- Increased leach residence time (36 hours).
- Finer grinding to P_{80} 75 µm.
- Oxygen sparging.

The repeat testwork results are summarised in Table 13.8.3 Results for the standard condition tests are included for comparison.

The higher cyanide addition and increased leach time were sufficient to enhance the kinetics and overall leach gold extraction. Pre-oxidation generally reduced the cyanide consumption but did not always improve extraction. Finer grinding and oxygenation also increased leach kinetics but are considered unnecessary when increased cyanide addition and extra residence time can be more readily implemented.

A full set of variability results, using the repeat tests where applicable, are presented in Table 13.8.4 and are the results used for estimation of the metallurgical recoveries and reagent consumptions.

Table 13.8.2 2019 Testwork Variability Composite Gravity Leach Testwork at Standard Conditions

Test ID	Variability	Test	Grind Size	Initial NaCN	Leach %	Sparge	Avg Head	Calc. Head	Res.	% GRG	Overall % Gold Extraction @ Time hours					Reag. Cons. kg/t	
	Comp No.		P_{80} µm	% w/v	Solids w/w	Gas	Au g/t	Au g/t	Au g/t		2	4	8	12	24	NaCN	Lime
JS4543	1	Std Cond	106	0.025	55	AIR	3.30	3.31	0.14	55	84	91	92	94	95.8	0.06	0.35
JS4544	2A	Std Cond	106	0.025	55	AIR	2.53	1.88	0.27	38	50	56	64	70	85.7	0.09	0.40
JS4545	3	Std Cond	106	0.025	55	AIR	7.31	5.65	0.05	77	86	95	98	98	99.1	0.06	0.30
JS4546	5	Std Cond	106	0.025	55	AIR	0.73	0.73	0.07	18	32	43	61	67	90.4	0.06	0.55
JS4547	6	Std Cond	106	0.025	55	AIR	1.30	1.09	0.09	19	36	45	69	81	91.8	0.09	0.50
JS4548	7	Std Cond	106	0.025	55	AIR	1.11	0.96	0.02	63	70	85	90	95	97.9	0.06	0.35
JS4549	8	Std Cond	106	0.025	55	AIR	0.68	0.83	0.07	40	48	53	72	87	91.6	0.15	0.75
JS4550	9	Std Cond	106	0.025	55	AIR	2.10	3.44	0.08	74	78	83	86	88	97.7	0.09	0.30
JS4551	10	Std Cond	106	0.025	55	AIR	4.27	2.14	0.21	69	83	87	88	89	90.2	0.04	0.30
JS4552	11	Std Cond	106	0.025	55	AIR	3.44	3.78	0.22	48	70	85	91	93	94.2	0.05	0.30
JS4553	14	Std Cond	106	0.025	55	AIR	0.66	0.92	0.03	65	72	76	84	90	96.8	0.06	0.20
JS4554	15	Std Cond	106	0.025	55	AIR	3.92	3.64	0.05	71	85	93	96	97	98.6	0.05	0.20
JS4555	16	Std Cond	106	0.025	55	AIR	4.09	1.33	0.03	73	87	95	96	97	97.7	0.07	0.30
JS4556	17	Std Cond	106	0.025	55	AIR	0.94	0.61	0.01	67	77	91	95	97	98.4	0.06	0.25
JS4557	18	Std Cond	106	0.025	55	AIR	2.24	1.78	0.07	46	60	76	86	91	96.1	0.06	0.25
JS4558	19	Std Cond	106	0.025	55	AIR	1.01	1.97	0.02	85	88	90	93	95	99.0	0.07	0.20
JS4559	20	Std Cond	106	0.025	55	AIR	2.52	6.86	0.11	71	91	95	98	98	98.4	0.08	0.20
JS4560	21	Std Cond	106	0.025	55	AIR	1.28	2.49	0.08	73	81	86	92	94	96.8	0.07	0.30
JS4567	29	Std Cond	106	0.025	55	AIR	0.55	0.89	0.03	45	57	68	90	94	96.6	0.05	0.20
JS4568	32	Std Cond	106	0.025	55	AIR	7.78	7.90	0.18	73	86	92	96	97	97.7	0.05	0.20
JS4569	33	Std Cond	106	0.025	55	AIR	1.66	1.88	0.07	46	73	85	92	95	96.3	0.05	0.20
JS4570	34	Std Cond	106	0.025	55	AIR	1.76	1.92	0.12	28	49	65	87	90	93.8	0.07	0.20
JS4571	35	Std Cond	106	0.025	55	AIR	1.59	3.33	0.02	91	97	98	99	99	99.4	0.04	0.25
JS4561	22 [OXIDE]	Std Cond	106	0.025	40	AIR	22.7	23.1	0.44	70	89	92	95	96	98.1	0.08	2.30
JS4562	23 [OXIDE]	Std Cond	106	0.025	40	AIR	1.11	1.67	0.02	70	95	96	98	98	98.8	0.11	6.70
JS4563	24 [OXIDE]	Std Cond	106	0.025	40	AIR	1.35	1.50	0.06	48	81	88	91	92	96.0	0.14	2.80
JS4564	25 [OXIDE]	Std Cond	106	0.025	40	AIR	7.08	7.53	0.02	87	97	99	99	100	99.7	0.08	0.60
JS4565	26 [OXIDE]	Std Cond	106	0.025	40	AIR	1.14	1.19	0.05	29	79	85	89	92	95.8	0.08	1.80
JS4566	28 [OXIDE]	Std Cond	106	0.025	40	AIR	1.65	0.50	0.02	50	92	93	96	96	96.0	0.08	2.15
					Average Fresh		2.47	2.58	0.09	58.0	58	71	80	88	91	95.6	0.31
					Median Fresh		1.76	1.92	0.07	65.4	77.5	65	77	85	91	94	96.8
					15th Percentile Fresh		0.80	0.90	0.02	38.7	49.2	39	49	59	75	88	91.6
					85th Percentile Fresh		4.04	3.74	0.17	73.7	86.8	74	87	94	96	97	98.6
					Average Oxide		5.83	5.91	0.10	59.0	88.8	59	89	92	95	96	97.4
					Median Oxide		1.50	1.59	0.04	60.0	90.3	60	90	93	95	96	97.1
					15th Percentile Oxide		1.14	1.02	0.02	43.1	80.7	43	81	87	90	92	96.0
					85th Percentile Oxide		11.0	11.4	0.16	74.4	95.5	74	96	97	98	98	99.0
JJ4501	Fresh MC	Bulk Leach	106	0.025	55	AIR	2.05	3.00	0.06	78	88	93	96	97	98.0	0.08	0.23
JS4513	Oxide MC	Bulk Leach	106	0.025	40	AIR	3.06	4.78	0.13	72	92	94	95	96	97.3	0.25	2.46

Figure 13.8.1 2019 Testwork Variability Gravity/Leach Testwork at Standard Conditions

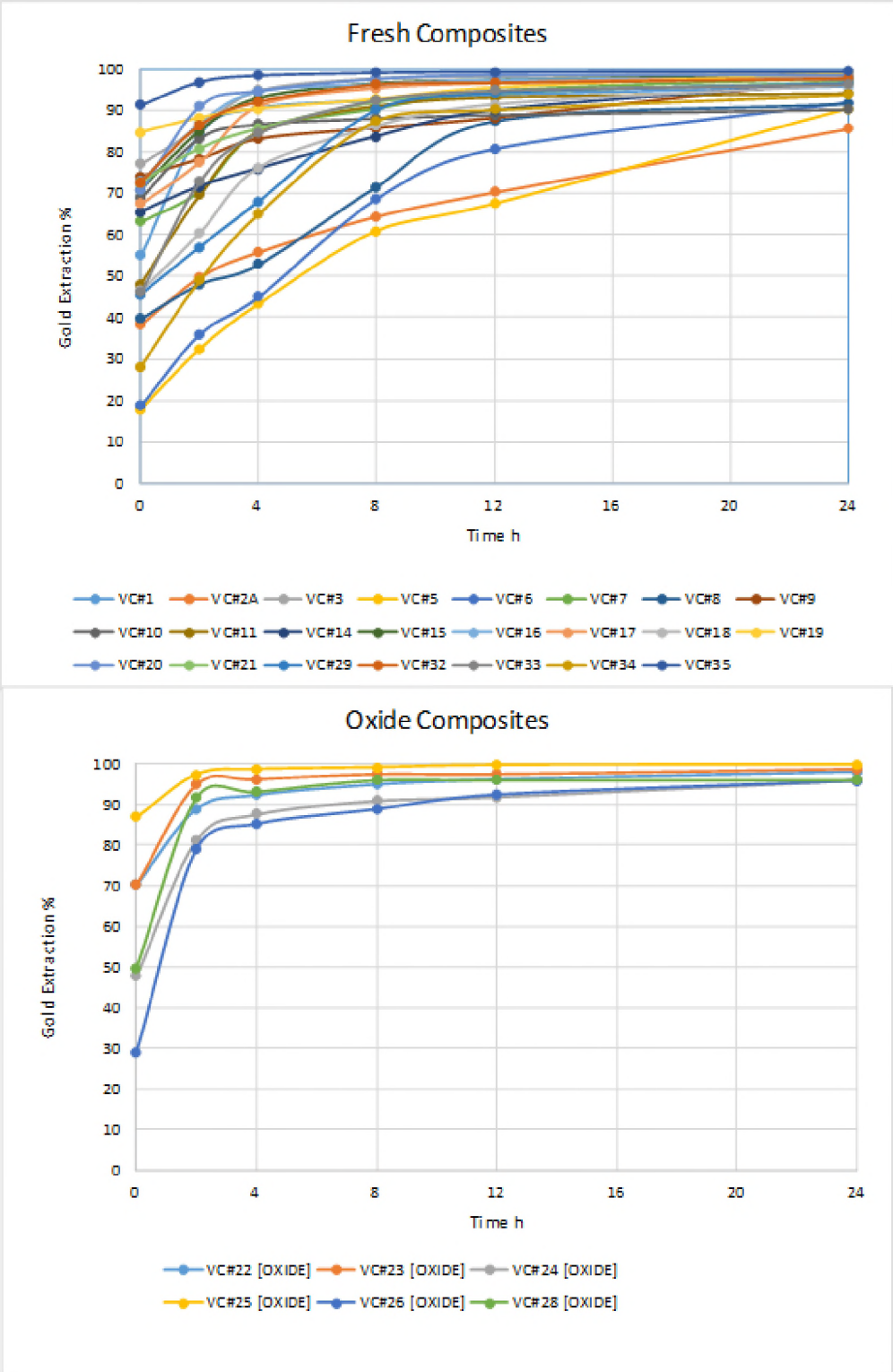


Table 13.8.3 2019 Testwork Variability Gravity / Leach Repeat Testwork Results

Test	Variability	Test	Grind	NaCN	Solids	Sparge	Avg Head	Calc. Head	Res.	GRG	Overall % Gold Extraction @ Time hours						Reag. Cons. (kg/t)		% Au Ext'n Incr.	
ID	Comp No.	Condition	P_{80} µm	% w/v	% w/w	Gas	Au g/t	Au g/t	Au g/t	%	2	4	8	12	24	36	NaCN	Lime	24 h	Over'l
JS4544	2A	Standard	106	0.025	55	AIR	2.53	1.88	0.27	38	50	56	64	70	85.7		0.09	0.40		
JS4666	2A	4hr Pre-Ox, Incr CN	106	0.071	55	AIR	2.53	2.06	0.09	47	69	84	93	94	94.6	95.6	0.24	0.62	9.0	10.0
JS4667	2A	Incr CN	106	0.035	55	AIR	2.53	2.21	0.10	43	67	83	92	93	95.0	95.5	0.17	0.52	9.3	9.8
JS4669	2A	4hr Pre-Ox, Incr CN	106	0.035	55	AIR	2.53	1.95	0.09	44	64	84	92	94	94.6	95.4	0.05	0.64	9.0	9.7
JS4546	5	Standard	106	0.025	55	AIR	0.73	0.73	0.07	18	32	43	61	67	90.4		0.06	0.55		
JS4581	5	4hr Pre-Ox	106	0.025	55	AIR	0.73	0.71	0.04	25	36	48	82	89	92.5	94.3	0.09	0.60	2.1	3.9
JS4547	6	Standard	106	0.025	55	AIR	1.30	1.09	0.09	19	36	45	69	81	91.8		0.09	0.50		
JS4661	6	4hr Pre-Ox	106	0.035	55	AIR	1.30	1.19	0.09	25	44	66	89	90	91.8	92.4	0.10	0.71	0.1	0.6
JS4662	6	Incr CN	106	0.035	55	AIR	1.30	1.10	0.07	31	55	77	90	91	93.0	93.6	0.15	0.66	1.2	1.8
JS4550	9	Standard	106	0.025	55	AIR	2.10	3.44	0.08	74	78	83	86	88	97.7		0.09	0.30		
JS4582	9	4hr Pre-Ox	106	0.025	55	AIR	2.10	2.96	0.05	65	67	69	72	76	87.9	98.3	0.10	0.35	-9.8	0.6
JS4584	9	+Oxygen	106	0.025	55	OXY	2.10	1.72	0.05	72	82	87	93	96	96.9	97.1	0.11	0.40	-0.8	-0.6
JS4585	9	Incr CN	106	0.035	55	AIR	2.10	3.00	0.01	84	85	86	89	94	99.0	99.7	0.08	0.54	1.4	2.0
JS4551	10	Standard	106	0.025	55	AIR	4.27	2.14	0.21	69	83	87	88	89	90.2		0.04	0.30		
JS4583	10	4hr Pre-Ox	106	0.025	55	AIR	4.27	1.55	0.06	62	76	86	93	95	95.9	96.1	0.07	0.40	5.7	5.9
JS4586	10	Finer Grind	75	0.025	55	AIR	4.27	2.23	0.03	77	87	93	97	98	98.7	98.7	0.07	0.40	8.5	8.5
JS4670	10	Incr CN	106	0.035	55	AIR	4.27	2.16	0.08	70	84	92	94	95	95.5	96.3	0.06	0.26	5.3	6.1
JS4552	11	Standard	106	0.025	55	AIR	3.44	3.78	0.22	48	70	85	91	93	94.2		0.05	0.30		
JS4663	11	Incr CN	106	0.035	55	AIR	3.44	4.34	0.15	56	73	88	95	95	96.1	96.5	0.08	0.30	1.9	2.4
JS4570	34	Standard	106	0.025	55	AIR	1.76	1.92	0.12	28	49	65	87	90	93.8		0.07	0.20		
JS4664	34	Incr CN	106	0.035	55	AIR	1.76	2.01	0.07	30	53	78	95	95	96.2	96.5	0.10	0.26	2.4	2.8
JS4561	22 [OXIDE]	Standard	106	0.025	40	AIR	22.7	23.1	0.44	70	89	92	95	96	98.1		0.08	2.30		
JS4587	22 [OXIDE]	Incr CN, Finer Grind	75	0.035	40	AIR	22.7	15.9	0.07	75	96	98	99	99	99.3	99.6	0.17	3.10	1.2	1.5

Table 13.8.4 2019 Testwork Variability Gravity / Leach Results with Select Repeat Tests

Test	Sample	Calc. Head	Residue	% GRG	Overall % Gold Extraction @ Time hours						Reagent Cons. kg/t	
No.	ID	g/t Au	g/t Au		2	4	8	12	24	36	NaCN	Lime
JS4543	#1	3.31	0.14	54.9	84	91	92	94	95.8		0.06	0.35
JS4667	#2A Rpt	2.21	0.10	43.4	67	83	92	93	95.0	95.5	0.17	0.52
JS4545	#3	5.65	0.05	77.0	86	95	98	98	99.1		0.06	0.30
JS4581	#5 Rpt	0.71	0.04	24.8	36	48	82	89	92.5	94.3	0.09	0.15
JS4662	#6 Rpt	1.10	0.07	31.5	55	77	90	91	93.0	93.6	0.15	0.66
JS4548	#7	0.96	0.02	63.2	70	85	90	95	97.9		0.06	0.35
JS4549	#8	0.83	0.07	39.7	48	53	72	87	91.6		0.15	0.75
JS4585	#9 Rpt	3.00	0.01	83.6	85	86	89	94	99.0	99.7	0.08	0.00
JS4670	#10 Rpt	2.16	0.08	69.6	84	92	94	95	95.5	96.3	0.06	0.26
JS4663	#11 Rpt	4.34	0.15	55.9	73	88	95	95	96.1	96.5	0.08	0.30
JS4502	HG Comp.	15.7	0.05	91.5	98	99	99	100	99.7		0.05	0.20
JS4553	#14	0.92	0.03	65.4	72	76	84	90	96.8		0.06	0.20
JS4554	#15	3.64	0.05	70.8	85	93	96	97	98.6		0.05	0.20
JS4555	#16	1.33	0.03	72.9	87	95	96	97	97.7		0.07	0.30
JS4556	#17	0.61	0.01	67.4	77	91	95	97	98.4		0.06	0.25
JS4557	#18	1.78	0.07	46.4	60	76	86	91	96.1		0.06	0.25
JS4558	#19	1.97	0.02	84.7	88	90	93	95	99.0		0.07	0.20
JS4559	#20	6.86	0.11	70.7	91	95	98	98	98.4		0.08	0.20
JS4560	#21	2.49	0.08	72.6	81	86	92	94	96.8		0.07	0.30
JS4567	#29	0.89	0.03	45.5	57	68	90	94	96.6		0.05	0.20
JS4568	#32	7.90	0.18	72.6	86	92	96	97	97.7		0.05	0.20
JS4569	#33	1.88	0.07	46.2	73	85	92	95	96.3		0.05	0.20
JS4664	#34 Rpt	2.01	0.07	30.2	53	78	95	95	96.2	96.5	0.10	0.26
JS4571	#35	3.33	0.02	91.2	97	98	99	99	99.4		0.04	0.25
Fresh Median		**2.08**	**0.06**	**66.4**	**79**	**87**	**93**	**95**	**96.8**		**0.06**	**0.25**
Fresh Average		**3.15**	**0.06**	**61.3**	**75**	**84**	**92**	**95**	**96.8**		**0.07**	**0.29**
15th Percentile		0.91	0.02	41.4	56	76	88	91	95.2		0.05	0.20
85th Percentile		5.06	0.11	80.6	88	95	97	98	99.0		0.10	0.35
JS4587	#22 [Ox] Rpt	15.9	0.07	75.2	96	98	99	99	99.3	99.6	0.17	3.10
JS4562	#23 [Ox]	1.67	0.02	70.3	95	96	98	98	98.8		0.11	6.70
JS4563	#24 [Ox]	1.50	0.06	47.9	81	88	91	92	96.0		0.14	2.80
JS4564	#25 [Ox]	7.53	0.02	86.9	97	99	99	100	99.7		0.08	0.60
JS4565	#26 [Ox]	1.19	0.05	29.0	79	85	89	92	95.8		0.08	1.80
JS4566	#28 [Ox]	0.50	0.02	49.8	92	93	96	96	96.0		0.08	2.15
Oxide Median		**1.59**	**0.04**	**60.1**	**93**	**95**	**97**	**97**	**97.4**		**0.10**	**2.48**
Oxide Average		**4.72**	**0.04**	**59.8**	**90**	**93**	**95**	**96**	**97.6**		**0.11**	**2.86**
15th Percentile		1.02	0.02	43.1	81	87	90	92	96.0		0.08	1.50
85th Percentile		9.62	0.06	78.1	97	98	99	99	99.4		0.15	4.00
Average All Var		**3.46**	**0.06**	**61.0**	**83**	**89**	**93**	**95**	**96.8**		**0.07**	**0.30**
Median All Var		**1.99**	**0.05**	**66.4**	**78**	**86**	**93**	**95**	**97.0**		**0.08**	**0.83**
15th Percentile		0.90	0.02	41.0	58	77	89	92	95.6		0.05	0.20
85th Percentile		6.44	0.09	81.3	94	96	98	98	99.1		0.13	1.43

13.9 2019 Programme - Determination of Engineering Design Parameters

The physical characteristics of the slurry and engineering design parameters were determined in this testwork phase. This testwork includes oxygen uptake tests, standard carbon loading (kinetic) tests, cyanide detoxification tests, slurry rheology and dynamic thickening testwork.

13.9.1 Oxygen Uptake Rate Tests

Oxygen uptake rate tests were performed with air sparging on the fresh and oxide master composites under the standard leach conditions of a grind size of P_{80} 106 µm, pulp densities of 55% for the fresh composite and 40% for the oxide composite, pH 10.5 and a starting cyanide concentration of 0.025%. The results are shown in Table 13.9.1.

Table 13.9.1 2019 Testwork Oxygen Uptake Rate Results Summary

Composite	Oxygen Uptake Rate (mg/L.min) at Elapsed Time (hours)							
	0	1	2	3	4	5	6	24
Oxide Composite	0.0018	0.0011	0.0012	0.0025	0.0020	0.0025	0.0006	0.0014
Fresh Composite	0.0055	0.0043	0.0054	0.0064	0.0065	0.0051	0.0062	0.0030

The results indicate that the oxygen demand is particularly low for both composites. The low oxygen uptake rate is consistent with the high DO levels throughout the testing to date and the lack of improvement resulting from oxygen sparging in the leach. The ores are clearly very clean with few side reactions that might demand oxygen.

13.9.2 Rheology Tests

Rheology testwork was completed on the oxide and fresh master composites and a blended 40% oxide/60% fresh composite at a grind size of P_{80} 106 µm with the pH adjusted to 10.5 using lime to be representative of conditions in the leach tanks. Testing was conducted to investigate the slurry flow properties at precent solids relevant to the ore types. A summary of the viscosity testing results is presented in Table 13.9.2 and Figure 13.9.1. The figure also shows the accepted viscosity design limits for equipment types.

The viscosities measured indicate that these slurries should be readily managed through the proposed treatment plant with low to moderate slurry viscosities across the range of densities tested.

The 2018 testwork on an oxide composite indicated exceptionally high viscosities (above the range of the viscometer) and the slurry density had to be reduced to 32.5% solids before readings could be achieved. The high viscosity oxide composite results are summarised in Table 13.9.2 and Figure 13.9.2 for comparison. Even after dilution to 32.5% w/w solids, this composite exceeds the viscosity limits for CIL and centrifugal pumping. This sample may have been an exception but blending with lower viscosity oxide ore or fresh ore at all times is recommended to improve material handling and thickener settling rates and ensure that the agitators and pumps can perform in line with their design intent.

Figure 13.9.1 2019 Testwork Slurry Viscosity vs Shear Rate

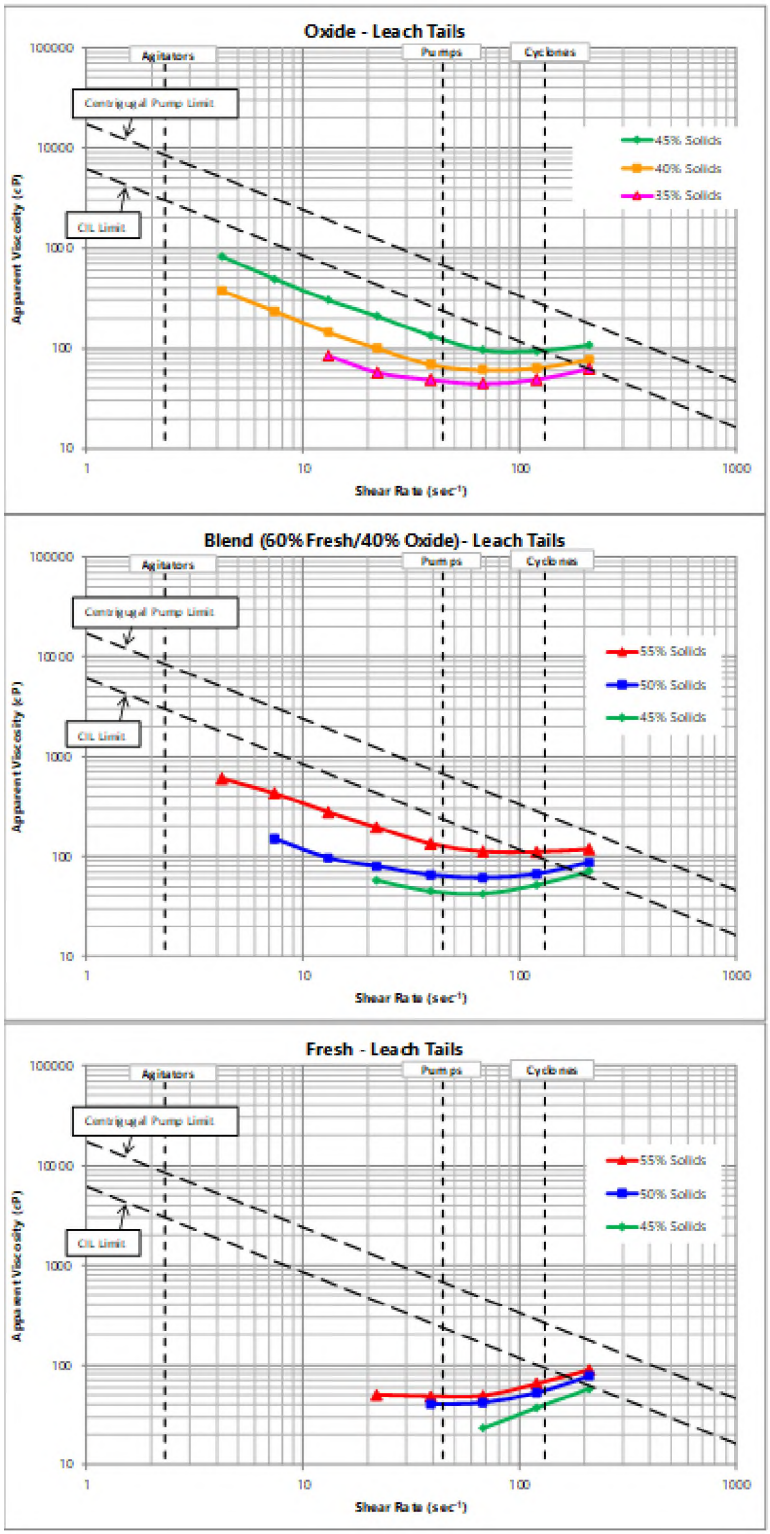


Figure 13.9.2 2018 Testwork High Viscosity Oxide Composite Results

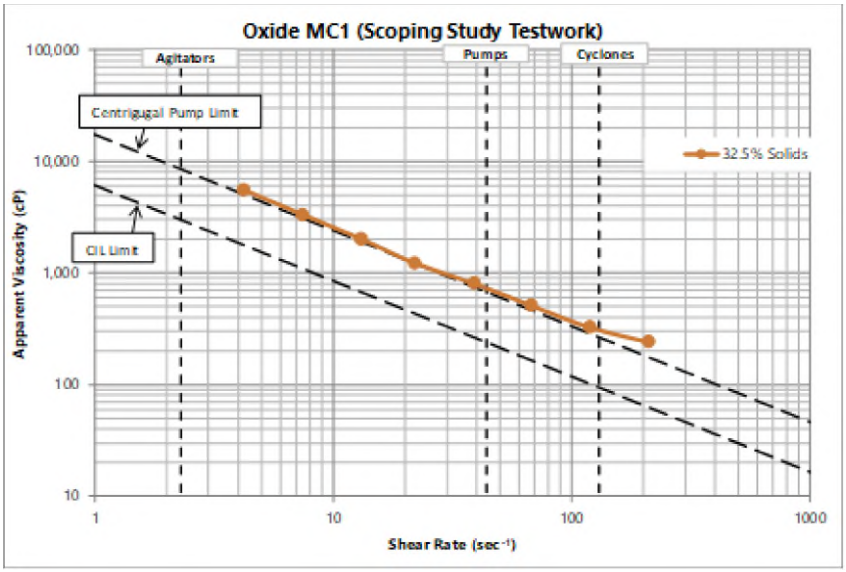


Table 13.9.2 2018 and 2019 Testwork – Rheology Results Summary

	Sample & Oxidation	% Solids w/w	Apparent Viscosity (cP) @ Shear Rate							
			4.2	7.4	13.1	21.9	38.9	67.4	119.2	209.5
2019 PFS Testwork	Oxide	35			84	57	48	44	49	62
		40	374	234	144	100	69	61	63	77
		45	823	489	300	208	133	96	94	107
	Fresh	45						23	37	58
		50					40	42	53	77
		55				50	48	49	65	89
	Blend (60%Fr: 40%Ox)	45				57	44	42	51	
		50		149	96	79	65	61	66	
		55	599	425	276	194	133	112	111	
2018 SS Testwork	Oxide	32.5	5,539	3,335	2,016	1,229	808	513	327	242
		40	Oxide sample was too viscous to test at higher % solids.							
		45								
	Fresh (High Grade)	40							24	47
		50						14	34	60
		60					48	58	76	108
	Fresh (Low Grade)	40							21	44
		50						16	30	32
		60				79	82	85	88	113

13.9.3 Carbon Adsorption Tests

Slurry from the fresh composite bulk leach test under optimised conditions following gravity separation was used for the adsorption testing. The results of the Fleming k and n constants arising from the triple contact sequential carbon loading kinetic tests are summarised in Table 13.9.3.

Table 13.9.3 2019 Testwork Summary of Fleming Constants for Carbon Adsorption

Fleming Rate Constant, k (h-1)	Fleming Equilibrium Constant, n	Cumulative Carbon Loading (g/t C)
184	0.687	917

The carbon loading kinetics are within the range typically observed in CIL/CIP operations. The residual gold in the fresh gravity tailings is relatively low, hence the low overall carbon loading. Carbon loading and adsorption kinetics would likely increase should higher gold solution grades be experienced with lower gravity recoveries or higher leach feed head grades.

13.9.4 Cyanide Detoxification Tests

The SO_2/Air oxidation process was adopted for the cyanide detoxification testwork. Standard continuous cyanide detoxification tests were performed on the fresh and blended oxide/fresh composites using slurry from the bulk leach tests. An ICP scan of the bulk leach tails solution was conducted to determine the cyanide assays and base metals in solution. As can be seen from the data in Table 13.9..4, the solution has very few elements with significant solution concentrations.

Table 13.9.4 2019 Testwork Bulk Cyanidation Tails Solution Assay

Element	JS4501 Fresh Ore Bulk Leach Tails	Fresh Ore Detox Tails Solution	JS4501 + JS4513 40% Oxide Blend Bulk Leach Tails	40% Oxide Blend Detox Tails Solution
CNfree	78.5	n/a	67.6	n/a
CNwad	83.6	0.49	73.7	2.90
pH	9.70	8.54	9.43	8.51
Ag	0.18	<0.02	0.30	<0.02
Al	0.40	0.20	<0.20	0.20
Cu	3.98	0.06	4.28	0.04
Fe	9.40	0.80	3.30	2.50
K	20.0	41.0	5.00	18.0
Mg	1.20	14.8	0.80	10.4
Mo	0.05	0.10	0.05	0.10
Ni	0.25	<0.05	0.55	0.40
Pb	<0.05	<0.05	<0.05	<0.05
Sr	0.04	0.30	0.06	0.46
Zn	0.14	<0.02	0.34	<0.02

Note: Data is in mg/L

The CNWAD concentration was determined directly by a colorimetric method using picric acid reagent. A residence time of nominally 60 minutes was selected. A summary of the cyanide detoxification testwork is provided in Table 13.9.5.

Table 13.9.5 2019 Testwork – Summary of Cyanide Detoxification Testwork

Sample	Detox Stream	Slurry % Solids	Slurry pH	Solution Analysis (mg/L)						
				Titrated NaCN	Calc CNT	CNwad	Cu	Fe	Ni	Zn
JS4501 (Fresh)	Feed	55	9.70	150	110	83.6	3.98	9.4	0.25	0.14
JS4501/4513 (Oxide/Fresh Blend)	Feed	48	9.43	130	83	73.7	4.28	3.3	0.55	0.34
Fresh	Disch.		8.56		2.4	0.5	0.07	0.70	0.25	<0.02
Oxide/Fresh Blend	Disch.		8.54		9.8	2.5	0.05	2.60	0.38	<0.02

6:1 SO2:CNWAD ratio

Tails solution assays are averages of last two samples

Reagent Consumptions: 0.66 kg/t Na2S2O5, 0.14 kg/t CuSO4.5HO2, 1.02 kg/t Lime (60% CaO)

The cyanide detoxification testwork indicated that the CN_{WAD} concentrations in the cyanidation tailings slurry comprised mainly CN_{free} (>92%) with only trace base metal concentrations in solution. The discharge liquor following cyanide detoxification for each of the composites was reduced to <3 ppm CN_{WAD} with a moderate excess stoichiometric addition of SMBS (equiv. 6:1 SO_2:CN_{WAD}).

13.9.5 Thickening Testwork

Thickening testwork was completed by Outotec on a fresh and oxide/fresh blend slurry composite. Settling rates for the fresh ores were moderately fast with high underflow densities and clear overflow at low flocculant dosages (10 g/t). The oxide blend settled more slowly requiring higher flocculant dosages (30 g/t) at the same flux rates. Lower underflow densities were achieved, suggesting that the maximum operating oxide blend should be reduced, despite the relatively low viscosities recorded. The poorer oxide thickening performance, appears to be mainly related to the natural fineness of the material. The yield stresses were not measured for these test products.

Table 13.9.6 2019 Testwork - Thickening Test Results for Oxide Blend and Fresh Ore

Ore Type	Settling/Flux Rate		Product U/F		Flocculant (g/t)	
	$t/m^2/h$	Feed Solids % w/w	% w/w	Yield Stress (Pa)	Testwork	Design
Oxide / Sulphide Blend	0.8	5	58.9	n/a	30	45
Fresh Composite	0.8	5	64.6	n/a	10	15

The design thickening settling rates will be 0.8 t/m^2 being the limiting rate for the oxide blend, which is size determining. Higher rates were not tested for the fresh ore, although it is likely that these would be sustainable at higher flocculant dosages than those tested. If it is planned to feed the plant with higher oxide blends, the facility to bypass the thickeners on high oxide feed blends should be in place.

13.10 Tailings Sample Preparation

13.10.1 Tailings and Geochemical Testwork

Bulk tailings samples were prepared for tailings settling and geochemical testwork carried out by the consultants, Knight Piésold, who were also responsible for the design of the tailings storage facility (TSF).

13.10.2 Acid Rock Drainage

An evaluation of acid rock drainage (ARD) for the Lafigué ore will be completed at the next study phase. It is expected that ARD issues will be minimal based on the current knowledge of the mineralogy of the Lafigué Project:

- The minimal amounts of sulphide material that have been observed are in the process plant feed ore types which will be processed and stored in the lined TSF. If the low amounts of contained sulphides generate ARD, it is expected to be minimal and fully contained in the TSF.

- Minimal sulphide minerals are expected to be in the waste rock and hence there should not be any issue with ARD at the waste dump locations.

13.11 Metallurgical Recoveries and Reagent Consumption

The 2019 variability composite testwork results at a grind size of P_{80} 106 µm were used to estimate the expected plant gold recoveries over a range of plant feed head grades and the expected plant reagent consumptions. The testwork results used in estimating the gold recovery and leach reagent consumptions are as presented in Table 13.8.4.

13.11.1 Gold Recovery

Although there was a spread of gold extractions in the variability testwork dataset, the majority of the results are clustered around the mean with 24 hour extractions within one standard deviation of the mean ranging from 95.6 to 99.1% for fresh ore (96.0 % to 99.6% for the oxide samples).

It is noted that all gold extractions were within a narrow band over the wide head grade range of 0.50 g/t Au to 15.9 g/t Au (median 1.99 g/t Au, average 3.46 g/t Au), as shown in the top graph in Figure 13.11.1. It is only when the gold extraction axis scale is expanded (showing 95% to 100% gold extraction), the variation in results is more noticeable.

The 24-hour median recovery from the variability data sets were selected for economic modelling of the project since the gold extraction results are closely grouped with few low recovery outliers. An allowance for a soluble loss of 0.02 g/t Au equivalent in the tails solution was added to the residue grade. This amounts to 1.0% to 1.3% gold recovery loss at the median head grade, as detailed in Table 13.11.1.

Table 13.11.1 Soluble Loss and Overall Recovery

Item	Unit	Fresh	Oxide
Median Calc Head Grade	Au g/t	2.08	1.59
Median 24 h Extraction	% Au	96.8%	97.4%
Soluble Au Loss Equiv.	Au g/t	0.02	0.02
Soluble Au Loss Equiv.	Au g/t	1.0%	1.3%
Overall Gold Recovery	**% Au**	**95.8%**	**96.1%**

On the basis of treating the Fetekro fresh and oxide rock types via the proposed gravity and direct cyanidation treatment route, an overall gold recovery of 95.8% is recommended for economic evaluation based on the median of the variability testwork results after allowing for likely soluble gold loss.

There is a weak correlation between head grade and gold extraction which can be represented by a standard rate equation. The following head grade / recovery relationship (based on the standard rate equation) can be applied to the resource model:

Overall Gold Recovery = 99.8%*(1-1/(25*Au Head g/t)) - 1%

The recovery model results are also shown in in Figure 13.11.1. When the recovery model is used to estimate the gold recovery based on the median calculated head grade of the fresh ore (2.08 g/t Au), the gold recovery is 96.9 % Au. This is higher than the 95.8% Au median recovery and suggests that the proposed median recovery value may be conservative.

13.11.2 Reagent Consumption

The variability testwork dataset provides a good basis for determining the reagent consumption rates to be used for engineering design and the operating cost estimate inputs. For the Lafigué ores it is appropriate to consider average reagent usage rates for the operating cost estimates since the usage rates across the testwork have been very consistent for each oxidation state.

Cyanide leaching reagent consumption was consistently low. Cyanide consumption for the fresh ore will be 0.13 kg/t while the oxide requires 0.16 kg/t based on the median from the variability tests. Cyanide usage includes an allowance for a free cyanide residual (100 mg/L NaCN) in the thickened CIL tails assuming no cyanide recovery in the tailings wash thickener or decant return water.

Lime demand to maintain a pH of 10.5 will typically be low for the fresh ores, but moderate for the oxides. Consumption is based on the median usage in the variability tests and the available active lime contents in the laboratory hydrated lime vs the industrial quicklime. The fresh ore industrial quicklime (90% CaO) consumption will be 0.17 kg/t ore while the oxide requires 1.65 kg/t.

Figure 13.11.1 Gold Extraction vs Head Grade

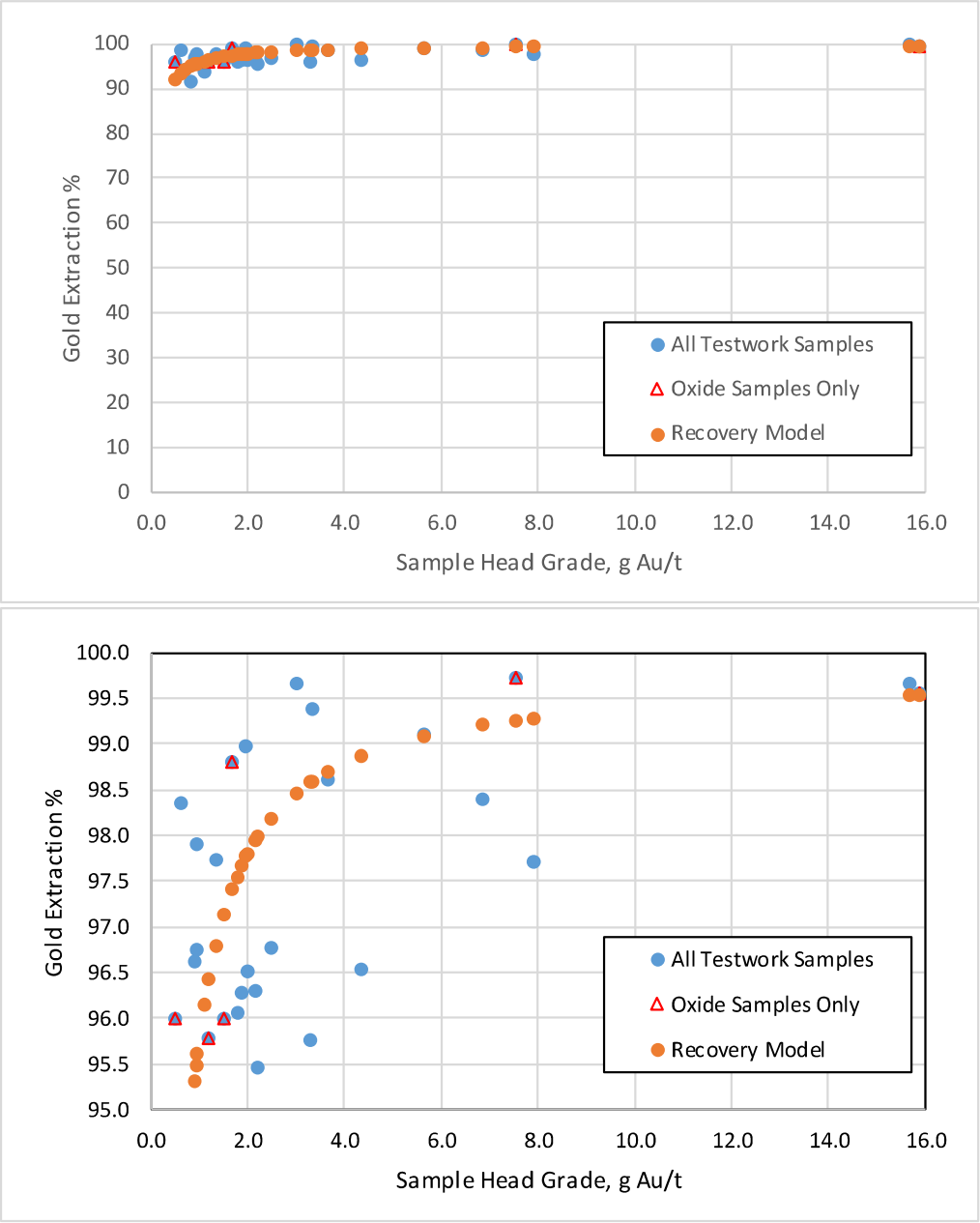


Table 13.11.2 Estimated Plant Leach Reagent Consumption

Ore Feed	Plant Consumption	
	NaCN kg/t*	Lime kg/t^
Oxide	0.16	1.65
Fresh	0.13	0.17

*Including 100 ppm NaCN solution loss to tails

^Based on 90% CaO

13.12 Metallurgical Testwork Conclusions

Core samples from the various ore lithologies, weathered states and mineralisation styles were selected to extend over appropriate minable widths, include likely dilution and be representative of expected life of mine (LOM) average grades.

Comminution samples aimed mainly to cover the typical lithological distribution and alteration around the contact zones where the gold mineralisation typically occurs. Examples of individual host lithologies and adjoining country rock types (likely dilution) were also sampled for comminution testing.

Sample head assays indicated that there are few deleterious elements for gold leaching with low levels of base metals and arsenic. The high gravity gold content resulted in a reasonable degree of gold assay variability. Silver grades were generally very low with a few exceptions where an isolated higher assay occurred.

The comminution testwork programme produced characteristic data for the fresh ore lithological examples selected. The fresh ore is very competent with a high breakage energy requirement for the coarse particles. Oxide ore was not tested as the material is too fine for comminution testwork.

A mineralogical investigation indicated that much of the gold occurred as free grains, but there is some locking of fine gold in pyrite and with silver tellurides. Pyrrhotite association with the gold mineralisation did not indicate any significant impacts on reagent consumption or gold leach extractions.

Cyanidation tests were conducted on the fresh master composite following gravity gold recovery at different grind sizes to evaluate the effect of grind size on gold extraction. Leaching was rapid for all grinds following high gravity gold recoveries with the bulk of the gold dissolution occurring within four to eight hours. The ore appeared to be relatively insensitive to grind with only a 1% difference in gold extraction over the size range tested. A grind size of P_{80} 106 µm was selected for design and further testing following an economic evaluation of optimum grind size. This grind had similar gold extraction and lower operating costs compared to the finer grind sizes tested.

Leach optimisation testing on the master composite samples indicated that high gold extractions were achieved with air only sparging (no high purity oxygen required), relatively low cyanide dosing and high fresh ore slurry densities (up to 55% solids w/w). These conditions were used for bulk leaching to produce larger slurry volumes for physical characterisation of slurry rheology, carbon adsorption, cyanide detoxification and dewatering testwork.

Conflicting slurry rheology results for the oxide ores were measured indicating that blending within oxide ores or with fresh ores may be beneficial in managing the materials handling characteristics. Fresh ores have low viscosities and good settling rates.

Variability leach tests were conducted on the 25 fresh samples and six oxide samples making up the master composites. Gold extractions from most samples were very high (>95%) with a few exceptions (five samples) that had slightly lower extractions. Increased cyanide addition (maintaining a higher free cyanide excess concentration since consumption remained low) and extra leach residence time (36 hours) improved extractions to expected levels. These conditions were accepted as improving the flowsheet robustness and the base design conditions were modified to suit, resulting in conservative operating cost inputs.

On the basis of treating Lafigué fresh and oxide ores via gravity and direct cyanidation, an overall gold recovery of 95.8% is recommended. This recovery is based on the median of the variability testwork results after allowing for the likely soluble gold loss equivalent of 1% Au.

Cyanide consumption for the fresh ore will be 0.13 kg/t while the oxide requires 0.16 kg/t. These figures include a free cyanide excess loss of 100 ppm NaCN.

Lime demand to maintain pH will typically be low for the fresh ores, but moderate for the oxides. The fresh ore plant supply quicklime consumption will be 0.17 kg/t ore while the oxide requires 1.65 kg/t.

14.0 MINERAL RESOURCES ESTIMATE

14.1 Introduction

The following section includes discussion and comment on all aspects relating to the Mineral Resource statement reported as of July 31, 2020 for the Lafigué deposit which lies within the Fetekro exploration permit. The specific areas covered comprise geological modelling, density, statistical analysis, variography, block modelling, grade estimation, Mineral Resource classification, reasonable prospects for economic extraction and the 2020 Mineral Resource Statement.

14.2 Basis of Supporting Data

The Mineral Resource Estimate (MRE) for the Lafigué Resource is reported as of July 31, 2020 and utilises a Lidar topographic surface from 2015. The Lafigué deposit was modelled in GEOVIA Surpac™ integrated geology and resource modelling software. The cut-off date for the drillhole database for the modelling is July 22, 2020.

The cut-off date for the drillhole database for the MRE is July 22, 2020 and therefore differs to that presented in Section 10 Drilling for the Project as of December 31, 2020. The RAB holes are not used in the MRE.

The drillhole database for the resource area was validated to ensure there were no unexplained gaps in the data, no overlapping intervals, unusual collar survey data or downhole survey irregularities. Any drill holes not able to be validated were excluded from the resource estimation.

The database used during the resource interpolation includes 93 core DD holes and 580 RC and 156 RC-DD drill holes, see Table 14.2.1.

Table 14.2.1 Summary of Drill Holes Used in the MRE

Deposit	Gatroci (1997)			Cominor (2002,2010)			La Mancha (2014)			Endeavour(2017-2020)			
	Meters	DDH	RC	Meters	DDH	RC	Meters	DDH	RC	Meters	DDH	RC	RC-DD
Lafique South	1,593	765	828	0	0	0	1,786	705	1081	1,310	343	967	0
Lafique Center	905	184	721	1,717	608	1109	3,213	751	2462	19,673	918	15198	3,557
Lafique North	497	497	0	1,075	250	825	1,503	408	1095	105,423	5337	54364	45,722
Total	**2,995**	**1446**	**1,549**	**2,792**	**858**	**1,934**	**6,502**	**1864**	**4,638**	**126,406**	**6598**	**70,529**	**49,279**

14.3 Lafigué Deposit

14.3.1 Lithology

The Lafigué deposit is located in a Birimian volcanic complex intruded by granodioritic bodies and quartz-porphyry dykes. The mineralisation is mainly controlled by an ENE-trending brittle-ductile thrust fault dipping 15° to 45° SSE. The shear bands are localised preferably at the edges of a granodiorite intrusive or at a basalt/gabbro interface and crosscuts the main regional foliation. The mineralisation has been recognised over two kilometres along an ENE axis and the down dip extension has been demonstrated over one kilometre to date.

Endeavour geological staff have interpreted the geology based on geological logging of the drill holes and fieldwork. Their interpretations have provided the base for the lithological and mineralisation models. The geology was interpreted in Leapfrog with 3D wireframe solids created (Table 14.3.2). The interpretation of the geology and mineralisation is based primarily on 40 m to 50 m spaced sections.

Table 14.3.1 Lithological Wireframes

Lithology	3DM
Breccia	LF_GEOLOGICAL_MODEL-Laterite.dxf
Mafic Intrusion	LF_GEOLOGICAL_MODEL-Mafic intrusive.dxf
Quartz Vein	LF_GEOLOGICAL_MODEL-Quartz vein.dxf
Schist	LF_GEOLOGICAL_MODEL-Saprolite.dxf
Mafic Volcanics	LF_GEOLOGICAL_MODEL-Mafic volcanic.dxf
Felsic Intrusion	LF_GEOLOGICAL_MODEL-Felsic intrusive.dxf

14.3.2 Weathering

For the July 2020 model, the weathering domain surfaces for the base of the laterite, the base of the saprolite and the top of the bedrock were created based upon cross-section interpretations using the geology and weathering logging data from the drillholes. DTM surfaces were created to represent the main weathering domains including overburden / laterite, saprolite, transition and bedrock weathering domains (Table 14.3.2).

Table 14.3.2 Weathering Interpretation Surfaces and Block Model Code

Weathering Type	Weathering Code	DTM Files
Air	AIR	topo_lidar_contours.dtm
Laterite	LATR	latr_base_july2020.dtm
Saprolite	SAPR	sapr_base.dtm
Transition	SPRK	Above BDRK/Below SAPR surfaces
Fresh	BDRK	bdrk_top.dtm

14.3.3 Mineralisation

Mineralisation is mainly hosted by a network of quartz veins showing a Tm-Chl-Po-Py-Au alteration assemblage.

The lodes take are located preferentially on lithological or structural discontinuities, typically at the granodiorite edges, on C planes or re-opening early Qz-Cb veins. The succession of hydrothermal events associated with C plane fracture phases and thrusting resulted in the formation of veins in extension, and thus a progressive enrichment of gold in the Lafigué deposit.

At the deposit scale, the lodes show pinch and swell figures both laterally and longitudinally with thicknesses of up to 40 m.

Mineralisation was interpreted on 40 m to 50 m spaced sections, based on the drillhole spacing. Twenty-two mineralised zones and four laterite domains were interpreted.

The gold mineralised domains are based on the interpreted geology and mineralisation using approximately a 0.3 g/t Au threshold.

Typically, a minimum width of two metres for the mineralised zone was used and may include lower grade material to provide continuity.

For each section, the interpreted polylines delineating the mineralisation were snapped to the drill hole sample positions. A minimum width of approximately two metres was used. For each section, the interpretation was not typically extended more than 25 m along strike or down-dip past the last drill hole; however, sectional interpretations were extended if needed to provide better continuity.

The final July 2020 interpretation included twenty-two mineralized zones / domains and four mineralised laterite zones / domains, which collectively make up the Lafigué deposit. The Lafigué deposit is separated into four main areas:

- Lafigué South - Domains 1-2;
- Lafigué Centre - Domains 3-7;
- Lafigué North (Domains 8-22); and
- Mineralised laterite (Domains 120, 130, 140, 150).

Figure 14.3.1 illustrates the different domains, whilst Figure 14.3.2 illustrates cross-sections A-B-C-D-E showing the mineralised domains and drillholes.

Figure 14.3.1 Lafigué Mineralisation Domains with Drilling – Plan View

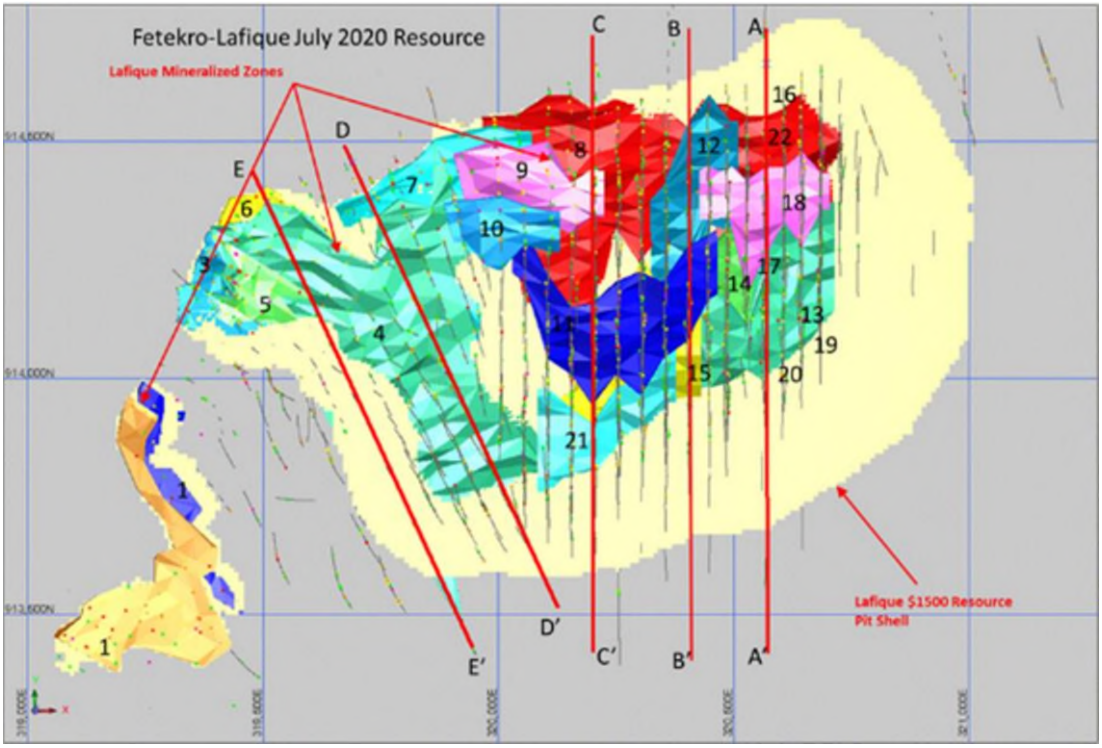
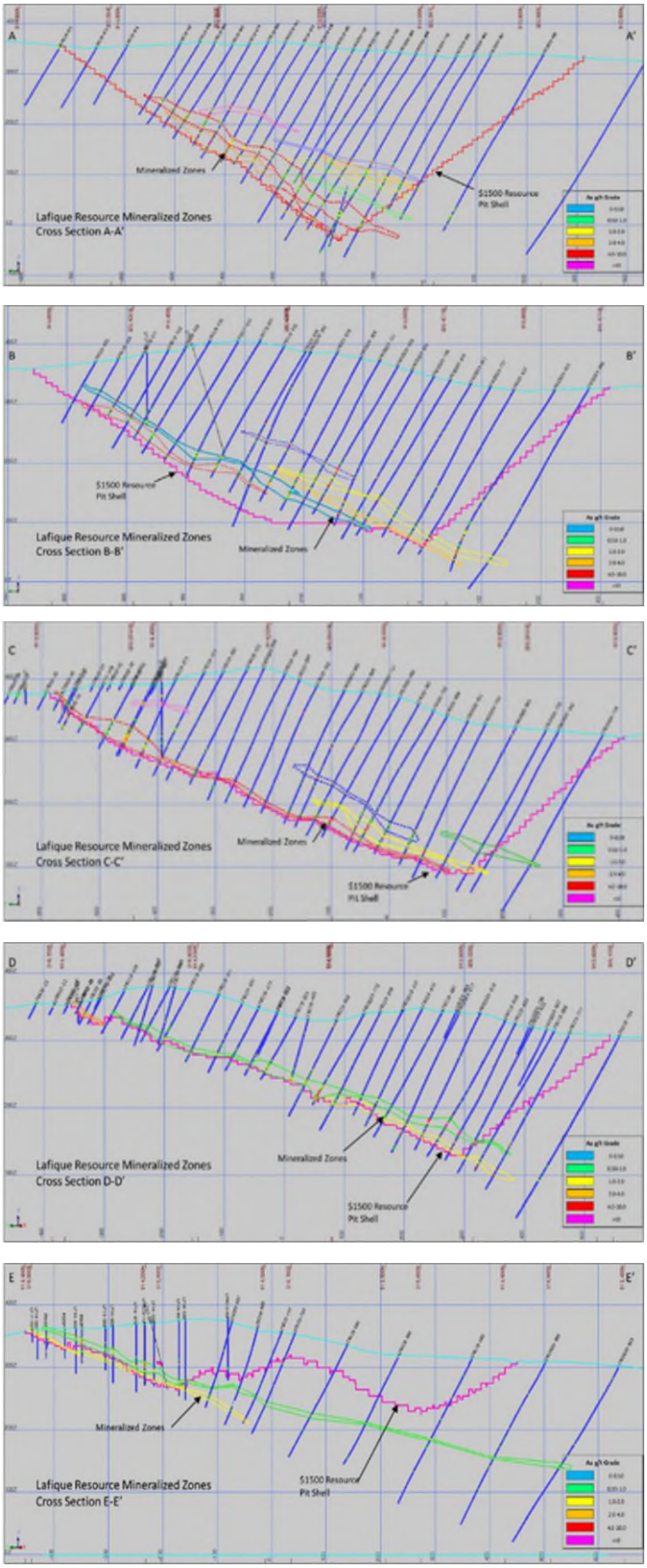


Figure 14.3.2 Lafigué Mineralisation - Cross-sections



14.3.4 Density

In the deposit area, a total of 2,282 density determination measurements were undertaken on diamond drill core which had been wax coated and measured using the standard water displacement method (Section 11.5).

Density measurements were grouped according to the weathering type and averaged to define the in-situ bulk density of the Lafigué deposit. The laterite, saprolite and transition zones are relatively thin so not many samples were measured. Table 14.3.3 summarises the mean of the density determinations by weathering, whilst the number of determinations for each lithology are shown in brackets.

Table 14.3.3 In-situ Bulk Density Data Summary

Weathering	Density (g/cm^3)
Laterite	2.0 (1)
Saprolite	1.80 (26)
Transition	2.40 (41)
Fresh	2.80 (2214)

14.3.5 Statistical Analyses

Composites of the assays within the Lafigué domains were reviewed by individual mineralised zones using histograms, log-histograms, log-probability plots, high grade sensitivity analysis and graphical inspection of the spatial grade distribution. The Lafigué assays were composited to one metre, using the best fit method to include composites over 50% by vein. Table 14.3.4 presents the uncapped composite statistical analysis by domain.

The one metre composites were capped at between 15 g/t Au and 30 g/t Au by mineralised zone depending on the statistics. Table 14.3.5 presents the capped composite statistical analysis by domain and Figure 14.3.3 presents the histograms and cumulative frequency plot of the different mineralised domains.

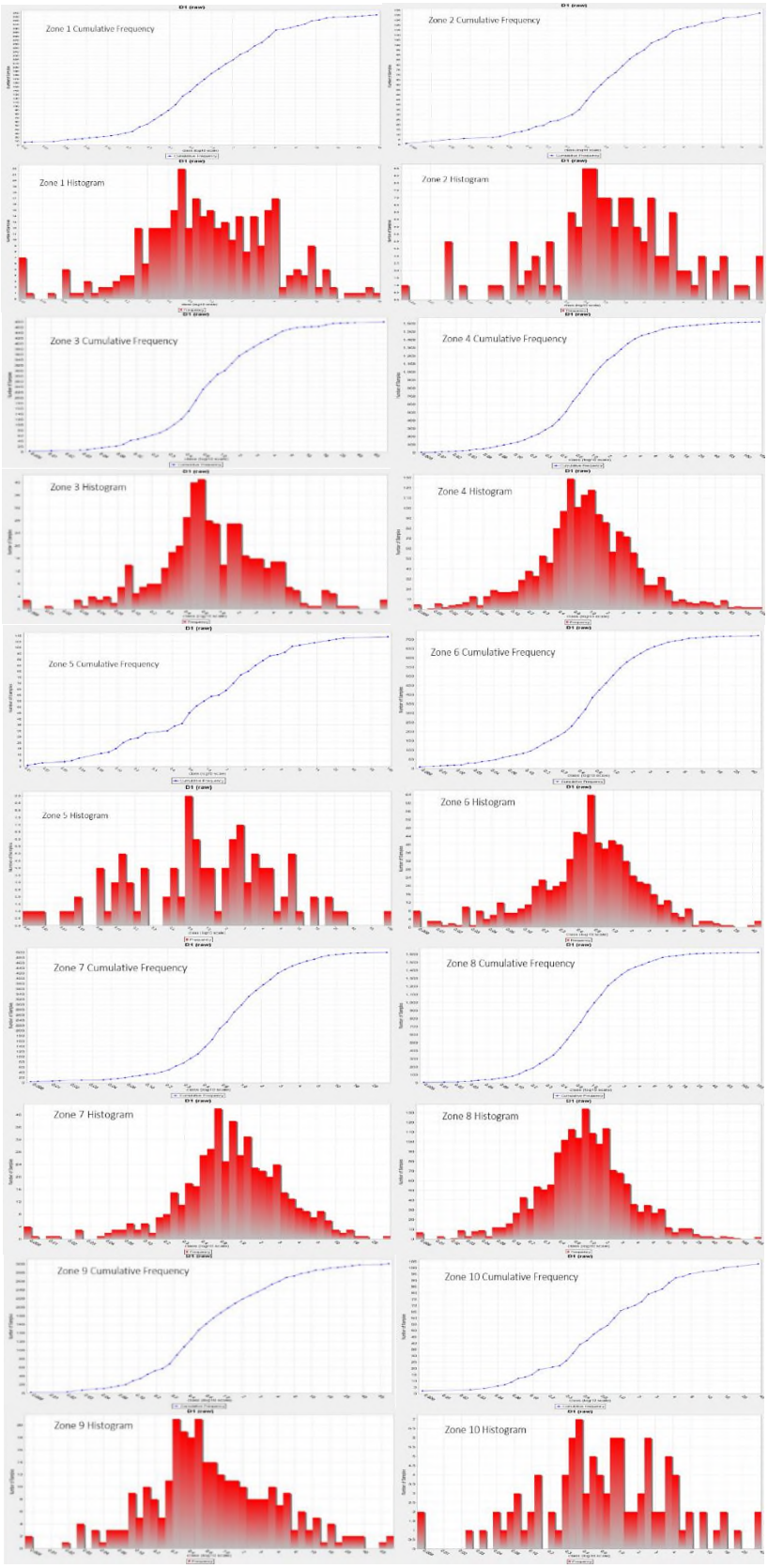
Table 14.3.4 Uncapped Composite Gold Grade (g/t Au) Basic Statistics

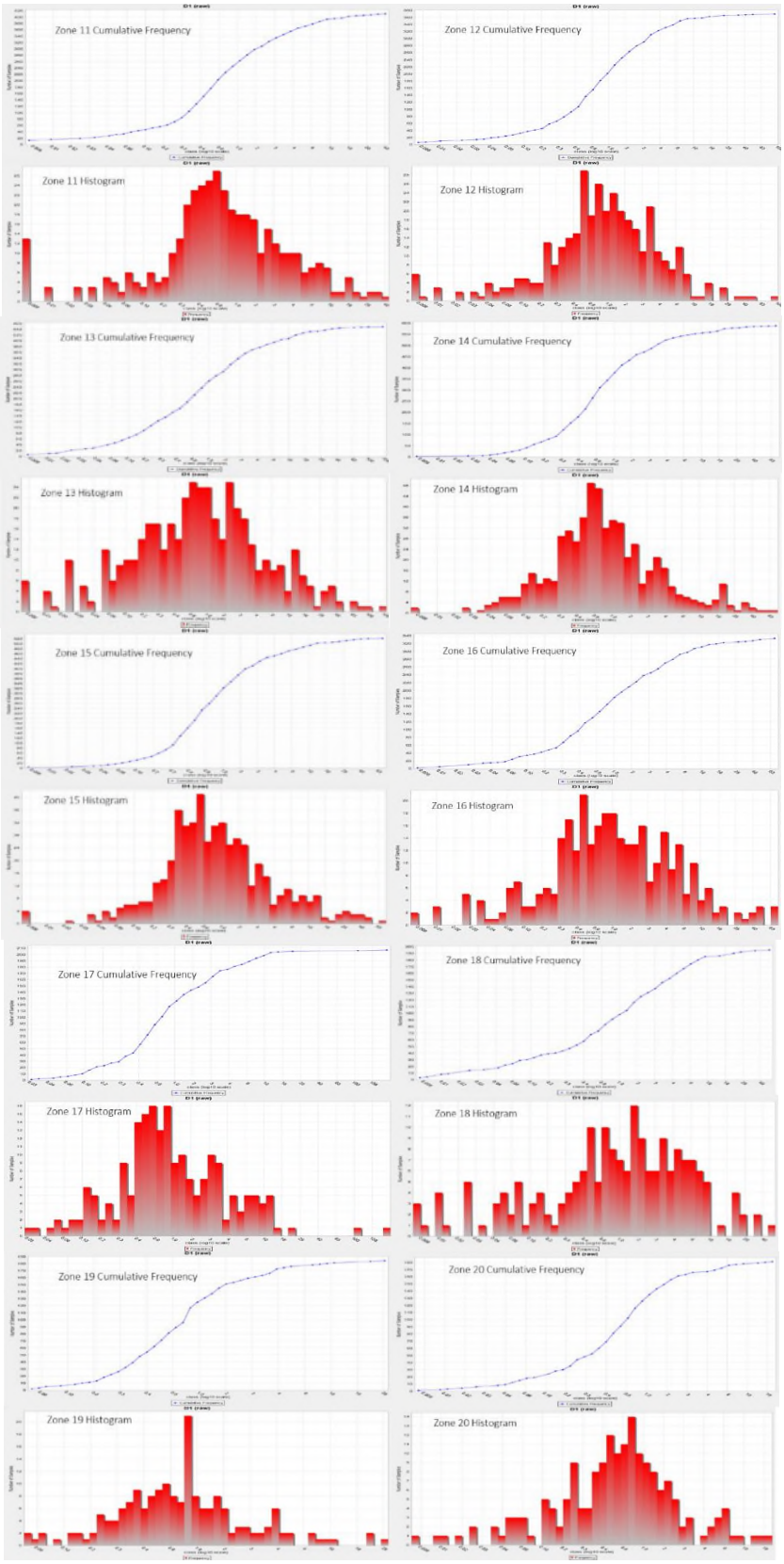
Lafigué Mineralised Domains Statistics Uncapped									
Domain	Number of samples	Minimum value	Maximum value	Mean	Variance	Standard Deviation	Coefficient of variation	Skewness	Kurtosis
1	335	0.02	40.80	2.49	21.82	4.67	1.88	4.69	30.62
2	132	0.01	26.81	2.24	20.09	4.48	2.00	3.87	19.11
3	480	0.01	85.90	2.45	53.21	7.29	2.98	8.42	86.72
4	1,620	0.01	161.95	3.07	103.60	10.18	3.31	8.69	97.76
5	109	0.01	104.69	3.90	121.64	11.03	2.83	7.30	64.62
6	721	0.01	49.86	1.41	14.11	3.76	2.67	8.93	99.19
7	500	0.01	37.70	1.67	8.16	2.86	1.71	6.05	60.14
8	1,747	0.01	170.70	2.37	55.52	7.45	3.14	13.16	241.29
9	301	0.01	86.72	3.10	76.74	8.76	2.83	6.16	48.29
10	103	0.01	40.10	2.58	35.43	5.95	2.31	4.61	26.36
11	410	0.01	43.60	2.24	21.88	4.68	2.09	4.77	31.38
12	369	0.01	100.30	2.56	49.24	7.02	2.75	9.27	113.08
13	449	0.01	166.10	4.04	146.70	12.11	3.00	7.92	87.05
14	588	0.01	78.70	2.62	45.11	6.72	2.56	6.03	50.21
15	522	0.01	78.56	2.78	46.14	6.79	2.44	5.65	44.87
16	333	0.01	76.70	3.73	85.26	9.23	2.47	5.19	33.41
17	207	0.02	249.60	3.76	361.62	19.02	5.05	11.35	140.14
18	195	0.01	54.60	3.28	42.51	6.52	1.99	4.52	28.46
19	184	0.05	27.30	1.58	10.13	3.18	2.02	5.43	37.20
20	181	0.01	19.09	1.30	5.77	2.40	1.85	4.39	26.37
21	303	0.01	55.50	2.06	24.98	5.00	2.42	6.43	55.92
22	1,599	0.01	127.60	3.61	95.87	9.79	2.71	6.69	60.81
Laterite	226	0.03	55.40	2.92	31.49	5.61	1.92	5.60	43.51

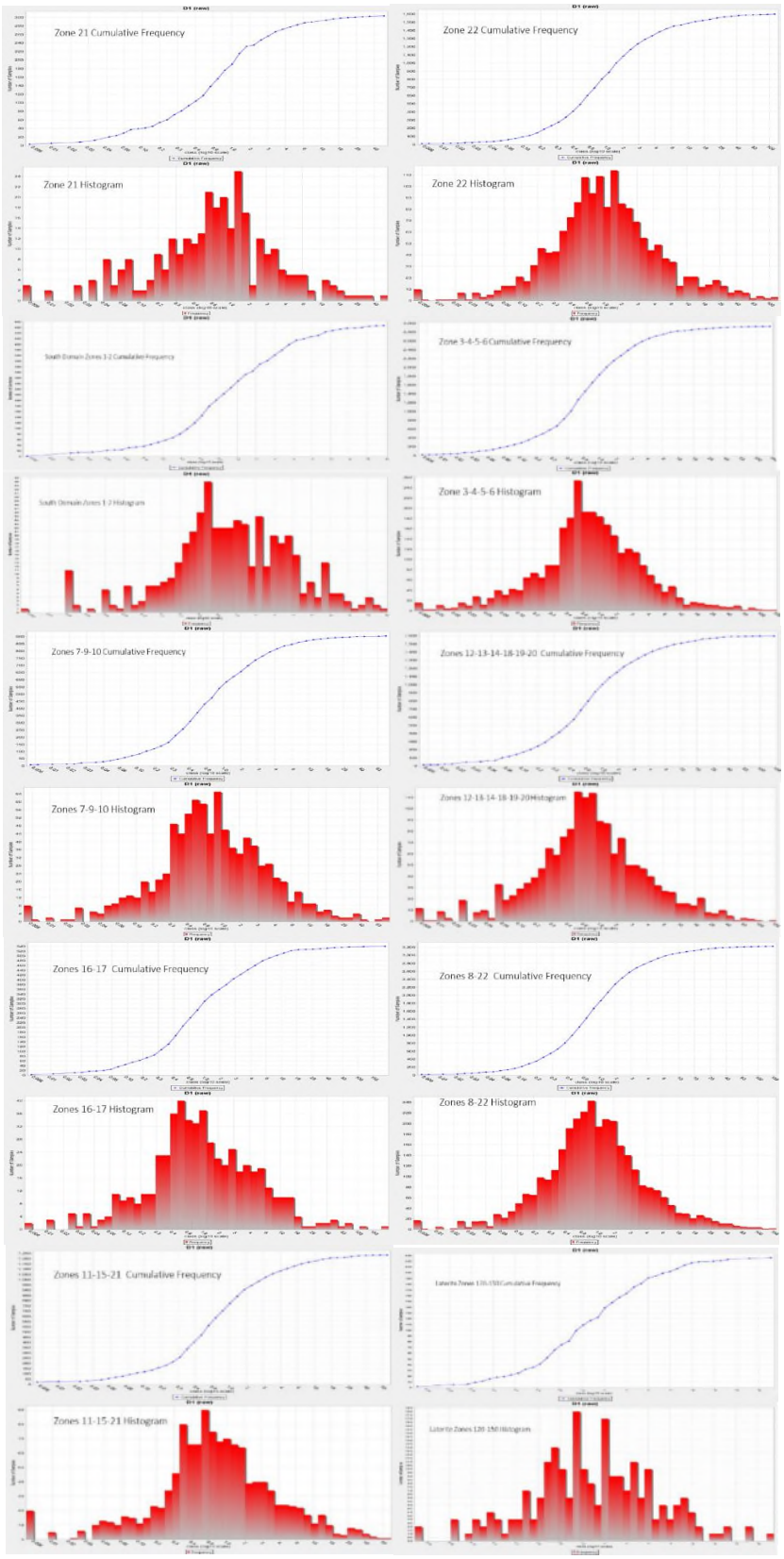
Table 14.3.5 Capped Composite Gold Grade (g/t Au) Basic Statistics

Lafigué Mineralised Domains Statistics Capped									
Domain	Number of samples	Minimum value	Maximum value	Mean	Variance	Standard Deviation	Coefficient of variation	Skewness	Kurtosis
1	335	0.015	15.00	2.22	10.24	3.20	1.44	2.46	8.92
2	132	0.005	15.00	1.98	10.71	3.27	1.65	2.77	10.27
3	493	0.005	15.00	1.83	8.83	2.97	1.63	3.15	13.37
4	1,658	0.005	25.00	2.21	18.14	4.26	1.93	3.92	19.46
5	109	0.0091	25.00	3.10	29.15	5.40	1.74	2.89	11.06
6	739	0.005	15.00	1.21	4.28	2.07	1.71	4.17	23.91
7	512	0.005	20.00	1.62	6.10	2.47	1.53	3.94	22.91
8	1,725	0.005	30.00	2.09	16.45	4.06	1.94	4.49	26.34
9	303	0.005	20.00	2.28	17.50	4.18	1.84	2.94	11.44
10	104	0.005	20.00	2.03	12.16	3.49	1.71	2.95	12.12
11	411	0.005	25.00	2.14	16.12	4.02	1.88	3.72	18.68
12	380	0.005	25.00	2.14	14.16	3.76	1.76	4.08	21.96
13	455	0.005	25.00	2.98	30.92	5.56	1.87	2.82	10.44
14	601	0.005	25.00	2.28	20.77	4.56	2.00	3.53	15.54
15	533	0.005	25.00	2.36	20.45	4.52	1.92	3.55	16.27
16	335	0.005	25.00	2.95	26.27	5.13	1.74	3.02	12.36
17	215	0.0225	25.00	2.16	12.59	3.55	1.65	3.28	16.21
18	197	0.005	25.00	2.99	24.93	4.99	1.67	2.99	12.18
19	186	0.03	25.00	1.51	7.40	2.72	1.80	5.09	36.60
20	189	0.005	18.23	1.29	5.47	2.34	1.81	4.54	27.84
21	304	0.005	25.00	1.90	14.27	3.78	1.99	4.17	22.46
22	1,625	0.005	30.00	2.91	31.27	5.59	1.92	3.47	15.22
Laterite	226	0.03	20.00	2.63	14.24	3.77	1.44	2.90	12.26

Figure 14.3.3 Capped Composite Gold Log Histogram and Cumulative Frequency Plots




Zone 11 Cumulative Frequency
Zone 12 Cumulative Frequency
Zone 11 Histogram
Zone 12 Histogram
Zone 13 Cumulative Frequency
Zone 14 Cumulative Frequency
Zone 13 Histogram
Zone 14 Histogram
Zone 15 Cumulative Frequency
Zone 16 Cumulative Frequency
Zone 15 Histogram
Zone 16 Histogram
Zone 17 Cumulative Frequency
Zone 18 Cumulative Frequency
Zone 17 Histogram
Zone 18 Histogram
Zone 19 Cumulative Frequency
Zone 20 Cumulative Frequency
Zone 19 Histogram
Zone 20 Histogram


Zone 21 Cumulative Frequency
Zone 22 Cumulative Frequency
Zone 21 Histogram
Zone 22 Histogram
Zone 3-4-5-6 Cumulative Frequency
Zone 3-4-5-6 Histogram
Zones 7-9-10 Cumulative Frequency
Zones 12-13-14-18-19-20 Cumulative Frequency
Zones 7-9-10 Histogram
Zones 16-17 Cumulative Frequency
Zones 8-22 Cumulative Frequency
Zones 16-17 Histogram
Zones 8-22 Histogram
Zones 11-15-21 Cumulative Frequency
Zones 11-15-21 Histogram

14.3.6 Variography

Variography was used to analyse the spatial continuity within the mineralised zones and to determine appropriate estimation inputs to the gold grade estimation by ordinary kriging (OK).

The variogram modelling process followed involved the following steps.

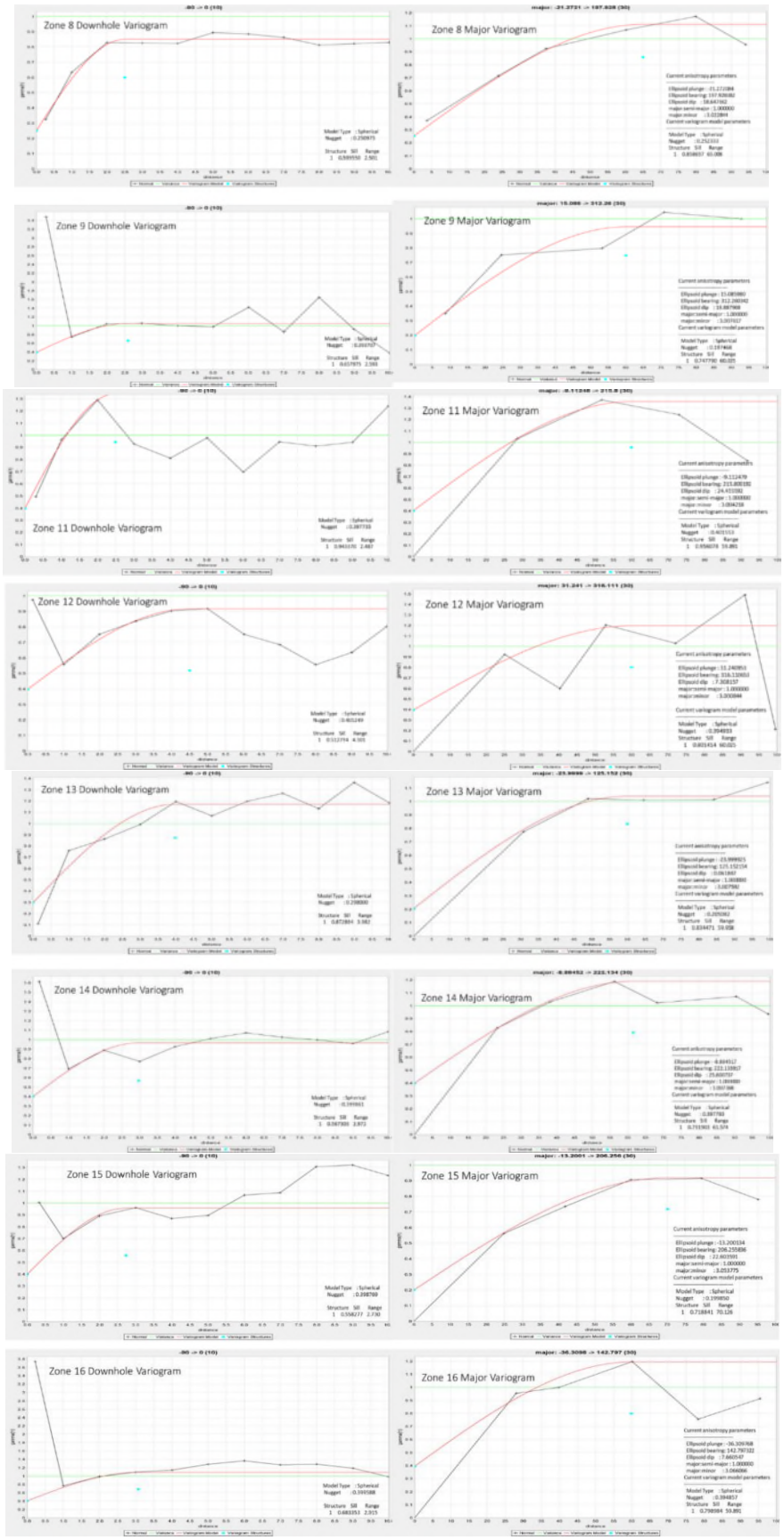
- Calculate and model the down hole variogram to characterise the nugget effect.
- Calculate a fan of variograms with a variogram map within the plane of greatest continuity to identify the direction of maximum continuity within the plane.
- Model the variogram in the direction of maximum continuity and the orthogonal directions.

Variogram modelling of the main mineralised zones produced reasonable results. The variogram modelling for the more sparsely sampled domains was less defined, so average parameters were defined for those domains. See Figure 14.3.4 for the variograms of the various domains.

Variogram nugget effects were typically in the range of 0.2 to 0.4, indicating a moderate degree of short scale variability as would be expected in gold deposits. Variogram ranges were typically in the order of 50 m to 70 m indicating maximum spatial continuity is greater than the average drill hole spacing.

Table 14.3.6 following summarises the variogram parameters and the relative variogram parameters used in the MRE. The laterite mineralisation did not yield reasonable variograms so inverse distance squared (ID2) was used to estimate the gold grade.

Figure 14.3.4 Variograms



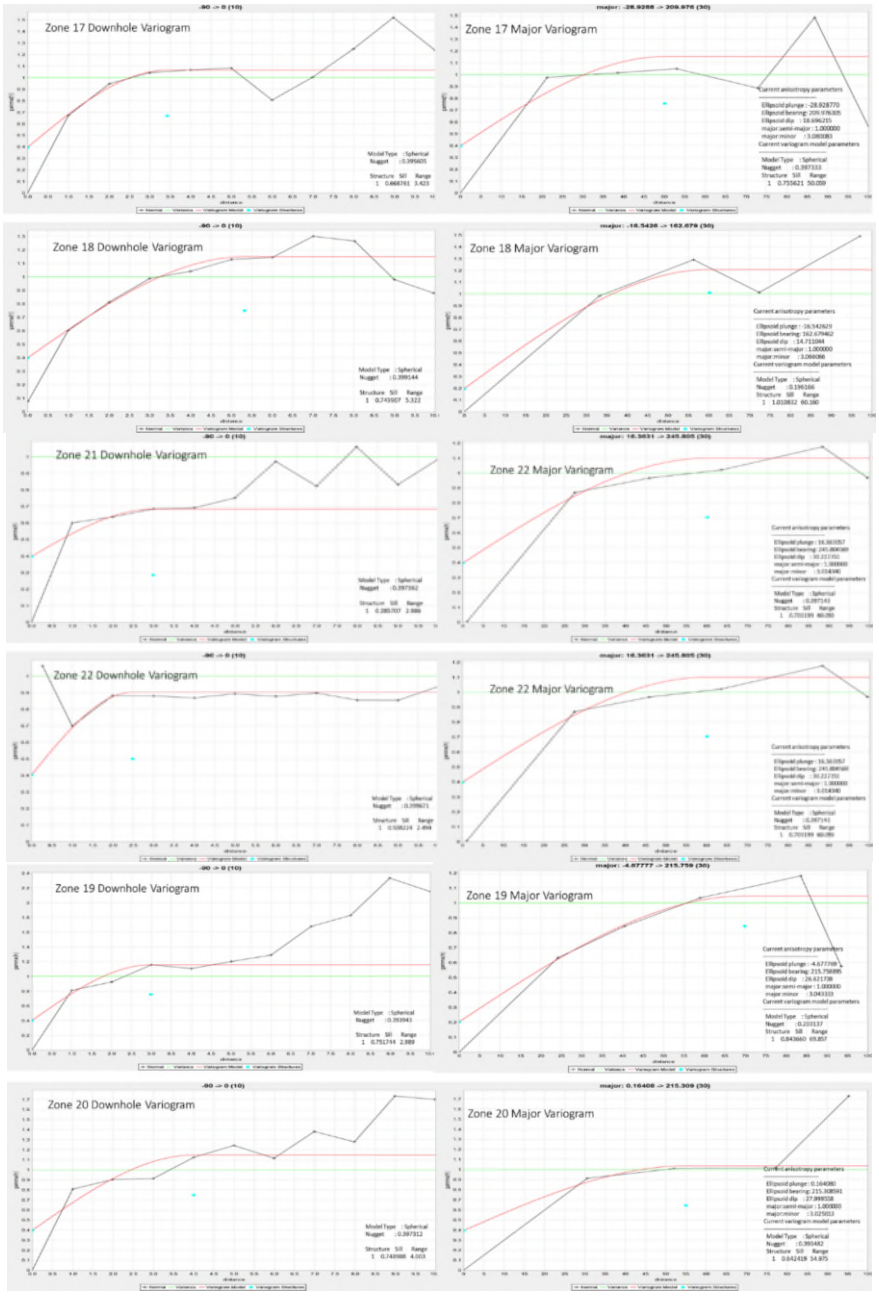

Zone 17 Downhole Variogram
Zone 17 Major Variogram
Zone 18 Downhole Variogram
Zone 18 Major Variogram
Zone 21 Downhole Variogram
Zone 22 Major Variogram
Zone 22 Downhole Variogram
Zone 22 Major Variogram
Zone 19 Downhole Variogram
Zone 19 Major Variogram
Zone 20 Downhole Variogram
Zone 20 Major Variogram

Table 14.3.6 Lafiqué Variogram Results and OK Modelling Parameters

Domain	Search Ellipse Orientation		Nugget	Sill	Range	Estimation Pass 1					Estimation Pass 2				
	Dip Dir.	Dip				Min. Number Samples	Max. Number Samples	Search Radius	Major/ Semi-Major Ratio	Major/ Minor Ratio	Min. Number Samples	Max. Number Samples	Search Radius	Major/ Semi-Major Ratio	Major/ Minor Ratio
1	253	6	0.4	0.81	55	7	20	50	1	3	3	20	75	1	3
2	121	-15	0.4	0.8	50	7	20	50	1	3	3	20	75	1	3
3	123	-18	0.4	0.8	50	7	20	50	1	3	3	20	75	1	3
4	160	-17	0.4	80	70	7	20	50	1	3	3	20	75	1	3
5	148	-14	0.4	0.8	60	7	20	50	1	3	3	20	75	1	3
6	175	-22	0.4	0.75	60	7	20	50	1	3	3	20	75	1	3
7	155	-17	0.4	0.6	55	7	20	50	1	3	3	20	75	1	3
8	155	-28	0.25	0.85	65	7	20	50	1	3	3	20	75	1	3
9	185	-24	0.2	0.75	60	7	20	50	1	3	3	20	75	1	3
10	190	-18	0.4	0.8	50	7	20	50	1	3	3	20	75	1	3
11	145	-26	0.4	0.95	60	7	20	50	1	3	3	20	75	1	3
12	150	-32	0.4	0.8	60	7	20	50	1	3	3	20	75	1	3
13	125	-24	0.3	0.83	60	7	20	50	1	3	3	20	75	1	3
14	150	-27	0.4	0.8	60	7	20	50	1	3	3	20	75	1	3
15	145	-26	0.4	0.72	70	7	20	50	1	3	3	20	75	1	3
16	130	-37	0.4	0.8	60	7	20	50	1	3	3	20	75	1	3
17	175	-34	0.4	0.75	50	7	20	50	1	3	3	20	75	1	3
18	120	-22	0.4	1	60	7	20	50	1	3	3	20	75	1	3
19	135	-27	0.4	0.85	70	7	20	50	1	3	3	20	75	1	3
20	125	-28	0.4	0.65	55	7	20	50	1	3	3	20	75	1	3
21	135	-18	0.4	0.75	60	7	20	50	1	3	3	20	75	1	3
22	130	-34	0.4	0.7	60	7	20	50	1	3	3	20	75	1	3

Laterite ID2 Estimation

Zone	Search Ellipse (Surpac ZXY LRL)								
	Bearing	Plunge	Dip	Major:Semi Major Ratio	Major:Minor Ratio	Range Pass1	Range Pass2	Min Samples	Max Samples
120	62	0	0	1.5	3	45	75	5	20
130	0	0	0	1.5	3	45	75	5	20
140	0	0	0	1.5	3	45	75	5	20
150	70	0	0	1.5	3	45	75	5	20

14.3.7 Block Modelling

The Lafigué block model was created in GEOVIA Surpac™, the set-up is presented in Table 14.3.7. Several criteria were considered when setting up the block model including mineralisation trend, drill hole (data) spacing, and potential mine design and mining selectivity implications (such as the current or anticipated selective mining unit (SMU)).

Table 14.3.7 Lafigué Block Model Setup

Type	Y	X	Z
Minimum Coordinates	913230	318970	-50
Maximum Coordinates	914820	321100	435
User Block Size (m)	10	10	5
Min. Block Size (m)	2.5	2.5	1.25
Rotation	0.000	0.000	0.000

The Lafigué block model attributes are presented in Table 14.3.8.

Table 14.3.8 Lafigué Block Model Attributes

Attribute Name	Type	Decimals	Background	Description
au_id2	Real	2	0	ID2 gold grade estimate
au_ok	Real	2	0	OK gold grade estimate
aver_dist	Real	2	0	average distance to sample ID2 estimate
aver_dist_ok	Real	2	0	average distance to sample OK estimate
block_var	Real	2	0	block variance
class	Integer	-	0	classification 2=indicated; 3=inferred
cond_bias	Real	2	0	conditional bias
density	Real	2	0	
geology	Character	-		
krig_eff	Real	2	0	kriging efficiency
krig_var	Real	2	0	kriging variance
nearest	Real	2	0	Nearest sample distance ID2 estimate
nearest_ok	Real	2	0	nearest sample distance OK estimate
numb_dhs	Integer	-	0	number drillholes ID2 estimate
numb_dhs_ok	Integer	-	0	number drillholes OK estimate
numb_samp_ok	Integer	-	0	number samples OK estimate
numb_sampl	Integer	-	0	number samples ID2 estimate
ore_zone	Integer	-	0	mineralized zone code
oxidation	Character	-		Oxide, transition, fresh
pass_id	Integer	-	0	estimation pass ID2 estimate
pass_ok	Integer	-	0	estimation pass OK estimate
rockcode	Integer	-	0	rockcode for whittle
weathering	Character	-		latr=laterite; sapr=saprolite; sprk=transition; fresh=fresh; air

14.3.8 Grade Estimation

The grade estimation interpolation was done using ordinary kriging (OK) for each mineralised domain using the uniquely coded capped one metre down-hole composite data specific to that domain. The laterite hosted mineralisation domains were estimated with ID2. ID2 estimation was also done in the in-situ domains for validation and comparison purposes. All block estimates were based on grade interpolation into parent cells of 10 m(Y) x 10 m(X) x 2.5 m (Z). Block discretisation points were set to 3(Y) x 3X) x 3(Z).

Search ellipse orientations for each domain interpolation were orientated to follow the direction of the mineralised domain. A two-pass search strategy was used for grade estimation. The first pass used a search radius of 50 m and was increased for the second search pass to 75 m. A minimum of seven and a maximum of 20 samples were used for the first pass, and a minimum of three and a maximum of 20 for the second pass. Table 14.3.6 following, illustrates the estimation parameters used for each domain.

14.3.9 Block Model Validation

Modelled estimates within the domains have been compared with the down-hole composite grades by:

- Visually inspecting the drillhole and composite grade with the block grades.
- Comparing the average composite grades with the block grades for each domain (see Table 14.9.3).
- Comparing the ID2 and OK estimated grades.
- By swath plots comparing the composite grade with the estimated grade across each domain.

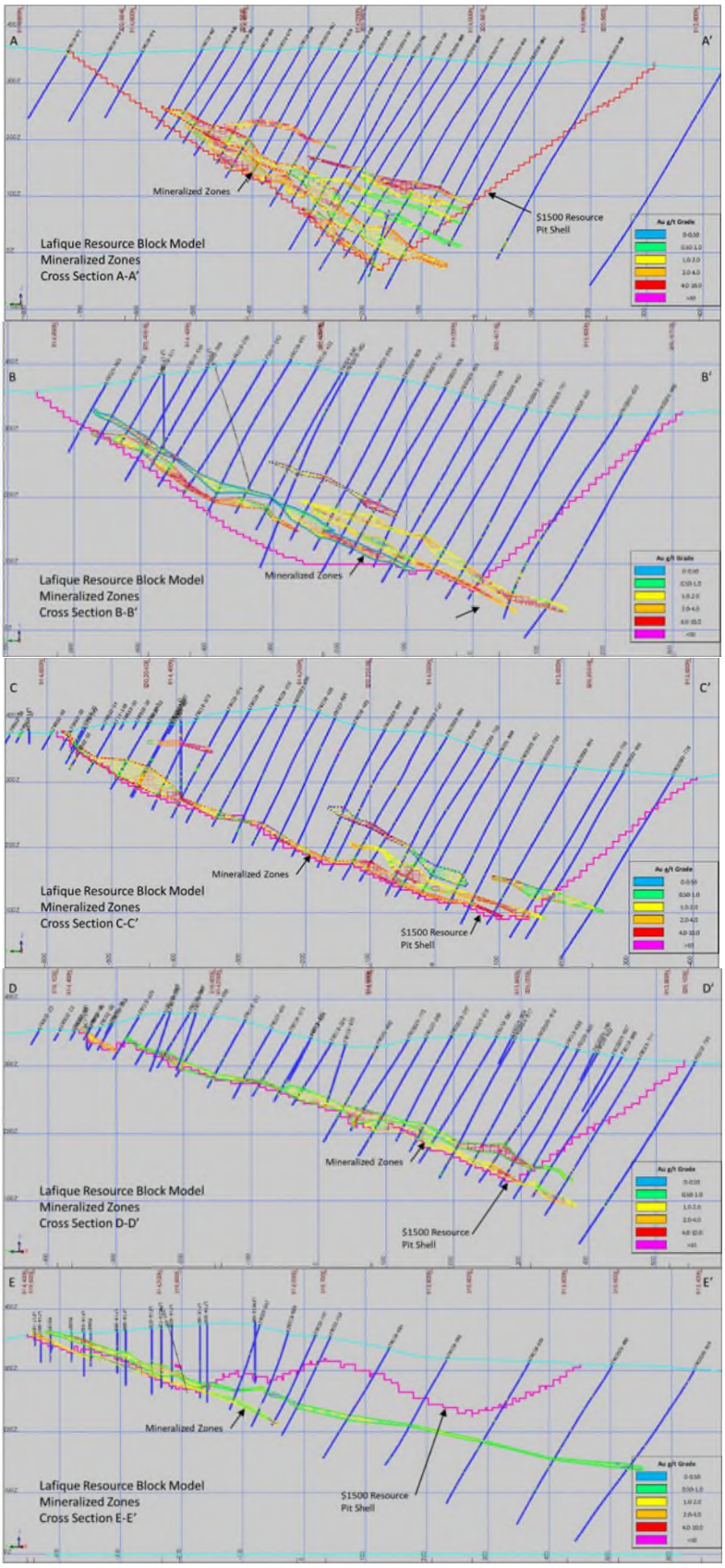
A visual validation, comparing the block estimates with the composite data in cross-section and in 3D was completed. The estimates compared well with the composite data well with some degree of grade smoothing of the block estimates as would be expected from OK.

Representative cross-sections, showing the block model grades versus drillhole assay values are presented in Figure 14.3 5.

Table 14.3.9 Block Model OK vs ID2 vs Composite Grade Comparison

Domain	Au	Number	Minimum	Maximum	Mean	Standard Deviation
1	Composite	335	0.02	15.00	2.22	3.20
	Au OK	1300	0.18	7.28	2.04	1.34
	Au ID2	1300	0.10	7.48	2.13	1.45
2	Composite	132	0.01	15.00	1.98	3.27
	Au OK	895	0.11	11.09	2.03	1.73
	Au ID2	895	0.10	12.71	2.02	1.99
3	Composite	493	0.01	15.00	1.83	2.97
	Au OK	1386	0.13	7.20	1.51	1.06
	Au ID2	1386	0.06	7.56	1.54	1.04
4	Composite	1658	0.01	25.00	2.21	4.26
	Au OK	8851	0.08	20.76	1.96	2.02
	Au ID2	8851	0.06	23.04	1.98	2.13
5	Composite	109	0.01	25.00	3.10	5.40
	Au OK	454	0.44	9.74	3.42	2.27
	Au ID2	454	0.43	10.91	3.47	2.34
6	Composite	739	0.01	15.00	1.21	2.07
	Au OK	4224	0.04	8.51	1.25	0.95
	Au ID2	4224	0.03	9.93	1.26	0.99
7	Composite	512	0.01	20.00	1.62	2.47
	Au OK	2672	0.19	12.49	1.59	1.11
	Au ID2	2672	0.04	12.74	1.63	1.28
8	Composite	1725	0.01	30.00	2.09	4.06
	Au OK	8225	0.13	22.74	2.16	2.02
	Au ID2	8225	0.15	24.05	2.20	2.02
9	Composite	303	0.01	20.00	2.28	4.18
	Au OK	1204	0.07	12.51	2.50	1.98
	Au ID2	1204	0.20	13.22	2.38	1.91
10	Composite	104	0.01	20.00	2.03	3.49
	Au OK	641	0.21	8.89	1.92	1.15
	Au ID2	641	0.11	9.71	1.91	1.20
11	Composite	411	0.01	25.00	2.14	4.02
	Au OK	2789	0.22	9.24	2.11	1.43
	Au ID2	2789	0.15	10.79	2.14	1.53
12	Composite	380	0.01	25.00	2.14	3.76
	Au OK	2372	0.19	10.52	2.08	1.46
	Au ID2	2372	0.16	12.82	2.05	1.47
13	Composite	455	0.01	25.00	2.98	5.56
	Au OK	2446	0.00	12.72	2.89	2.32
	Au ID2	2446	0.00	14.10	2.92	2.38
14	Composite	601	0.01	25.00	2.28	4.56
	Au OK	2820	0.25	11.51	2.59	1.98
	Au ID2	2820	0.25	13.40	2.61	2.07
15	Composite	533	0.01	25.00	2.36	4.52
	Au OK	3451	0.26	17.07	2.39	1.90
	Au ID2	3451	0.18	19.53	2.39	2.04
16	Composite	335	0.01	25.00	2.95	5.13
	Au OK	1670	0.23	15.84	2.47	2.03
	Au ID2	1670	0.20	18.14	2.49	2.14
17	Composite	215	0.02	25.00	2.16	3.55
	Au OK	1079	0.24	11.21	1.97	1.54
	Au ID2	1079	0.29	13.31	1.98	1.67
18	Composite	197	0.01	25.00	2.99	4.99
	Au OK	1180	0.26	16.23	2.67	2.19
	Au ID2	1180	0.22	18.01	2.74	2.36
19	Composite	186	0.03	25.00	1.51	2.72
	Au OK	1045	0.31	9.68	1.52	1.19
	Au ID2	1045	0.30	10.60	1.50	1.22
20	Composite	189	0.01	18.23	1.29	2.34
	Au OK	1030	0.16	7.41	1.30	1.01
	Au ID2	1030	0.13	8.69	1.26	1.07
21	Composite	304	0.01	25.00	1.90	3.78
	Au OK	2037	0.18	12.30	2.14	1.93
	Au ID2	2037	0.09	14.98	2.11	2.07
22	Composite	1625	0.01	30.00	2.91	5.59
	Au OK	8335	0.00	19.47	2.80	2.30
	Au ID2	8335	0.00	18.98	2.82	2.44
Iatr	Composite	226	0.03	20.00	2.63	3.77
	Au OK	1564	0.00	18.25	2.30	1.90
	Au ID2	1564	0.00	18.25	2.30	1.90

Figure 14.3.5 Representative Cross-sections of OK Block Model Grade vs Drillhole Assays

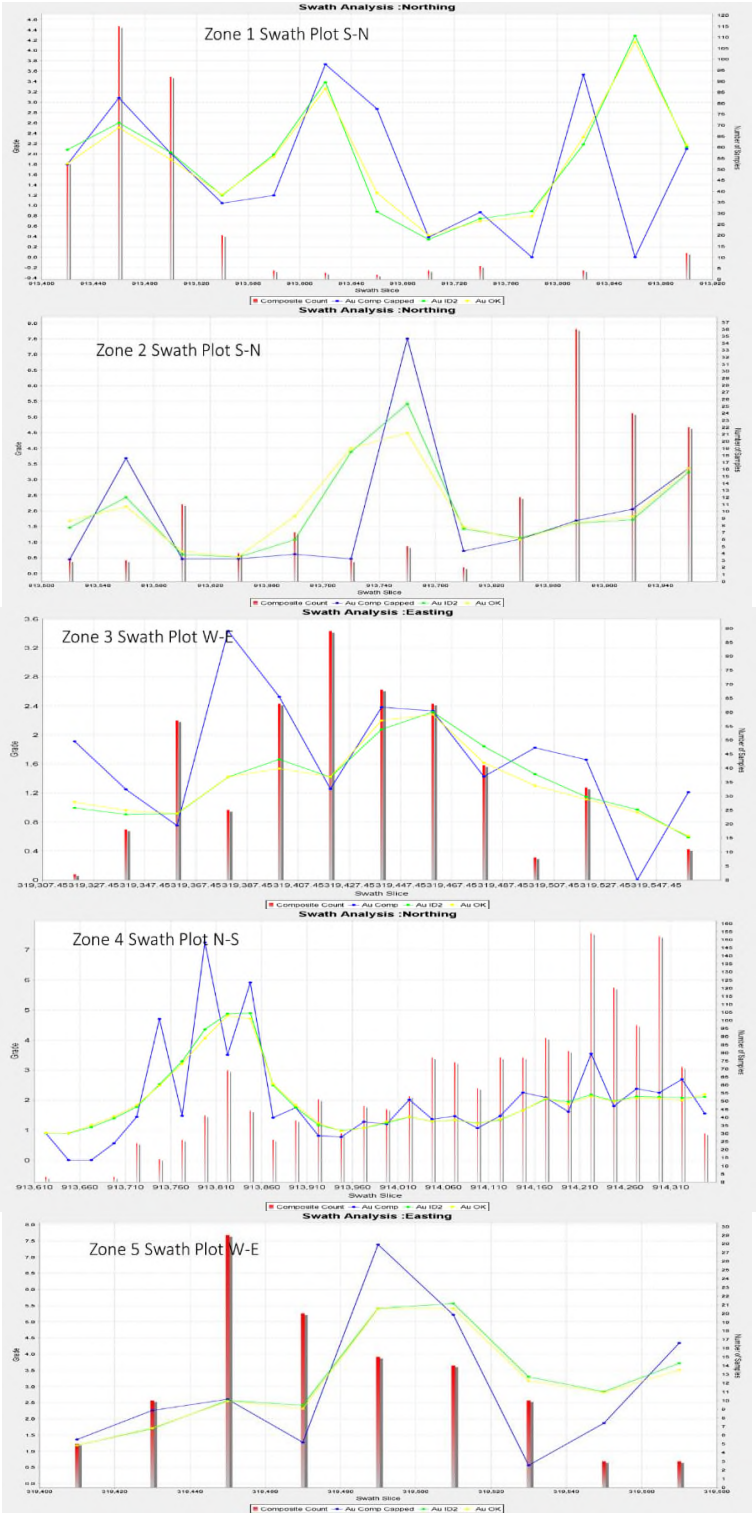


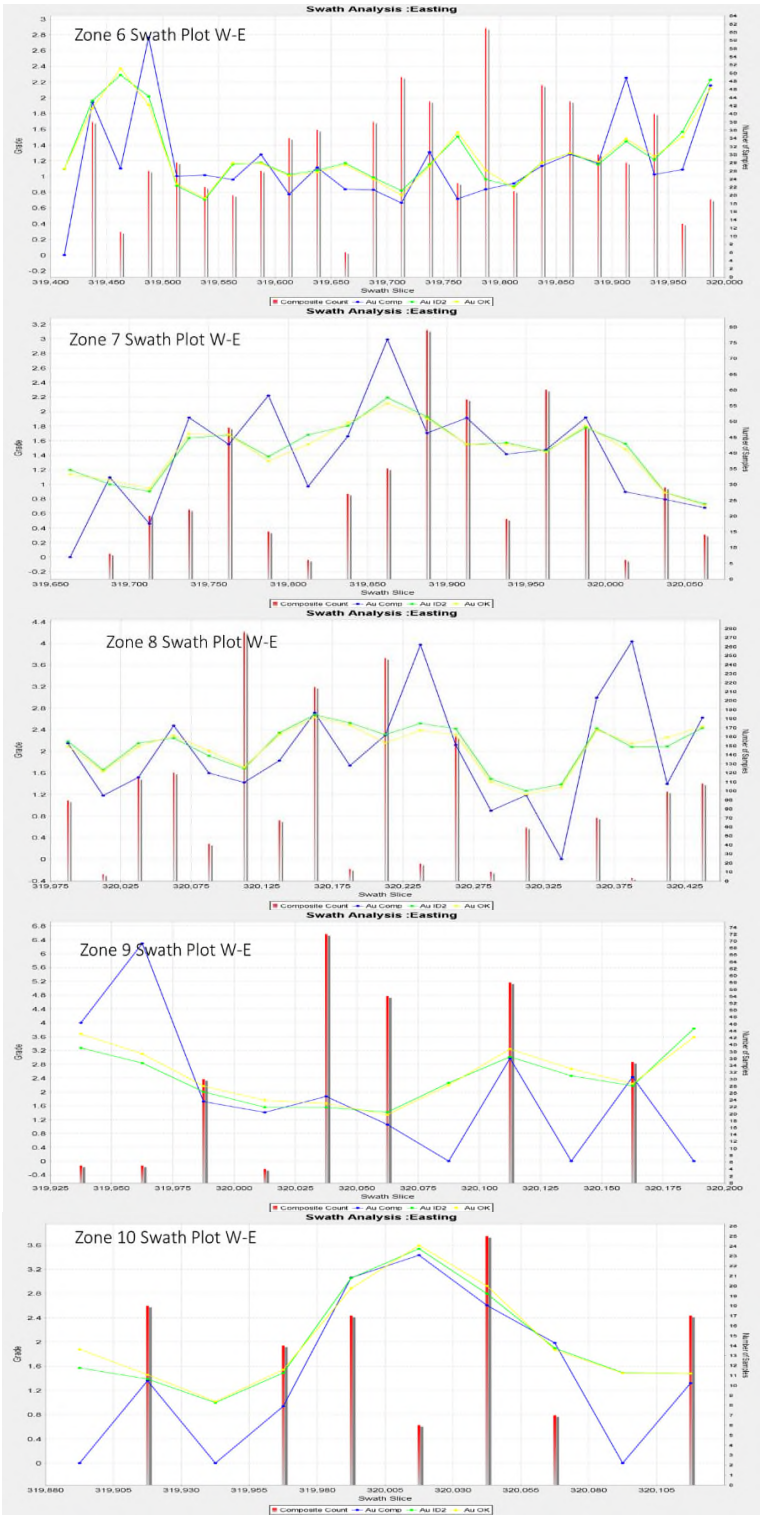
Swath plots showing the estimated grade (OK and ID2), the composite sample value and number of samples were also used for validation.

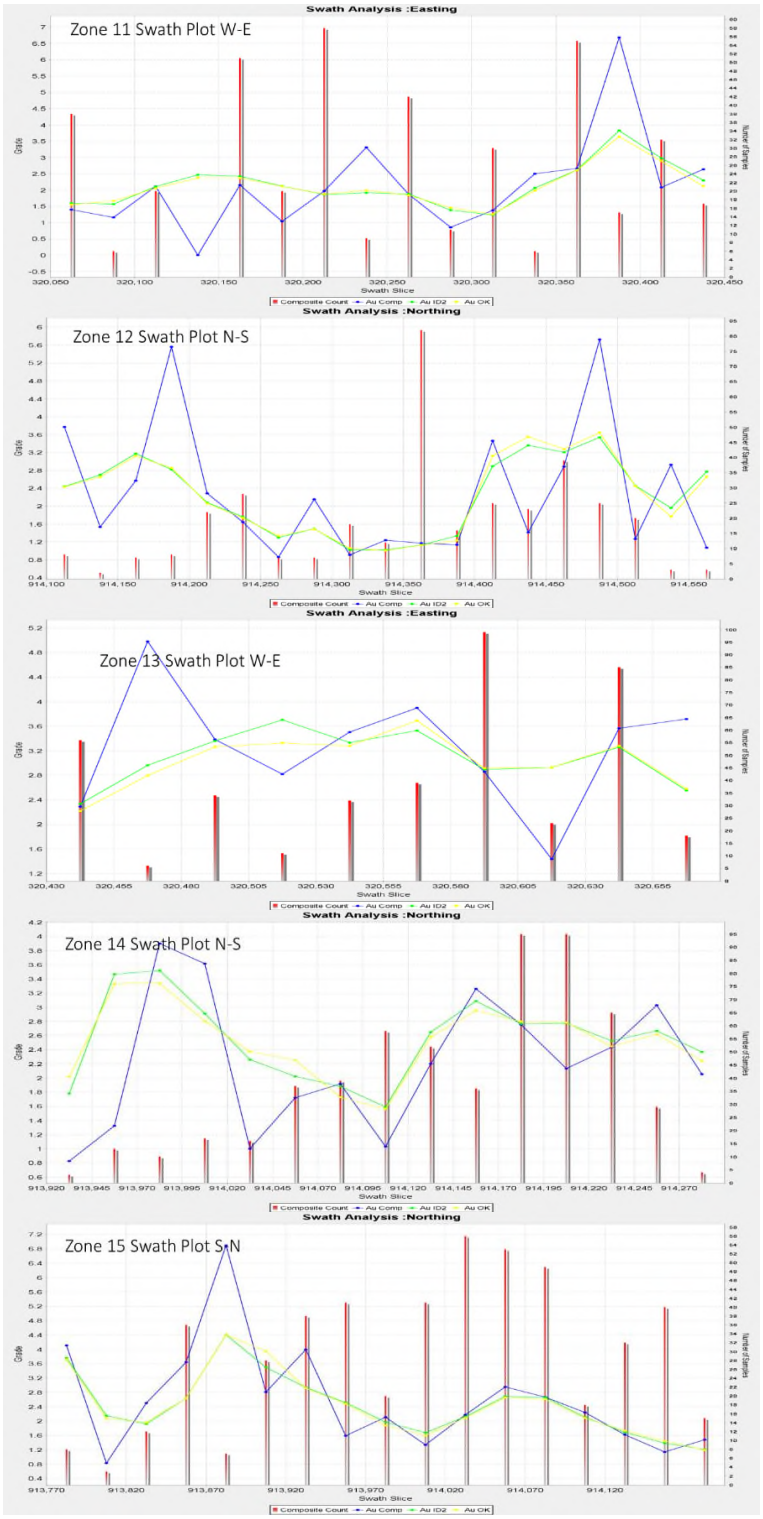
There is generally good agreement between the block estimate and composite mean gold grades for all domains. As expected, the estimated gold grade is smoother when compared to the often variable, composite mean gold grades.

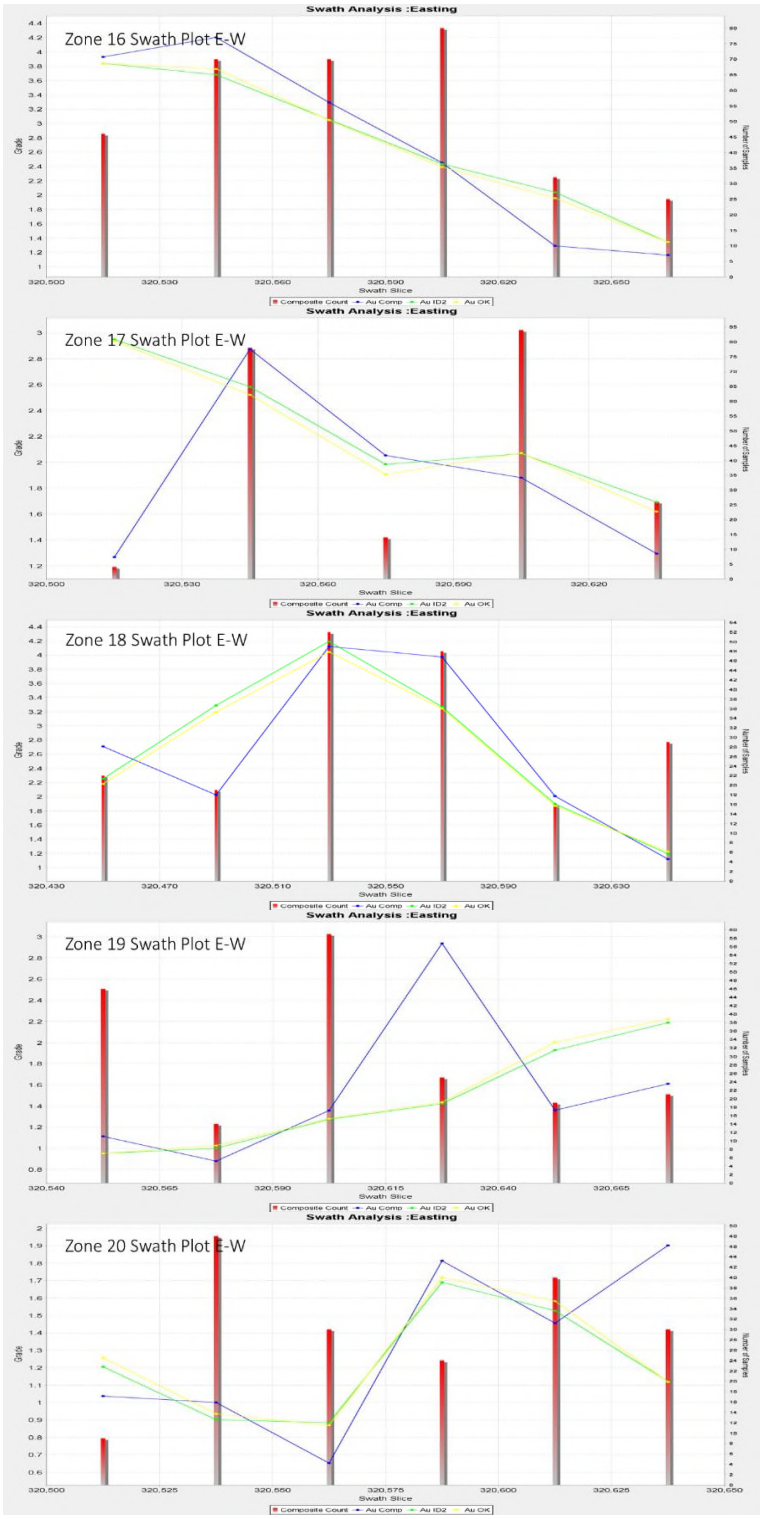
Figure 14.3.6 illustrates swath plots by domain.

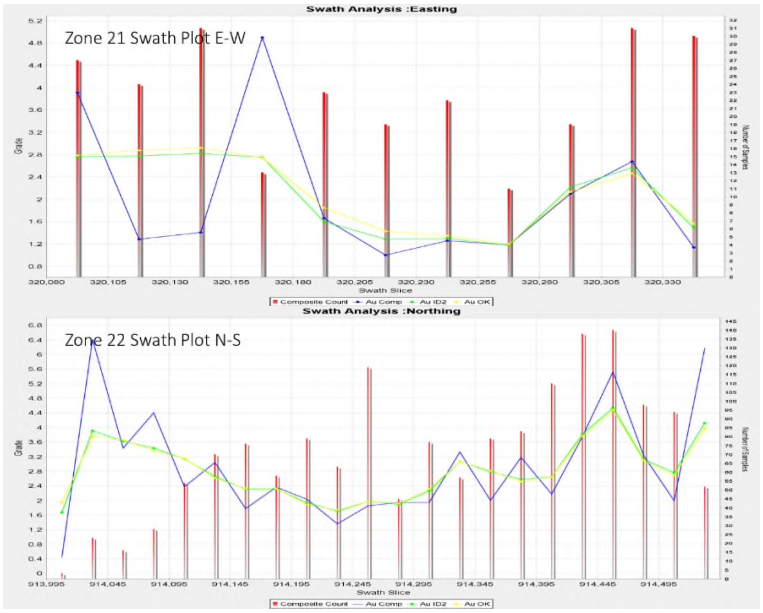
Figure 14.3.6 Swath Plot Block Model Validation




Swath Analysis :Easting
Zone 6 Swath Plot W-E
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK
Swath Analysis :Easting
Zone 7 Swath Plot W-E
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK
Swath Analysis :Easting
Zone 8 Swath Plot W-E
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK
Swath Analysis :Easting
Zone 9 Swath Plot W-E
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK
Swath Analysis :Easting
Zone 10 Swath Plot W-E
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK


Swath Analysis :Easting
Zone 11 Swath Plot W-E
Swath Slice
Swath Analysis :Northing
Zone 12 Swath Plot N-S
Swath Slice
Swath Analysis :Easting
Zone 13 Swath Plot W-E
Swath Slice
Swath Analysis :Northing
Zone 14 Swath Plot N-S
Swath Slice
Swath Analysis :Northing
Zone 15 Swath Plot S-N
Swath Slice


Swath Analysis :Easting
Zone 16 Swath Plot E-W
Swath Slice
Swath Analysis :Easting
Zone 17 Swath Plot E-W
Swath Slice
Swath Analysis :Easting
Zone 18 Swath Plot E-W
Swath Slice
Swath Analysis :Easting
Zone 19 Swath Plot E-W
Swath Slice
Swath Analysis :Easting
Zone 20 Swath Plot E-W
Swath Slice


Swath Analysis :Easting
Zone 21 Swath Plot E-W
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK
Swath Analysis :Northing
Zone 22 Swath Plot N-S
Grade
Number of Samples
Swath Slice
Composite Count
Au Comp
Au ID2
Au OK

14.3.10 Mineral Resource Classification

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 the defined criteria. The final outcome reflects a practical combination of geological knowledge and estimation quality parameters that may be more numerical in nature.

Mineralisation at Lafigué shows very good continuity over most domains. The drill spacing of 40 m to 50 m over the majority of the resource area facilitates classification into the Indicated Mineral and Inferred Mineral Resource categories.

The Indicated Mineral Resource is classified as the area with sample density and drill spacing as defined by estimation pass 1. The pass 2 search strategy is used to classify the Inferred Mineral Resource blocks within the mineralised zones but not meeting the pass 1 (Indicated) criteria.

In addition, the Indicated classification was smoothed to redefine some of the isolated pass 2 blocks within the higher continuity confidence areas as Indicated.

No Measured Mineral Resource is defined.

14.3.11 Mineral Resource Statement

The Lafigué Project has an Indicated Mineral Resource of 32.0 Mt at a grade of 2.40 g/t Au containing 2.47 Moz gold and an Inferred Mineral Resource of 0.82 Mt at a grade of 2.52 g/t Au containing 66 koz of gold. The effective date of the MRE is July 31, 2020 and is reported above a cut-off grade of 0.50 g/t Au and inside a $1,500/oz Au optimised pit shell.

Reporting within an optimised pit shell satisfies the requirement for the Mineral Resource to have 'reasonable prospects for eventual economic extraction (RPEEE). The pit shell optimisation was also done at US$1,300/oz Au and $1,700/oz Au for comparison with the potential reserve and cost sensitivity. Table 14.3.10 following, summarises the economic parameters used to define the pit shells.

Table 14.3.10 Resource Pit Optimisation Parameters

Resource Whittle Pit Shell Estimation Parameters	
Mining Ore Loss	95%
Mining Dilution	10%
Pit Slope	40 degrees
Base mining Cost/t	$2.50
Oxide Mining Cost + Haulage/t	$2.75
Transition Mining Cost + Haulage/t	$3.25
Fresh Mining Cost + Haulage/t	$3.75
Oxide Processing Cost/t	$19.85
Transition Processing Cost/t	$21.17
Fresh Processing Cost/t	$21.17
Gold Recovery Oxide	96%
Gold Recovery Transition	95%
Gold Recovery Fresh	94%
Gold selling cost $/oz (royalties + refining + selling)	$60

Table 14.3.11 below shows the Indicated and Inferred Mineral Resource.

Table 14.3.11 Lafigué Mineral Resource Estimate, as of 31 July 2020

Indicated			Inferred		
Tonnes (kt)	**Grade (Au g/t)**	**Au (koz)**	**Tonnes (kt)**	**Grade (Au g/t)**	**Au (koz)**
32,030	2.40	2,471	820	2.52	66

Notes: Effective date 31 July 2020. 0.5 Au g/t cut-off grade. $1,500/oz Au MII optimised Whittle pit shell constrained. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

Table 14.3.12 shows the comparison and sensitivity to a pit shell at $1,300/oz and $1,700/oz gold prices.

Table 14.3.12 Lafigué Mineral Resource as of July 31, 2020 Gold Price Sensitivity

Pit Shell Au Price ($/oz)	Indicated			Inferred		
	Tonnes (kt)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kt)**	**Grade (Au g/t)**	**Au (koz)**
1,300	30,350	2.44	2,381	672	2.60	56
1,700	32,847	2.38	2,513	928	2.45	73

Table 14.3.13 shows the Mineral Resource by weathering type, constrained by the $1,500/oz Au optimised pit shell and above a 0.5 Au g/t cut-off grade

Table 14.3.13 Lafigué Mineral Resource, as of July 31, 2020 by Weathering Type

Weathering Type	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Au (koz)	Tonnes (kt)	Grade (Au g/t)	Au (koz)
Oxide	1,268	2.12	86	194	2.43	15
Transition	876	2.08	59	79	2.55	7
Fresh	29,886	2.42	2,325	546	2.54	45
Total	**32,030**	**2.40**	**2,470**	**820**	**2.52**	**66**

Table 14.3.13 shows the Mineral Resource by rock type constrained by the $1,500/oz Au optimised pit shell and above a 0.5 Au g/t cut-off grade. (Note, there has been some rounding of numbers resulting in a slight variation in the numbers reported.)

Table 14.3.14 Lafigué Mineral Resource as of July 31, 2020 by Rock Type

Rock Type	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Au (koz)	Tonnes (kt)	Grade (Au g/t)	Au (koz)
laterite	250	2.57	21	110	2.37	8
saprolite	1,024	2.11	69	84	2.37	6
intrusive_felisic	10,758	1.97	681	103	2.12	7
intrusive_mafic	10,311	2.51	831	158	2.27	12
mafic_volcanic	8,836	2.69	763	360	2.82	33
quartz_vein	849	3.76	103	4	2.75	0
Total	**32,030**	**2.40**	**2,468**	**820**	**2.52**	**66**

Table 14.3.15 following shows the Mineral Resource sensitivity to cut-off grade (constrained by the $1500/oz pit shell).

Table 14.3.15 Lafigué Mineral Resource as of July 31, 2020 Cut-off Grade Sensitivity

Cut-off Grade (Au g/t)	Tonnes (kt)	Average Grade (Au g/t)	Au (koz)
0.1	32,871	2.35	2,484
0.3	32,788	2.36	2,488
0.5	32,030	2.40	2,471
0.7	29,783	2.54	2,432
0.9	26,464	2.75	2,340
1.1	23,141	3.01	2,239
1.3	20,258	3.27	2,130
1.5	17,915	3.51	2,022
1.7	15,829	3.76	1,914
1.9	14,090	4.01	1,817
2.1	12,585	4.25	1,720
2.3	11,288	4.49	1,630
2.5	10,166	4.72	1,543

14.4 Previous NI 43-101 Mineral Resource Estimates

Endeavour previously disclosed NI 43-101 MREs for the Lafigué deposit on the Fetekro Property in August 2017, October 2018 and July 2019, as summarised in the table below. The three previous MREs followed CIM Standards, were constrained by a respective MII $1,500/oz Au pit shell and a cut-off grade of 0.5 Au g/t. They were not accompanied by Technical Reports as Lafigué was not considered to be a material property or represented a significant change to the Endeavour Mineral Resource Statement at the time of their disclosure.

Kevin Harris CPG, Vice-President Resources, Endeavour Mining was the Qualified Person for all of the previous MREs. The previous MREs have been superseded by the MRE reported herein and are presented below to illustrate the evolution of the Project and for transparency purposes.

Table 14.4.1 Previous Lafigué NI 43-101 MREs

Date	Indicated			Inferred		
	Tonnes (kt)	Grade (Au g/t)	Au (koz)	Tonnes (kt)	Grade (Au g/t)	Au (koz)
August 2017	4,981	2.34	375	898	2.19	63
October 2018	6,833	2.25	494	3,039	2.25	225
July 2019	14,577	2.54	1,190	867	2.17	60

14.5 Risks and Opportunities

Kevin Harris CPG, Vice-President Resources Endeavour Mining as the Qualified Person confirms that the Mineral Resources as of July 31, 2020 for the Lafigué deposit on the Fetekro Property are reported in accordance with the guidelines and terminology provided in the CIM Standards and Definitions.

The Mineral Resource has been established with due recognition of all the necessary multi-disciplinary technical inputs required to ensure compliance with the Requirements as defined herein and are considered to be both a reasonable and unbiased estimate at the time of reporting.

Notwithstanding the above, there are number of generic and specific risks which apply, including:

- **Reasonable Prospects for Economic Extraction:** The current Mineral Resources are reported at a fixed in-situ cut-off grade within an optimised shell assuming a long-term gold price of US$1,500/oz with generalised mining costs. Accordingly, there remains a risk that in the event that changed parameters are utilised which reflect current Life of Mine Plan assumptions, then the Mineral Resource as reported herein, may change under updated assumptions. Furthermore, as no detailed sensitivity analysis is undertaken at a range of gold prices, it is not possible to assess the impact of these changed assumptions on the Mineral Resource statement as reported herein.

The key opportunities relating to the Mineral Resources reported at Lafigué are:

- **Mineral Resources which are potentially economically extractable by underground methods:** The current Mineral Resources are constrained within an optimised shell established at a long-term gold price assumption of US$1,500/oz. Presently no estimate or analysis of the underground mining potential has been completed and as such, there remains in certain instances the potential for reporting of additional Mineral Resources which meet the appropriate criteria for underground mining.

- **Exploration targets:** In the vicinity of the Lafigué deposit, Endeavour is investigating a number of exploration targets, and the potential of extending some of the mineralised zones. Accordingly, there remains some potential, pending completion of additional exploration drilling, that Mineral Resources may be established on these exploration targets.

- **Mineral Resource sensitivity:** Presently only limited sensitivity analysis has been undertaken and ultimately reported, to confirm the potential beyond the currently assumed long-term gold price. Accordingly, should gold prices rise, there remains the possibility of increasing the total quantum of Mineral Resources which are economically extractable by open-pit methods.

14.6 Interpretation, Conclusions and Recommendations

The Mineral Resource as reported herein, are considered to reflect an unbiased and reasonable estimate given the geological data gathered to date and are reported in accordance with the guidelines and terminology provided in the CIM Standards and Definitions. This aside, there are a number of areas of potential improvement which if completed, would increase the confidence and transparency relating to the current declarations.

The MRE has been compiled by Endeavour's resource modelling and exploration team.

The principal recommendations pertaining to the reporting of the Mineral Resource at the Lafigué deposit are:

- Engage an independent consultant to validate the model and update the next MRE; and,
- Include the next phase of drilling results, which are scheduled to be available in early 2021.

15.0 MINERAL RESERVE ESTIMATE

15.1 Mining and Mineral Reserve Estimation Approach

The Mineral Reserve statement for the Lafigué 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:

- Lafigué open pit.

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 Resource model was depleted above topography and provided the initial surface for the pit optimisations.

15.2.3 Dilution and Mining Recovery

The Lafigué 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.

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	17.9	17.9
Saprolite	17.9	17.9
Saprock	23.4	23.4
Fresh	38.8	49.2

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	3.5	96.3
Saprolite	3.5	96.3
Saprock	3.5	94.6
Fresh	3.0	94.6

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.

Table 15.2.3 Pit Optimisation Mining Costs by Weathering

Weathering	Above Lafigué Reference Level ($/t rock)	Lafigué Reference Level (mRL)	Below Lafigué Reference Level ($/t rock)
Laterite	2.70 – 0.0022 * level	380	2.70 + 0.0022 * level
Saprolite	2.70 – 0.0022 * level	370	2.70 + 0.0022 * level
Saprock	3.03 – 0.0022 * level	310	3.03 + 0.0022 * level
Fresh	3.03 – 0.0022 * level	210	3.03 + 0.0022 * level

15.2.7 Ore Costs

Table 15.2.4 summarises the ore processing costs applied in the pit optimisation.

Table 15.2.4 Pit Optimisation Ore Processing Costs by Weathering

Weathering	Laterite	Saprolite	Saprock	Fresh
Fixed cost	3.28	3.28	3.48	3.83
Variable cost	6.09	5.95	7.40	8.90
Process G&A	5.74	5.74	5.74	5.74
Total ore cost	**15.11**	**14.97**	**16.62**	**18.47**

15.2.8 Other Costs

A royalty of 4% and stamp duty of 0.5% was applied to all sales based on the price.

A transport and refining charge of $4/oz was also applied.

15.2.9 Price and discounting

A 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.

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	Cut-off (g/t)
Laterite	0.34
Saprolite	0.34
Saprock	0.38
Fresh	0.43

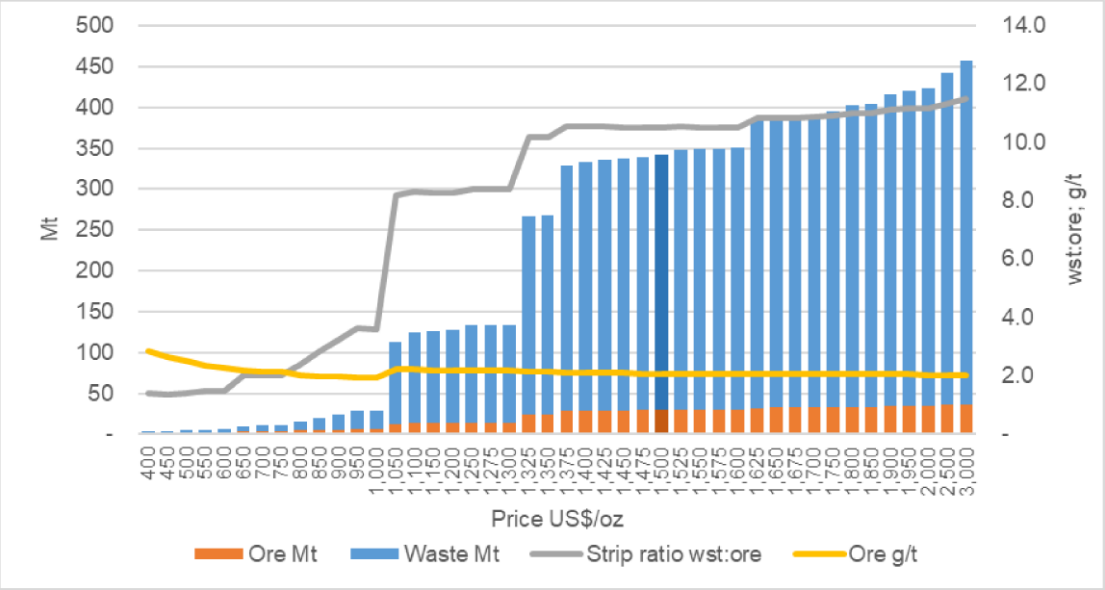
15.3 Pit Optimisation

Pit optimisations were run separately on both deposits using the standard Lerch-Grossman (LG) algorithm to produce incremental pit shells (nested) based on varying the input price. Table 15.3.1 and Figure 15.4.1 summarises the pit optimisation physicals by input price for Lafigué. There is a noticeable drop in mass around US$1,375/oz due to the surface topography and the pit progressing past a hill.

Table 15.3.1 Lafigué Pit Optimisation Summary

Pit Shell	Units	21	23	28	44
Revenue factor		0.88	0.92	1	2
Apparent gold price	US$/oz	1325	1375	1,500	3,000
Pit bottom	mRL	30	15	-5	-45
Physicals					
Ore	Mt	23.9	28.5	30.9	38.9
Waste	Mt	243.4	300.8	319.1	450.7
Total	Mt	267.3	329.3	349.9	489.6
Strip ratio	waste:ore	10.2	12.6	10.3	11.6
Diluted gold grade	g/t	2.1	2.1	2.1	2
In-situ gold	Moz	1.645	1.925	2.055	2.496
Recovered gold	Moz	1.56	1.823	1.946	2.363
Metallurgical recovery	%	94.8	94.7	94.7	94.7
Economics					
Mining cost	US$M	889.2	1100	1171.3	1,661
Mining cost	US$/t	3.33	3.34	3.35	3.39
Ore cost	US$M	434.3	519.4	562.2	710
Ore cost	US$/t ore	18.19	18.24	18.22	18.27
Selling costs	US$M	111.6	130.3	139.1	168.9
Selling costs	US$/recovered oz	71	71	71	71
Total cost	US$/in situ oz	1435.1	1749.7	911	1017
Revenue at US$1,500/oz	US$M	2340	2733.8	2,919	3,544
Undiscounted cash flow	US$M	905	984.1	1046	1004.4
Indicative present value	US$M	732	768.6	802.5	726.5

Figure 15.3.1 Lafigué Pit Optimisation Physicals



Sensitivities were also run on input parameters, varying them by ±20%, to assess the impact on the pits. When considering the impact on cash flow, gold price (which is also equivalent to metallurgical recovery) is the only variable tested that resulted in a greater percentage change (i.e. price results in about a 50% cash flow change whereas mining costs, ore costs, selling costs, slope angles and dilution result in changes of about 20% or less). Downsides are generally more than upsides due to a lack of additional resources.

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	Hanging Wall	Footwall	Hanging Wall	Footwall	Hanging Wall
Batter angle (°)	55	55	65	65	70	75
Batter height (vertical m)	10	10	10	10	10	10
Batter berm interval (vertical m)	10	10	10	10	10	10
Berm width (m)	10	10	10	10	10	10
Inter-ramp slope (toe to toe, no ramp) (°)	30.5	30.5	34.3	34.3	49.2	52.5
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 shows the ultimate pit designs for Lafigué. The Lafigué resource is predominately in fresh rock but also has a significant proportion in the saprolite rock and saprock.

Figure 15.4.1 Lafigué Ultimate Pit Design

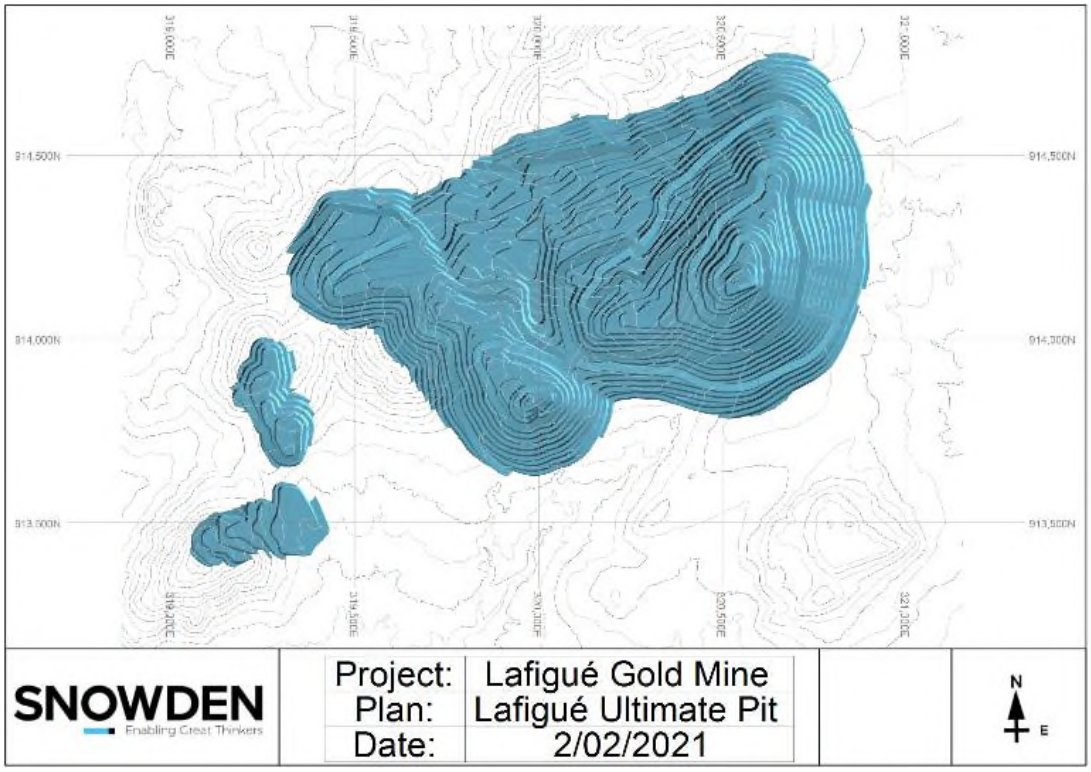
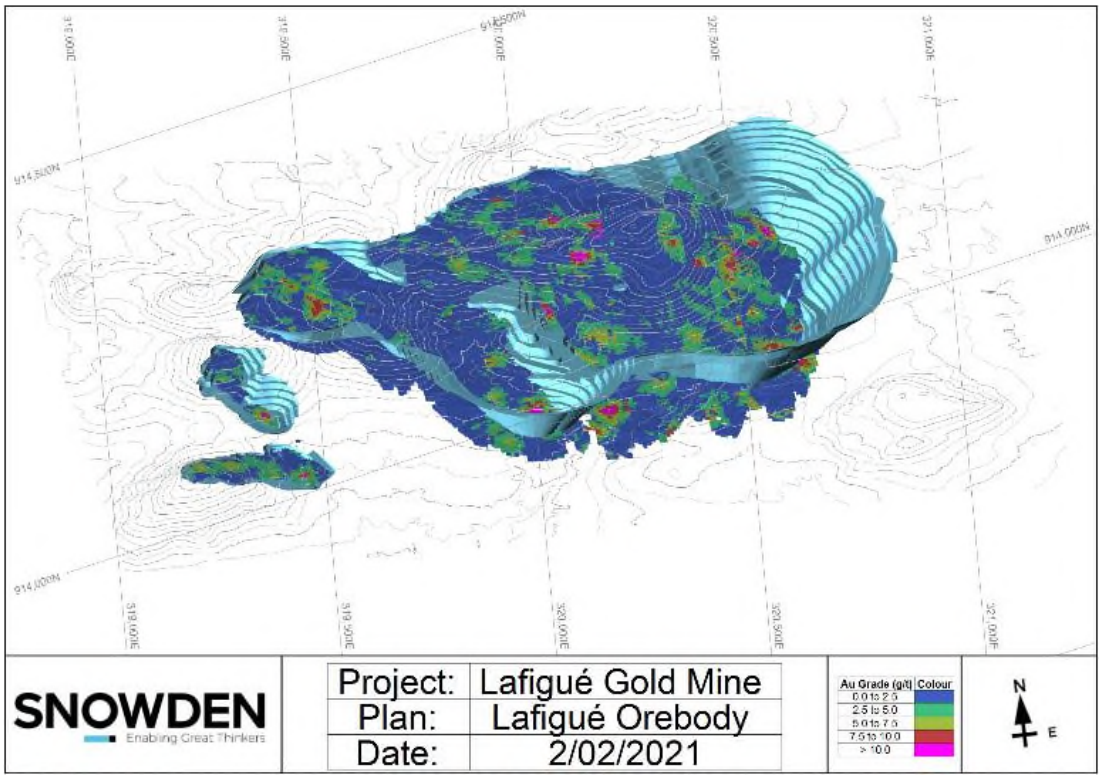


Figure 15.4.2 shows the Lafigué orebody against the ultimate pits. The orebody is coloured by gold grade and exclude the Inferred Resources.

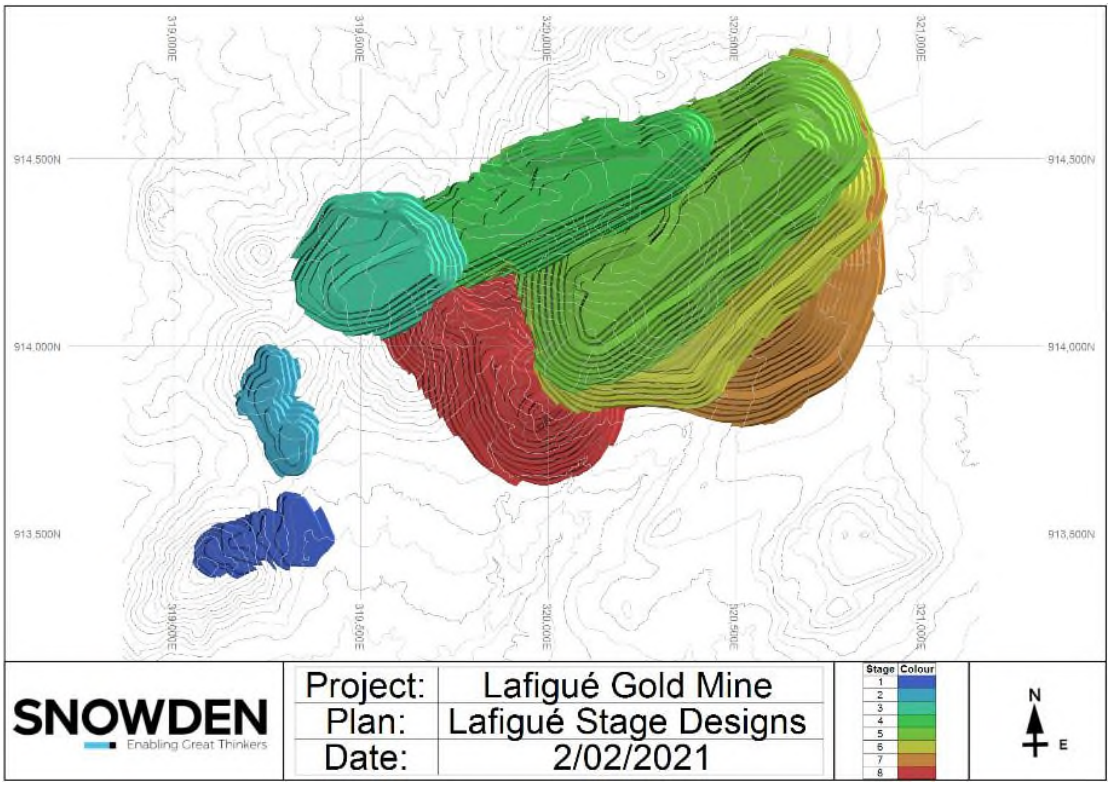
Figure 15.4.2 Lafigué Orebody



15.4.3 Stage Pit Designs

Figure 15.4.3 shows the stage designs for Lafigué that comprise eight stages with the earlier stages focused on accessing higher value ore. Two stages are satellite pits to the west of the main pit.

Figure 15.4.3 Lafigué Stage Pit Designs



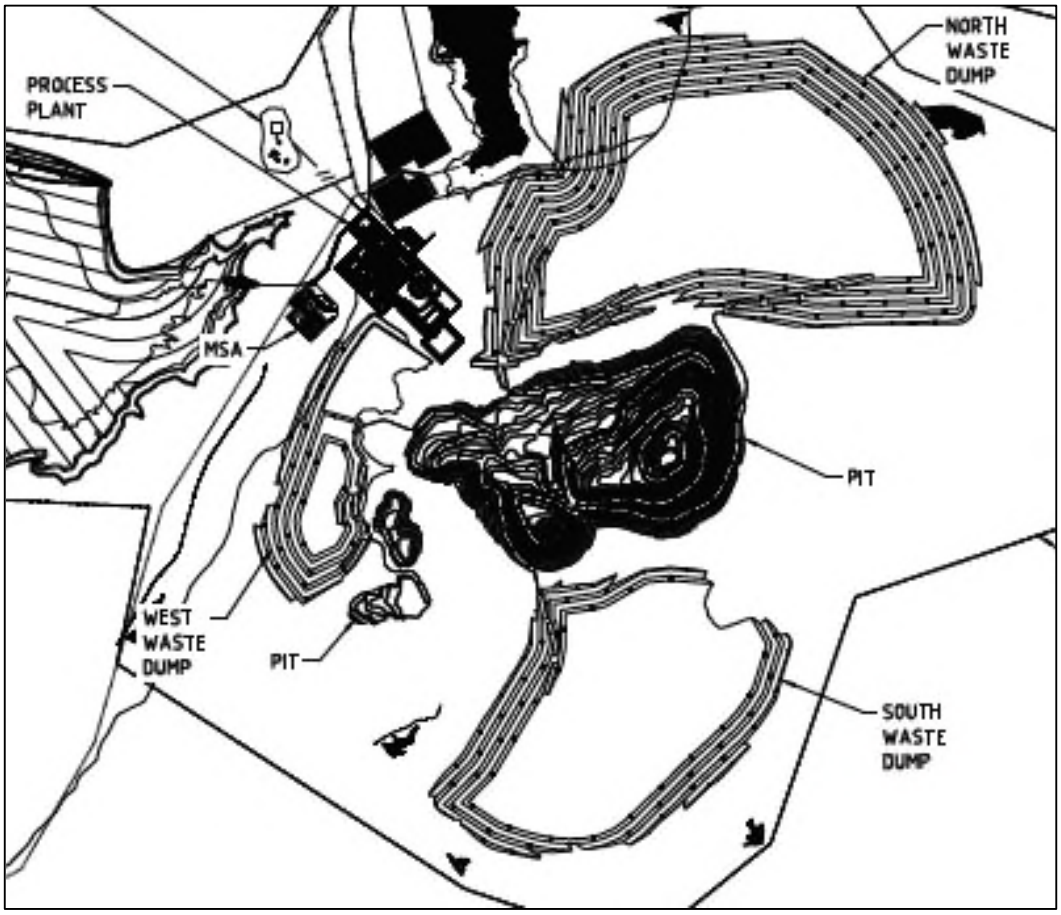
15.4.4 Mining Infrastructure Layout

The detailed overall project site layout is included in Section 1 and Section 18 of this report.

Figure 15.4.4 shows a section of the overall layout, highlighting the key mining features including:

- Primary waste dumps to the north and south of the open pit.
- ROM and long term ore stockpiles are located centrally to the deposit.
- Mine infrastructure is located close to the open pit and processing plant.

Figure 15.4.4 Lafigué Stage Pit Designs



15.5 Mining Quantities and Mineral Reserve Estimate

The Lafigué open pits contain 31.9 Mt at 2.0 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 Lafigué Mineral Reserve Estimate (December 2020)[1]

Item	Proved	Probable	Total Mineral Reserve
Tonnes (Mdt)	0	31.9	31.9
Gold grade (g/t)	0	2.0	2.0
Contained gold (Moz)	0	2.05	2.05

[1] ***Some numbers may not sum correctly due to rounding Risks and Opportunities***

The Mineral Reserve statement for the Lafigué 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, selling costs, slope angles and dilution) impact the undiscounted cash flow and pit size. However, a 20% change in any of these would change the Reserve ounces by around 10%.
- Inferred ore of 0.5 Mt inside pit design could be realised, producing an additional 32 koz.

Additional stage designs and scheduling could delay waste stripping, reducing the upfront strip ratio.

16.0 MINING METHODS

16.1 Mining Methods

Lafigué Gold Project is intended to be conventional open pit mining using a drill, blast, load, haul and tip mining cycle. Endeavour plan to contract out the mining to a suitable mining contractor whilst maintaining operation oversight. Given the high grades, this does not preclude future underground mining if the orebody extends at depth.

The saprolite and laterite is anticipated to be primarily free-dig, potentially requiring ripping. Production drilling of saprock and fresh material 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 0.36 kg/bcm. All the fresh rock will be blasted with powder factors estimated at 0.83 kg/bcm.

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.
  - In-pit backfill may be potentially available but has not been incorporated into the mining schedule.

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.
- Firefighting.

16.2 Hydrogeology

EMSA carried out field visit and investigations to infer the characteristics of the aquifers that will need to be dewatered during development and operation. The initial conceptual model and recommendations from the site investigation are:

- Two distinct groundwater zones exist:
  - A confined fractured aquifer, covering almost two-thirds of the pit shell (from the northern catchment divide to the south). The water table is typically below 320 m RL. This aquifer has isolated perched zones and a distinct deep zone with water level under 280 mRL.
  - A semi-confined aquifer in the upper north, characterised by water table above 320 mRL, with a peak at 370 mRL.
- Water bearing fractures typically occur between 70 and 110 mbs. The shallow saprolites are not water-bearing due to the nominal thickness (<10 m). Any recharge to these saprolites flows laterally to the shallow streams, with minor vertical flow to the underlying saprock and later to the deeper bedrock where the water is added to the confined fractured storage.
- Groundwater occurrence is associated with open fractures in mafic intrusive and mafic volcanic. Where no mafic volcanic exists, the mafic intrusive typically have blown yields less than 5 m^3/h. The rock contacts between mafic intrusive and mafic volcanic yields water in excess of 10 m^3/h. The mafic volcanic are recommended to be targeted for any further groundwater exploration project.
- For a water supply borehole to be considered sustainable, the abstraction rate should not lead to dewatering of the main water strike during the life of the project. If FTBH03 is to be used for water supply, it can be sustainably be pumped at 7.9 m^3/h (2.2 L/s) with an available drawdown of 85 m. The sustainable abstraction rate for FTBH02 is evaluated at 2.9 m^3/h (0.8 L/s), with an available drawdown of 50 m.

- With a project water supply requirement of 42 L/s from pit dewatering and boreholes, if 30 L/s were to be secured from boreholes, 15 successful boreholes with sustainable yields of at least 2 L/s are required.

- Due to drawdown interference, the boreholes in the wellfield would have to be between 50 to 150 m apart, depending on the prevailing aquifer properties.

16.3 Geotechnical Evaluation

Based on lithological and geotechnical models Lafigué was divided into six design sectors. Bench and Inter ramp design parameters were done for selected cuts that represent the most critical sections across the area. Specific bench and IRA design parameters to be used to create slope cuts for the most critical sections, which were then numerically modelled to determine the over-all slope stability using finite element and limit equilibrium analyses. Both numerical modelling methods indicate that the OSA factor of safety is above the recommended design criteria.

Based on the results of the study the recommended design parameters based on the weathering profile within the pit were recommended with 15 m wide geotechnical berms between weathering zones, and in fresh material maximum stack heights of 80 m separated by geotechnical berms.

Figure 16.3.1 and Table 16.3.1 summarises the slope angles used in the PFS based on the pit design parameters provided by Golder and include reductions to account for ramps and slope estimation errors in Whittle. The steeper wall angles reflect a lower reduction for ramps and steeper OSAs in the hangingwall.

Figure 16.3.1 Lafigué Geotechnical Design Parameters

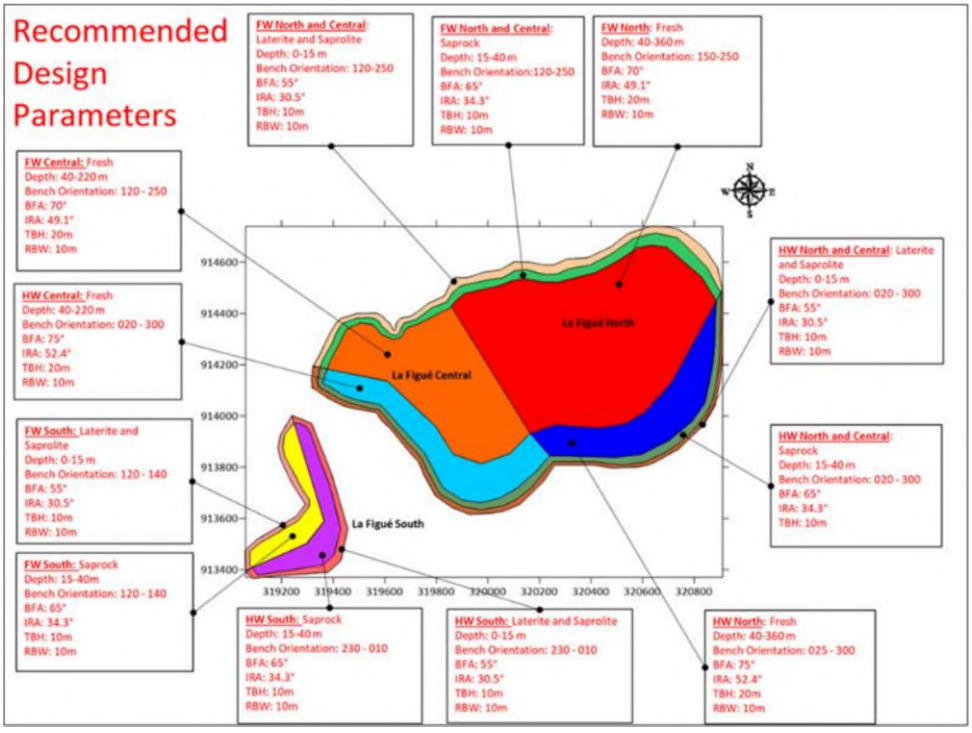


Table 16.3.1 Pit Optimisation Slope Angles Applied by Weathering

Weathering Type	Hangingwall (°)	Footwall (°)
Laterite / Saprolite	17.9	17.9
Saprock	23.4	23.4
Fresh	49.2	38.8

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 <1.0
Medium Grade	MG	Reserve	≥1.0 and <2.0
High Grade	HG	Reserve	≥2.0

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 six 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.

Active Mining Areas

The number of active stages that can be mined in any period was not constrained.

Bench Turnover

Bench turnover was varied depending on the strip ratio. For the waste portion where the strip ratio was above 10 it was restricted to 100 vertical metres per year, 80 vertical metres where the strip ratio was above one, and 60 vertical metres below one, while the ore portion was restricted to 40 vertical metres per year. Benches in the lower size quartile which were weighted proportionally. This resulted in higher rates at the top of the deposit and lower at the bottom (i.e. benches with tighter working areas and requiring more selectivity were mined at a slower rate). The average bench turnover at about 77 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 (66%, 98%, 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

Table 16.4.3 Loading Units Available

	Years 1 to 7	Year 7 Onwards
200 t excavator (#)	4	2
100 t excavator (#)	3	2
Capacity range based on productivity (Mt/a)	51	28

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	3.5
Saprolite	3.5
Saprock	3.5
Fresh	3.0

Product

The minimum amount of gold produced in the final schedule was constrained to produce a more consistent production profile. Production of at or above 200 koz/a for the first eight years.

16.4.2 Mining schedule

Figure 16.4.1 shows the annual movements from the pits by stage. Mining peaks at around 50 Mdt/a with the variation largely due to the strip ratio and surface topography. (Figure 16.4.1). 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

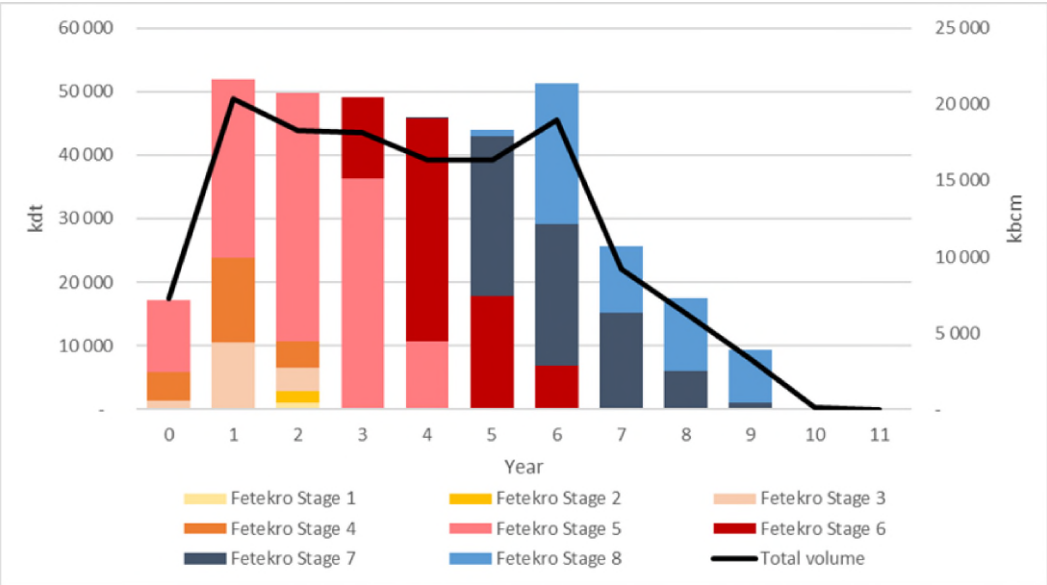


Figure 16.4.2 Total Ex-Pit Movement by Weathering

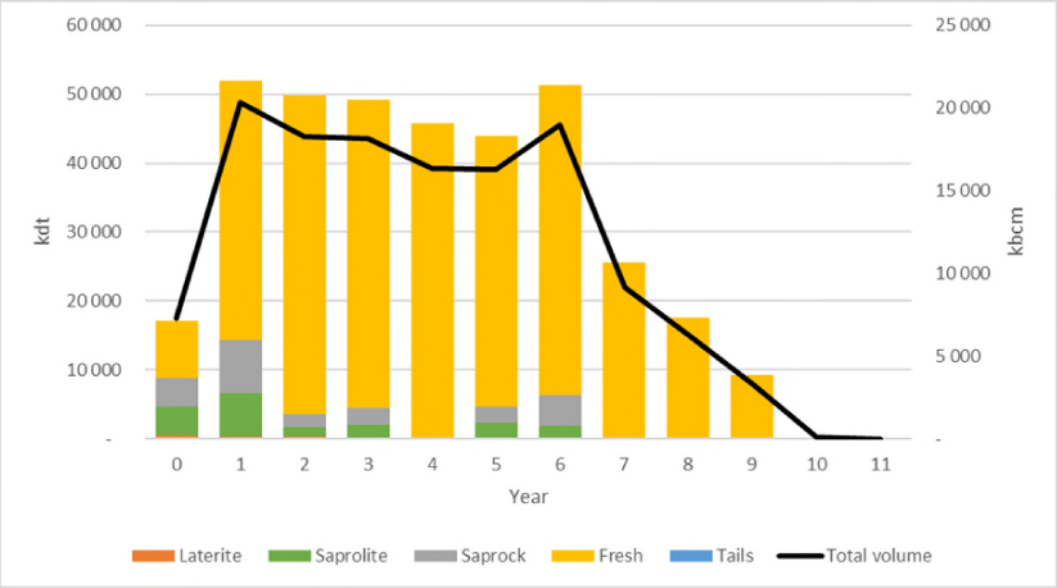


Figure 16.4.3 shows the annual movements from the pits by material type. Ore is mined consistently over the life of mine in quantities above the required processing rate. The lower grade ore is placed on long-term stockpiles (Figure 16.4.4) allowing the higher grades to be fed to the processing plant 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

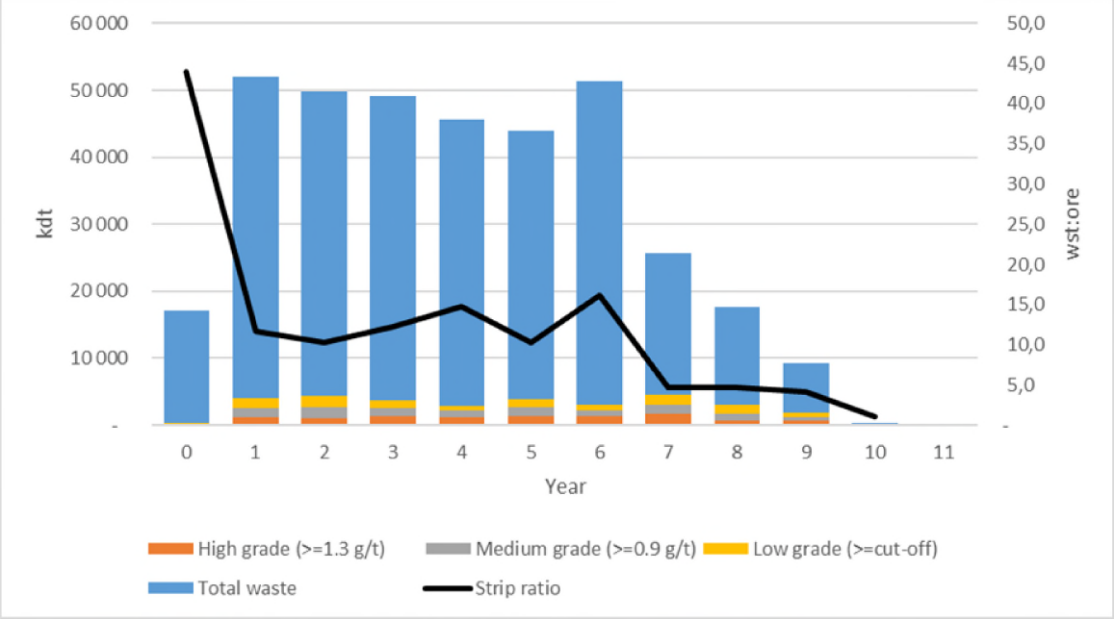
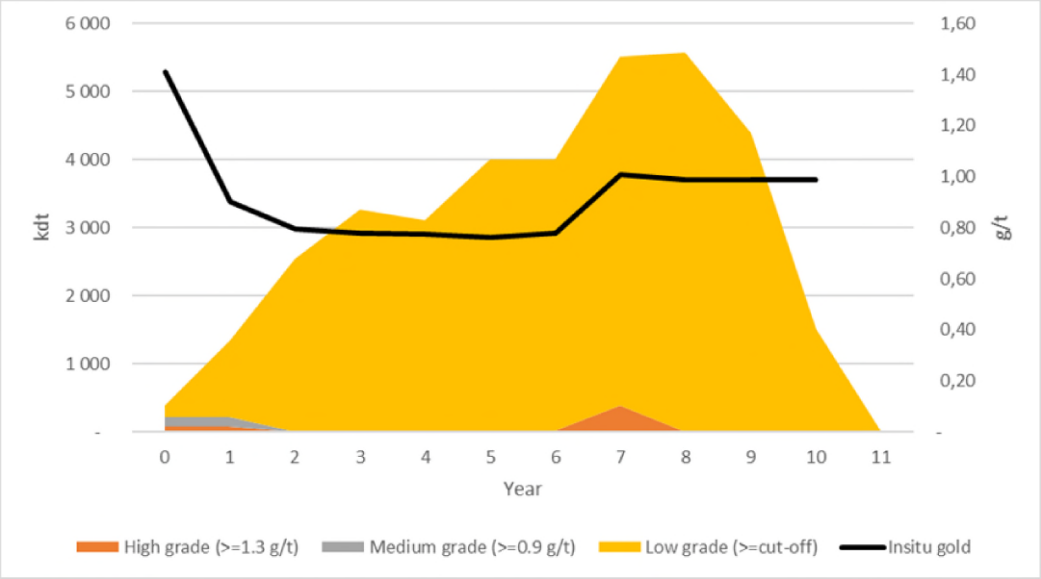


Figure 16.4.4 shows the amount and grade of the ore on the long-term stockpile. Low grade is the main ore type stockpiled as the higher grades are sent to the ROM and fed to the plant. The long-term stockpile is primarily rehandled at the end of mining (Year 8).

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 the initial years 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

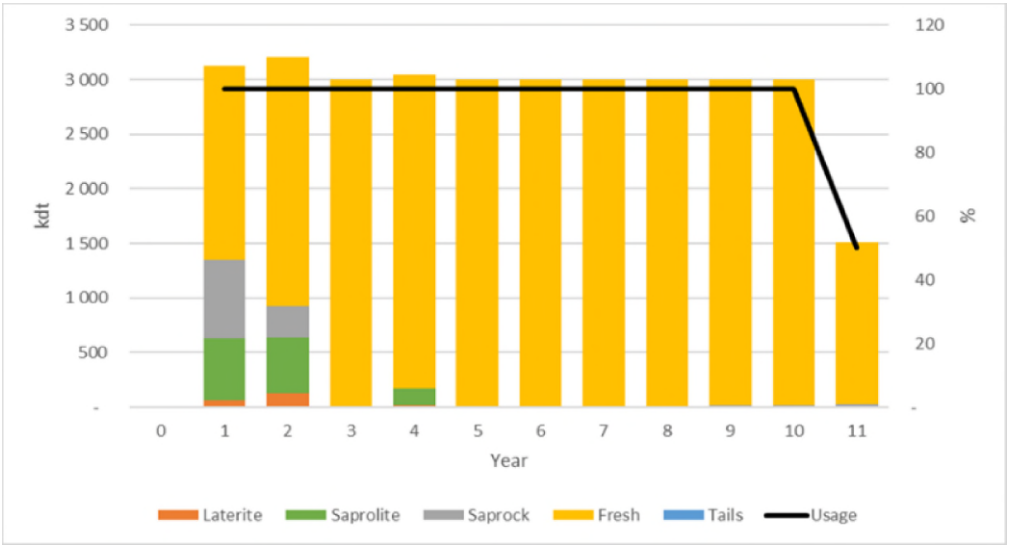


Figure 16.4.6 shows the ore feed to the processing plant by type and the feed grade. After an initial high grade peak, the feed grade is predominately around 2.4 g/t except when feeding from the long-term stockpiles.

Figure 16.4.6 Ore Feed by Material Type

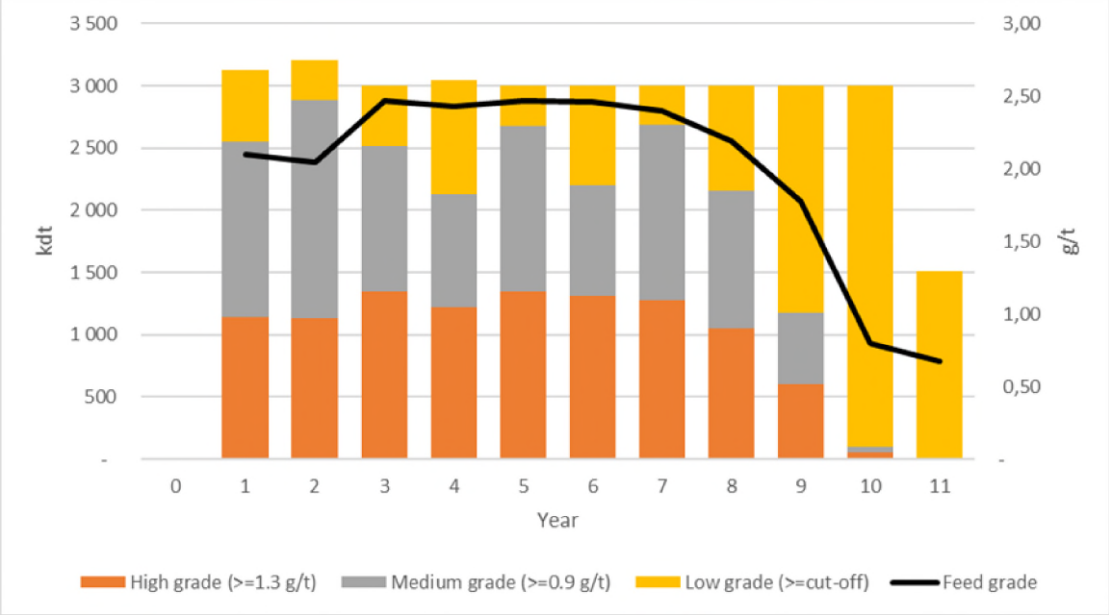
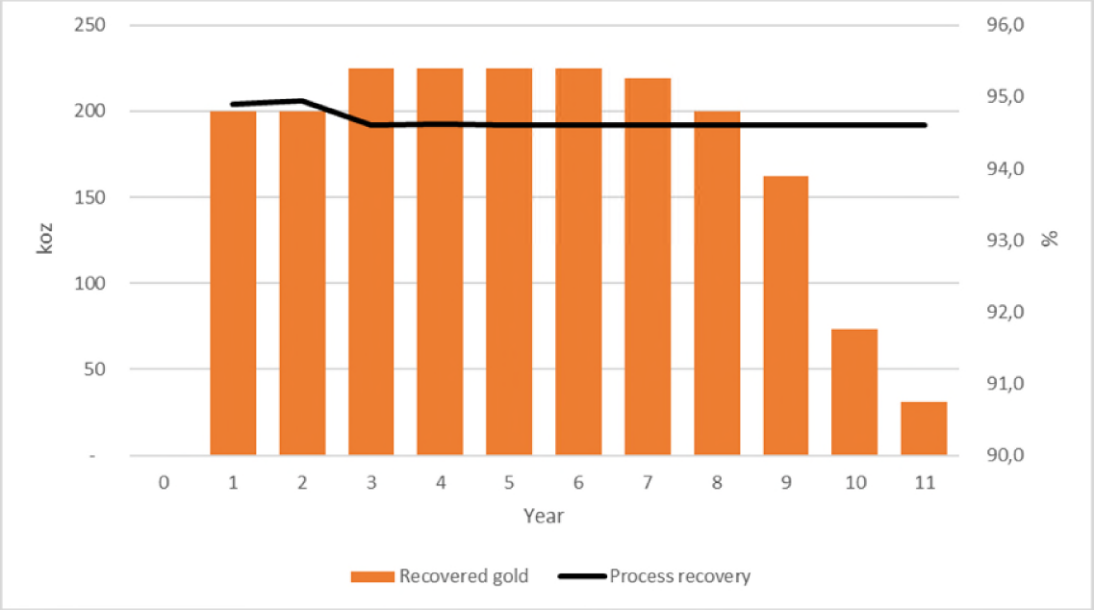


Figure 16.4.7 shows the metallurgical recovery and gold produced. Gold production is at or above 200 koz/a for the first eight years.

Figure 16.4.7 Gold Production and Metallurgical Recovery



16.5 Fleet Size and Personnel Numbers

Table 16.5.1 summarises the proposed 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 Anticipated 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 two shift per day roster.

Table 16.5.2 Mining Personnel

Role	Peak Full-Time Employees
Management	3
Production supervision	13
Geology & labourer	40
Technical services	41
Training and safety	11
Production operators, contract	300
Maintenance	111

17.0 RECOVERY METHODS

17.1 Process Selection

17.1.1 Design Philosophy

The process plant design for the Lafigué Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimal operating costs. The flowsheet is based on unit operations that are well proven in industry.

The Lafigué plant will process fresh and oxide ores and is expected to operate on either 100% fresh ore (possibly with a small portion of oxide) or 100% oxide ore. The fresh and oxide ores have different comminution and material handling characteristics with the fresh ore significantly more competent than the oxide ore. The ore types require different comminution flowsheets, specifically:

Fresh ore – closed circuit secondary crushing with crushed ore storage, closed circuit High Pressure Grinding Rolls (HPGR) crushing with HPGR crushed ore storage and ball milling
Oxide ore – open circuit primary crushing with direct feeding of the ball mill.
The key project and ore specific design criteria that the plant design must meet are as follows:

- 3,000,000 t/y of fresh ore.
- When processing fresh ore, mechanical availabilities of 70% for the closed circuit secondary crushing plant, 86.7% for the closed circuit HPGR crushing plant 91.3% for the remainder of the plant, supported by crushed ore storage and standby equipment in critical areas.
- When processing oxide ore, mechanical availability of 88.0% with direct feed of primary crushed ore to the ball mill.
- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

Several value engineering assessments (VEA) and a process flowsheet selection memo were completed to assess key process options for the Lafigué flowsheet.

A process design criteria document has been prepared incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical testwork and comminution circuit modelling. The design document forms the basis for the design of the processing plant and required site services.

17.1.2 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Primary jaw crushing to produce a coarse crushed product.

Fresh Ore

- Secondary cone crushing in closed circuit with a dry sizing screen to produce an intermediate crushed product
- A live secondary crushed ore stockpile, providing coarse crushed ore storage and reclaim to feed the HPGR crushing circuit
- Tertiary HPGR crushing in closed circuit with a wet sizing screen with undersize slurry reporting to the milling circuit via the mill discharge hopper.

Oxide Ore

- Direct feeding of primary crushed ore to the ball mill feed chute.

Oxide and Fresh Ores

- A ball mill in closed circuit with hydrocyclones to produce a grind size P_{80} of 75 µm (micron).
- 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.
- Trash screening to remove any wood trash or oversize material prior to cyanidation.
- Pre-leach thickening of the trash screen underflow to produce a higher solids concentration leach feed to reduce leach and adsorption tankage and reagent requirements.
- A leaching circuit to leach gold from the milled ore with one pre-leach oxidation tank and five leach tanks providing a total of 22 hours (oxide ore) to 36 hours (fresh ore) residence time.
- A CIP circuit to adsorb leach solution gold onto activated carbon in seven tanks operating in carousel.
- A split AARL elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré.
- Tails wash thickening of the CIP tails to produce a higher solids concentration tails and to recover process water and cyanide from the tails slurry.

- Dilution of the tails thickener underflow with decant return / raw water in order to meet the target plant tails cyanide discharge level.
- Tailings pumping to the tailings storage facility (TSF).

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet is shown in Figure 17.1.1.

17.1.3 Plant Design Basis

The key issues considered in the process and equipment selection are outlined in this section.

Process Plant

The plant design is based on a nominal capacity of 3.0 Mtpa of fresh ore and is expected to process 3.5 Mtpa of oxide ore. The plant will process either 100% fresh ore (possibly with a small portion of oxide) or 100% oxide ore. The plant feed schedule indicates that the life of mine (LOM) plant feed is 8% oxide / transition ore and 92% % fresh ore, with the majority of oxide ore processed in the first two years of operation.

ROM Pad

The ROM pad will be used to provide a buffer between the mine and the plant. The ROM stockpiles will allow blending of feed stocks and ensure a consistent feed ore type, rate and grade to the plant.

The ROM bin will be designed to accommodate both direct tipping from mine trucks and blended feed addition by FEL. A mobile rock breaker will be used to break any oversize rocks on the ROM pad.

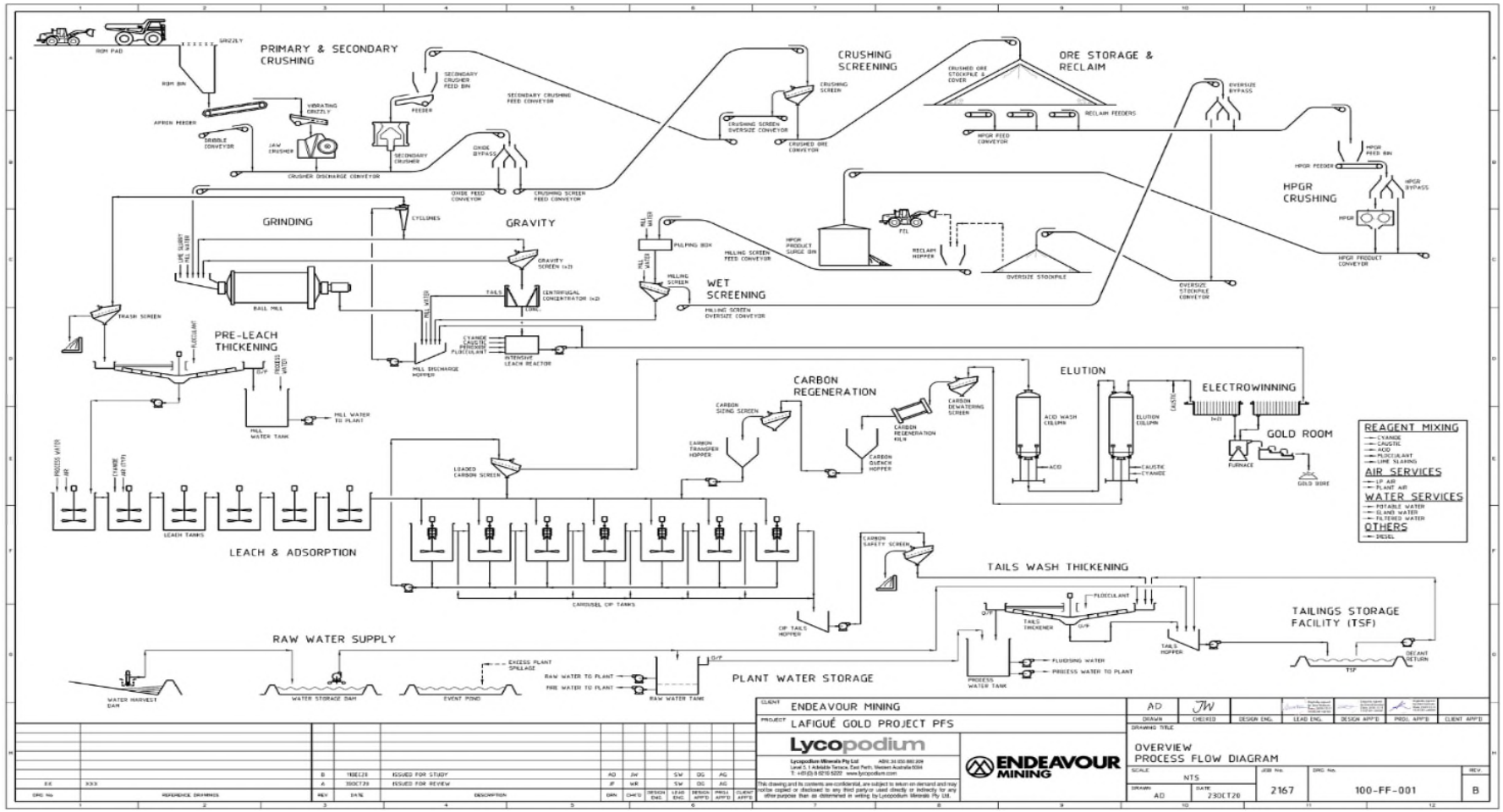
Comminution Circuit Selection

A range of comminution circuits were reviewed after the Fetekro Scoping Study (SS) was completed. The comminution circuits reviewed were based on processing the SS throughput of 1.5 Mtpa of ore and included the following options:

- Primary crush, SABC (SAG and Ball milling and pebble crushing) – as per the Scoping Study.
- Secondary crush with tertiary HPGR crushing, ball milling.
- Secondary crush with tertiary cone crushing, ball milling.
- Primary crush, SS SAG (Single stage SAG milling).

After review of the report, Endeavour selected the HPGR circuit.

Figure 17.1.1 Lafigué Process Plant Flowsheet



As part of the PFS, a value engineering assessment (VEA) comparing the Scoping Study SABC circuit and the preferred HPGR circuit was completed based on processing the PFS throughput of 3.0 Mtpa. The VEA indicated that the HPGR circuit was the more economical circuit for the Lafigué Project due to the lower power and grinding media requirements.

Endeavour selected the secondary crush with HPGR crushing and ball milling circuit for the PFS plant design. The circuit will achieve the target P_{80} grind size of 75 µm when processing fresh ore.

Comminution data was provided to Orway Mineral Consultants (OMC) for comminution circuit modelling and equipment sizing. No HPGR testwork has been completed on the Lafigué ores at this stage due to the considerable sample mass required.

The existing comminution data was benchmarked against other ores where HPGR testwork results were available to estimate characteristic properties and select the HPGR unit. OMC also provided benchmarked comminution data for the Lafigué oxide ore.

The OMC modelling was originally completed using their typical design point of the 85th percentile results of the comminution data and the benchmarked HPGR data. Endeavour then selected the more conservative approach of designing the circuit based on the 100th percentile comminution data (i.e. hardest ore) in order to ensure that the design plant throughput could be achieved for all ores. The comminution circuit design basis and selected comminution circuit is detailed in an OMC report.

The key design criteria used in modelling of the comminution circuit and the key results for the selected circuit are summarised in Table 17.1.1 and Table 17.1.2.

The comminution average power and consumables requirements are summarised in Table 17.1.3 and are used in the estimation of the processing operating costs.

Table 17.1.1 Comminution Circuit Design Parameters

Parameters		Units	Fresh Ore	Oxide Ore*
Testwork Design				
CWi	Calc from Axb	kWh/t	28.2	3.2
RWi	100th %ile	kWh/t	20.7	8.0
BWi	100th %ile	kWh/t	19.6	6.8
Ai	100th %ile / Average	g	0.251 / 0.144	0.015 / 0.015
Mic	100th %ile	kWh/t	0.251 / 0.144	0.015 / 0.015
Axb	0th %ile		25.9	
SG	Average	t/m^3	2.83	2.70
HPGR Specific Pressure*		N/mm^2	3.5	
HPGR Specific Energy*	Per Pass	kWh/t	1.75	
HPGR Specific Throughput (m dot)*		ts/hm^3	284	
Comminution Circuit Design				
Plant Throughput		Mt/y	3.0	3.5
Crushing Feed F_{100}		mm	800	500
Primary & Secondary Crushing	Availability	%	70%	
	Throughput	dry t/h	489	
HPGR (Tertiary) Crushing	Availability	%	86.7%	
	Throughput	dry t/h	395	
Primary Crushing	Availability	%		88%
	Throughput	dry t/h		454
Grinding Circuit	Availability	%	91.3%	88%
	Throughput	dry t/h	375	454
Milling Feed F_{80}		mm	7.2	100
Milling Product P_{80}		µm	75	75

Benchmarked data only

Table 17.1.2 Summary of Selected Comminution Circuit

Equipment	Unit	Fresh Ore	Oxide Ore
		HPGR Crush / Ball Milling	Primary Crush / Ball Milling
Primary Crushing			
Type		Jaw Crusher	
Model		Metso C150 or Equiv.	
Closed Side Setting	mm	140	
Secondary Crushing			
Type		Cone Crusher	-
Model		Metso HP900 or Equiv.	-
Closed Side Setting	mm	35	-
Machine Load	%	71	-
Crushing Screen			
Type		Double Deck Dry Multi-slope	-
Size (Width x Length)	m x m	2.4 x 6.1	-
Top Deck Aperture	mm	65	-
Bottom Deck Aperture	mm	35	-
Crushing Product Size P_{80}	mm	25	101
HPGR			
Size (Diameter x Width)	m x m	1.59 x 1.45	-
Net Throughput	t/h	395	
Total Throughput	t/h	819	-
Speed	m/s	1.25	-
Power - Duty	kW	1,433	-
Power - Installed	kW	2 x 1,100	-
Milling Screen			
Type		Double Deck Wet Multi-slope	-
Size (width x length)	m x m	3.6 x 6.1	-
Top Deck Aperture	mm	8	-
Bottom Deck Aperture	mm	4	-
Milling Screen Undersize P_{80}	mm	2.2	-
Ball Mill			
Mill Diameter x length (EGL)	m x m	6.40 x 10.96	
Discharge Arrangement		Overflow	
Mill Speed Range	% Nc	60 - 80	
Mill Speed	% Nc	75	60
Top Ball Size	mm	65	65
Ball Charge - Duty	% Vol	27	16
Pinion Power - Duty	kW	6,532	3,522
Installed Power	kW	7,900	

Table 17.1.3 Comminution Consumables by Ore Type

Equipment / Parameter	Unit	Fresh 3.0 Mtpa	Oxide 3.5 Mtpa
Abrasion Index – Average		0.144	0.015
Primary Crusher			
Liner Consumption – Fixed	hours	2,000	5,000
Liner Consumption – Swing	hours	3,333	10,000
Gross Power Consumption	kWh/t	0.3	0.1
Secondary Crusher			
Liner Consumption – Bowl	hours/unit	2,000	-
Liner Consumption – Mantle	hours/unit	2,000	-
Gross Power Consumption	kWh/t	1.0	-
HPGR			
Liner Consumption	hours/unit	8,000	-
Gross Power Consumption	kWh/t	3.7	-
Ball Mill			
Media Consumption	kg/t milled	0.660	0.032
Liner Consumption	kg/t milled	0.089	0.006
Gross Power Consumption	kWh/t milled	15.0	8.4

Circuit Availabilities

The open circuit primary crushing and closed circuit secondary crushing and screening equipment is sized based on 70% availability (489 dry t/h) on fresh ore. This was driven by the lower availability of the secondary crusher due to the expected higher maintenance requirements.

The crushing circuit is decoupled from the downstream plant by the secondary crushed stockpile, which provides surge capacity between the lower availability crushing circuit and the downstream higher availability plant.

The closed circuit HPGR crushing is sized based on 86.7% availability (395 dry t/h) on fresh ore. HPGRs are capable of higher availabilities, similar to milling circuits, however the HPGR and mill availabilities differ. HPGRs required more frequent, shorter downtimes (typically 2 – 4 hours per week) for preventative maintenance requirements while mills require less frequent but longer downtime. Therefore the selected HPGR availability is lower than that of the milling circuit so that the HPGR circuit can crush ore at a slightly higher rate than the milling throughput in order to build-up sufficient surge capacity to maintain the milling circuit feed when the HPGR is off-line.

The HPGR is decoupled from the milling circuit by the HPGR product surge bin which provides surge capacity between the HPGR and the downstream plant.

The milling circuit and downstream plant is sized on 91.3% availability (375 dry t/h) on fresh ore.

A primary crushing, milling and downstream plant availability of 88% (454 dry t/h) was selected for when the plant is processing oxide ore. With the potential for ore handling problems due to wet and sticky oxide ore, the ball mill will be direct fed with primary crushed ore, with no surge capacity, resulting in the overall lower plant availability.

Primary Crushing, Secondary Crushing and Dry Screening

Closed circuit secondary crushing is required to achieve the fresh ore particle size suitable for downstream HPGR crushing. HPGR are sensitive to feed size and it is important that the top feed size is less than the HPGR operating gap. The feed size to the HPGR impacts on the HPGR tyre wear life with the wear life decreasing as the top size increases. A secondary closing screen aperture of 35 mm was selected for the secondary crushing circuit in order to maximise HPGR tyre wear life.

The primary jaw crusher, secondary cone crusher and dry crushing screen were sized by OMC based on the expected crushing circuit throughput rates, crushing and screening efficiencies and recirculating load.

An apron feeder was selected to draw material from the ROM bin being suited to both clayey oxides and harder primary ore. The apron feeder will discharge onto a vibrating grizzly which will allow crusher product sized ore to bypass the jaw crusher, reducing the load and wear on the jaw crusher. When treating clayey or wet ores, the vibrating grizzly may bridge with all ore reporting to the jaw crusher. Consequently, the jaw crusher is sized to handle the full oxide throughput.

A fixed rock breaker will be located adjacent to the primary crusher.

A double deck secondary crushing closed circuit screen will be provided. This will be fitted with a coarse aperture upper deck to protect the bottom deck and reduce the material bed depth on the lower deck for higher screening efficiency.

Belt magnets and a metal detector will be located on the crushing circuit conveyor belts to prevent tramp metal from reporting to the secondary crusher or the crushed ore. The crushing circuit conveyors will be covered to prevent rain increasing the crushed ore moisture and to minimise dust emissions from the crushed ore.

On oxide ore, primary crushed material will bypass, via a diverter gate, to the oxide feed conveyor and will report directly to the ball mill feed.

Secondary Crushed Ore Storage

Secondary crushed ore will report to a live stockpile with sufficient capacity to allow for regular maintenance at the primary and secondary crushers without interrupting feed to the HPGR. The stockpile will be covered to prevent rain increasing the crushed ore moisture and to minimise dust emissions from the crushed ore.

Three feeders (as requested by Endeavour) will reclaim secondary crushed ore onto the HPGR feed conveyor as new feed to the HPGR crushing circuit. The HPGR feed conveyor will be covered to prevent rain increasing the HPGR feed moisture.

HPGR Crushing, HPGR Product Storage and Wet Screening

The HPGR and wet screen were sized by OMC based on the benchmarked HPGR design parameters, expected HPGR circuit throughput rates, screening efficiency and recirculating load.

The HPGR will operate in closed circuit with the wet milling screen with screen oversize reporting back to the HPGR feed (along with new feed) and screen undersize reporting to the milling circuit. All conveyors with material reporting to the HPGR will be covered to prevent rain increasing the HPGR feed moisture.

The HPGR product will contain considerable oversize due to the pressure profile across the roll width, with little crushing occurring near the roll edges. Wet screening allows efficient screening down to fine sizes and as the size decreases, the overall process becomes more energy efficient due to reduced ball milling requirement. A closing screen aperture of 4 mm was selected for the HPGR circuit based on OMC experience with other HPGR operations.

A belt magnet will be located on the HPGR feed conveyor to remove magnetic tramp metal ahead of the HPGR feed bin. Tramp metal not removed will be detected by a metal detector prior to the HPGR crusher. The metal detector will activate a diverter gate to allow the metal to bypass the HPGR.

Intermediate storage of the HPGR product has been allowed between the HPGR and milling screen. The HPGR product surge bin will have sufficient capacity to allow for short term stoppages of the HPGR without interrupting feed to the milling circuit. The surge bin will overflow so that a larger stockpile of HPGR product can be built up to cater for periods of longer HPGR downtime. The stockpile HPGR product will be reclaimed by FEL to the milling screen feed conveyor via the HPGR product reclaim hopper.

The HPGR product will be mixed with water in a re-pulping box to assist in de-agglomeration of the HPGR flake product to improve the subsequent wet screening process. Low cyanide milling water will be used in this area to minimise HSE and cyanide compliance issues. Wet screening will be conducted at less than 55% solids w/w in order to optimise screening efficiency. The wet double deck vibrating milling screen will be fitted with a coarser aperture upper deck to protect the bottom deck and reduce the material bed depth on the lower deck for higher screening efficiency.

The milling screen has been sized to optimise oversize dewatering in order to minimise HPGR feed moisture. The milling screen will be located adjacent to the ball mill to allow the screen undersize to gravitate directly to the mill discharge hopper.

Milling screen oversize will normally report to the HPGR feed conveyor. If the HPGR is offline, oversize can be diverted, via the oversize bypass chute and oversize conveyor, to the oversize stockpile. The oversize will be reclaimed by FEL to the milling screen feed conveyor via the HPGR product reclaim hopper.

Milling and Classification

When treating fresh ore through the HPGR circuit, the ball mill will be reverse fed via the cyclone underflow. This will remove final product size material generated by the HPGR effectively reducing the new feed rate to the mill. This HPGR advantage has been taken into account when sizing the ball mill. The milling screen undersize will contain significant water (from the wet screening stage) and minimal additional water will be required to the mill discharge hopper.

When treating oxide ore, the ball mill will be direct fed via the mill feed spout. Dilution water will be added to the mill feed and mill discharge hopper.

The ball mill will be equipped with a variable speed drive and will typically be operated between 60% and 75% of critical speed. The ball mill has been sized based on fresh ore and will typically operate at 75% critical speed when treating fresh ore. When treating oxide ore, the mill will be operated at lower speeds (and lower ball charge) to minimise overgrinding of the softer material.

Lime for pH control in the leach and adsorption circuits will be added as a slaked lime slurry to the ball mill feed.

The ball mill will discharge through a trommel screen into the mill discharge / cyclone feed hopper for normal closed circuit operation. The cyclone underflow stream will be split to feed the gravity circuit as well as recycle the underflow to the ball mill feed.

A cluster of cyclones with stand-by units has been selected for the classification duty to ensure that there are sufficient duty cyclones to allow maintenance on the units without compromising operability.

When treating fresh ore, the cyclones will be fed at lower slurry densities due to the high milling screen water addition and reverse feeding of the ball mill. In order to maintain the required cyclone underflow density to feed the ball mill, the water split to cyclone overflow is high resulting in lower cyclone overflow densities. When direct feeding oxide ore to the ball mill, high cyclone feed densities and resultant higher cyclone overflow densities can be achieved, unless the oxide slurry exhibits high viscosity.

Selection of a pre-leach thickener to increase leach feed density allows operation of the wet screening, milling and classification circuit to be optimised by allowing operation the cyclones at the required feed densities, thereby increasing classification efficiency, reducing circulating load and overall circuit power consumption.

Gravity Concentration

Gravity testwork indicated that 20 to 90% of the contained gold in the Lafigué ores is recoverable through gravity gold methods.

A gravity recovery circuit with intensive cyanidation of the concentrates allows the feed grade to the leaching circuit to be smoothed by removing spikes of high grade from the feed stream thus allowing the leach / adsorption operation to be optimised. A separate 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.

The gravity circuit will be fed from the cyclone underflow splitter box, from which a variable split to the gravity circuit is possible. Cyclone underflow to the gravity circuit provides a pre-concentrated feed and allows a greater proportion of the milling recirculating load to be processed via the gravity circuit. The option of feeding the circuit from the mill discharge hopper (cyclone feed) via dedicated gravity feed pumps reduces the water balance constraints (due to wet screening) however the benefits of pre-concentrated feed are lost.

The gravity circuit will consist of two parallel trains of gravity screens and large centrifugal concentrators feeding a single, large intensive cyanidation reactor. Parallel circuits have been selected in order to maintain gravity treatment during the batch cycling of the concentrators. This large gravity circuit can treat a high throughput, treating a significant portion of the recirculating load and thereby increasing the likely overall gravity gold recovery.

Fluidising water for the gravity concentrators will be supplied by dedicated pumps from the process water tank. Process water (instead of raw water) will be used due to the constraints on the water balance. In-line filters on the fluidising water pumps suctions will be installed.

Pre-leach Thickening

As well as allowing the grinding circuit to operate at lower cyclone overflow densities, inclusion of a pre-leach thickener ahead of the leach circuit achieves higher and more constant leach feed density and recovers milling water for immediate re-use in the plant. The higher leach feed density reduces the overall tankage volume required to achieve the target residence time, improves slurry mixing characteristics and reduces leach reagent costs.

The pre-leach thickener recovers water for immediate re-use in the milling circuit. The thickener will be operated at high underflow densities on fresh ore to recover maximum cyanide free milling water for use in re-pulping and wet screening as well as the milling circuit. The high density thickener underflow will be diluted with process water which will also allow re-use of the recycled free cyanide and solution gold.

Laboratory testwork indicates that the fresh ore settles relatively quickly at the target grind size, with a settling rate of 0.8 $t/m^2.h$ at low flocculant requirements to achieve the target underflow density of 65% solids. The oxide ore settling rate is expected to be significantly lower and, given the possible oxide viscosity issues, a lower target underflow density of 50% w/w solids has been used for design.

Leach Circuit

The metallurgical testwork indicated the following:

- Very high gravity gold content in the Lafigué ores with gravity gold recovery prior to leach beneficial on overall gold recovery.
- No preg-robbing occurring, therefore a CIL circuit is not necessary.
- The initial leach kinetics for the majority of the samples was rapid, with between 95% and 100% of total gold extraction being achieved in the first twelve hours of leaching. Several samples benefitted from additional leaching through to 36 hours.
- There are only low levels of deleterious elements present in the Lafigué ores and these do not appear to affect leaching.
- High leach slurry density did not impact on gold extraction. A leach slurry density of 55% solids w/w was selected for the fresh ore.
- Cyanide and lime consumptions are typically very low, although the oxide ores exhibited higher lime consumptions.
- Very low oxygen requirement during leach with no benefit noted with oxygen sparging instead of air. Air sparging was selected for the flowsheet.

Endeavour advised that the leach and adsorption circuit configuration should be leach / CIP rather than CIL.

The leach circuit was sized based on five leach tanks providing 36 hour residence time when treating 3.0 Mtpa (375 t/h) fresh ore at 55% solids w/w. Endeavour requested an additional leach tank of equivalent size be included, for use as a pre-oxidation tank or as additional leach capacity. A 6 tank leach circuit was selected for the flowsheet.

Adsorption Circuit

A carousel CIP circuit has been selected for the flowsheet. This circuit has a number of advantages over conventional CIP including no back mixing of carbon, smaller tanks and lower gold in circuit.

Carousel CIP circuit vendor Kemix modelled the CIP circuit based on processing fresh ore at the design feed gold and silver grades, maximum leach gold and silver extractions (assuming no gravity gold and silver recovery) and leach discharge slurry density. A 7 x 150 m^3 tank carousel CIP circuit, with 7.5 tonnes of carbon per tank, was recommended with either a one or two day carbon change option possible. Kemix provided expected loaded carbon gold and silver grades for both options. A one day cycle was selected for the flowsheet.

Elution and Goldroom

A 7.5 tonne capacity elution circuit with seven strips per week is required based on the CIP circuit selection.

As part of the PFS, a VEA comparing a 7.5 t elution circuits was completed recommending a SAARL elution circuit on the basis of operating benefits and reduced risks. Endeavour selected the SAARL elution circuit for the flowsheet.

Electrowinning requirements were calculated based on the modelled loaded carbon grades and the expected gravity gold pregnant solution grade. The goldroom will contain two electrowinning cells, with a third electrowinning cell dedicated to the gravity circuit.

A diesel fired elution heater, carbon regeneration kiln and smelting furnace were selected.

Plant Tailings Treatment

As part of the PFS, CIP tails cyanide reduction methods were evaluated and reported on. Based on a 1.5 Mt/a process plant, the three tails treatment methods evaluated were:

1. Air/SO_2 Cyanide Destruction. CN_{WAD} is destroyed by SO_2 and oxygen (In the presence of a copper catalyst).

2. Tails Wash Thickening and Tails Dilution. Cyanide removal from the tails stream using a tails thickener with feed and underflow dilution. Recovered cyanide is recycled to leaching via process water.

3. ReCYN Process. A proprietary process from Greengold Technology using resin beads to remove cyanide from the tails stream. Recovered cyanide is recycled to leaching.

Endeavour selected the tails wash thickener and tails dilution circuit for the flowsheet.

The CIP tails will be washed and thickened in a high rate thickener and the thickener underflow diluted as required to achieve the target discharge cyanide level. Wash and dilution water will be decant return and plant raw water makeup. The main benefits of this method are as follows:

- Recover process water from the plant tails prior to discharge and therefore reduce plant raw water requirements.

- Recover soluble gold and free cyanide in the process water for recovery / re-use in the leaching and CIP circuits, thereby reducing overall cyanide consumption and increasing precious metal recovery.

- Reduce the volume of plant tailings thereby reducing the size of the tailings pipeline.
- Smooth out minor density fluctuations from the CIP circuit to provide a consistent thickened tails slurry density for pumping.
- Lowest operating cost, highest incremental NPV and highest tails discharge slurry density.

As per the pre-leach thickener, the tails thickener was sized so that it has sufficient capacity for the higher throughput and lower settling rate oxide ore. Based on a CIP tails weak acid dissociable cyanide (CN_{WAD}) level of 100 ppm:

- For fresh ore at a CIP tails slurry density of 54% w/w solids, a target tails thickener underflow density of 65% w/w solids was selected with diluted plant tails at 55% w/w solids meeting the target discharge level of less than 50 ppm CN_{WAD}.
- For oxide ore at a CIP tails slurry density of 44% w/w solids, a target tails thickener underflow density of 55% w/w solids was selected with diluted plant tails at 43% w/w solids meeting the target discharge level of less than 50 ppm CN_{WAD}.

Raw Water

The majority of raw water for the Project will be sourced from rainfall with additional raw water available from pit dewatering. Raw water will be supplied to the plant, mine and camp and will be the feed stream for the filtered and potable water treatment plants.

Water will be harvested from a rainwater collection dam (water harvest dam [WHD]) approximately nine kilometres from the plant, and will be pumped to the water storage dam (WSD) situated adjacent to the plant and designed with the capacity to bridge the water requirement gap between wet seasons. Water from pit dewatering will also be pumped to the WSD. Raw water will be pumped from the WSD to the plant raw water storage tank.

Recycling of water will be maximised in order to minimise raw water usage. The pre-leach thickener overflowing to the mill water tank will allow direct recycle of the majority of milling water required for the wet milling screen and milling circuit with make-up process water pumped as required from the process water tank.

With TSF decant return and the plant make-up raw water added to the tails wash thickener, the thickener overflow will report directly to the process water tank and will provide the majority of process water requirements.

17.1.4 Key Design Criteria

The key process design criteria listed in Table 17.1.4 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1.4 Summary of Key Process Design Criteria

	Unit	Fresh	Oxide[1]	Source [2,3,4,5,6,7,8,9]
Plant Throughput	t/y	3,000,000	3,500,000	EDV
Life of Mine (LOM) Ore Blend		92%	8%	Snowden
Design Gold Head Grade	g Au/t	2.5	2.0	Assumed
Gravity Gold Recovery	%	60	60	EDV
Design Overall Gold Recovery	%	97	97	Testwork
Primary Crushing Plant Utilisation	%	70	88	Lyco / OMC
Secondary Crushing Plant Utilisation	%	70	-	Lyco / OMC
HPGR Crushing Plant Utilisation	%	86.7	-	Lyco / OMC
Milling / Leaching Plant Utilisation	%	91.3	88	Lyco / OMC
ROM Ore Top Size	mm	800	500	OMC
Ore SG		2.83	2.70	Test / OMC
Comminution Circuit		Prim/Sec/HPGR Crush / Reverse fed Ball Mill	Primary Crush / Direct fed Ball Mill	Endeavour / OMC
Crush Size, P_{80}	mm	7.2	101	OMC
Milling Circuit Top Size, P_{100}	mm	4	101	OMC
Target Grind Size, P_{80}	µm	75	75	Testwork
Cyclone Overflow Density	% solids w/w	25	30	Testwork
Pre-leach Thickener Solids Loading	$t/m^2.h$	0.8	0.5	Testwork / Lyco
Pre-leach Thickener Underflow Density	% solids w/w	65	50	Testwork / Lyco
Leach Feed Slurry Density	% w/w	55	45	Testwork / Lyco
Leach Residence Time	hrs	36	27	Testwork / Lyco
Number of Leach Tanks		1 Pre-Ox / 5 Leach	6 Leach	Lyco / EDV
Average Cyanide Consumption[6]	kg/t	0.13	0.21	Testwork
Average Quicklime Consumption[7]	kg/t	0.19	1.91	Testwork
Adsorption Circuit		Carousel CIP	Carousel CIP	EDV / Lyco
Number of Adsorption Tanks		7	7	Kemix
Carbon per CIP Tank	t/tank	7.5	7.5	Kemix
Carbon Cycle Time	days	1	1	Lyco
CIP Carbon Loading	g Au+Ag/t	3,893	4,017	Kemix / Lyco
Elution Circuit Type		Split AARL		EDV
Elution Circuit Capacity	t/strip	7.5		Lyco
Frequency of Elution	strips/week	7		Lyco
Tailings Thickener Solids Loading	$t/m^2.h$	0.8	0.5	Testwork / Lyco
Tails Thickener Underflow Density	% solids w/w	65	55	Testwork / Lyco

Notes:
1. Oxide includes oxide and transition ores
2. 'Testwork' refers to metallurgical testwork conducted.
3. 'EDV' refers to advice / agreement from Endeavour Mining
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 100 ppm residual NaCN in the CIP tail solution.
7. Lime consumption based on 90% CaO.
8. 'Kemix' refers to carousel CIP vendor.
9. 'Snowden' refers to advice from Snowden Optiro.

17.2 Process and Plant Description

The following section provides a description of the process plant, reagents and services area. A summary of the major equipment sizes is provided in Table 17.2.1. Detailed flowsheets, general arrangement drawings, mechanical equipment list, electrical load list are available on request.

Table 17.2.1 Summary of Key Equipment

	Units	
ROM Bin	t	200
Primary Crusher (w x l)	mm	1,400 x 1,200
Secondary Crusher Capacity	t/h	855
Crushed Ore Stockpile	t	7,420
HPGR rolls (dia x EGL)	m	1.59 x 1.45
Ball mill (dia x EGL)	m	6.40 x 10.96
Gravity Concentrators Capacity	t/h	800
Intensive Cyanidation Reactor Capacity	t/d	5
Hydrocyclones - (No. x dia)	mm	20 x 400
Pre-Leach Thickener dia	m	34
Leach Tanks - (No. x vol)	m^3	6 x 3,200
CIP Tanks - (No. x vol)	m^3	7 x 150
Tails Wash Thickener diameter	m	34
Elution Circuit Type		Split AARL
Elution Circuit Capacity	t	7.5
Elution Heater Capacity	kW	2,500
Carbon Regen Kiln Capacity	kg/h	375
Pregnant Solution Capacity	m^3	2 x 80
Electrowinning cells (No. x size)		3 x 1m sq x 22 cathode
Smelting Furnace Capacity	L	49
Event Pond Capacity	m^3	4,800
Cyanide Storage	m^3	25
Lime Storage	t	80
Flocculant Storage	m^3	100
Caustic Storage	m^3	10
Leach Air Blowers Capacity	m^3/h FAD	5,670
Raw Water Storage	m^3	1,000
Process Water Storage	m^3	500
Filtered Water Storage	m^3	300
Filtered Water Plant Capacity	m^3/h	60
Potable Water Plant Capacity	m^3/h	2

The Lafigué process plant will operate on either 100% fresh ore (possibly blended with a small amount of oxide) or 100% oxide ore. The fresh and oxide ores have different comminution flowsheets, specifically:

- Fresh ore – closed circuit secondary crushing with crushed ore storage, closed circuit HPGR crushing with HPGR crushed ore storage and reverse feeding of the ball mill.
- Oxide ore – open circuit primary crushing with direct feeding of the ball mill.

17.2.1 ROM Pad

Haul trucks operating from the open pit will deliver run-of-mine (ROM) ore to the ROM pad where it will be direct tipped to the ROM bin or dumped in blending 'finger' stockpiles arranged by ore gold grade and lithology. A front end loader (FEL) will be used to reclaim and tram ore from the various stockpiles to the ROM bin.

Ore will be blended under the guidance of mine geologists and process personnel to maintain a relatively constant feed grade to the process plant.

17.2.2 Crushing Circuit

Primary Crushing

ROM ore will be loaded into the crushing circuit feed bin (ROM bin) by direct tipping from the mining trucks or by FEL from the ROM pad stockpiles. A static grizzly will be fitted to the ROM bin to protect the downstream equipment from oversize material and will be inclined for easy removal of oversize to minimise stoppages. A mobile rock breaker will be utilised to break oversize rocks on the grizzly or ROM pad.

Water sprays on the ROM bin will provide dust suppressions during ore tipping. The ROM bin will be fitted with bin blasters to assist in material handling of the oxide ores.

ROM ore will be drawn from the ROM bin by a variable speed apron feeder and discharge onto a vibrating grizzly. The grizzly oversize will report to the primary jaw crusher. A fixed rock breaker mounted adjacent to the primary crusher will be utilised to break oversize rocks in the crusher.

The primary crusher product and grizzly undersize will discharge onto the crusher discharge conveyor. A weightometer will indicate the primary crushed ore tonnage. A magnet mounted above the crusher discharge conveyor will remove tramp metal.

The crusher discharge conveyor will discharge, via the oxide bypass chute, to either:

- the crushing screen feed conveyor if fresh ore is being processed or
- the oxide feed conveyor if oxide ore is being processed. The oxide feed conveyor will transfer the primary crushed oxide ore directly to the ball mill feed spout. A weightometer will indicate the ball mill feed tonnage.

Screening and Secondary Crushing (Fresh Ore)

Crushed ore will be transferred via the crushing screen feed conveyor to the crushing screen. Crushing screen oversize will report to the crushing screen oversize conveyor and will be transferred to the secondary crusher feed conveyor. A magnet mounted at the head of the crushing screen oversize conveyor will remove tramp metal from the screen oversize. A metal detector located above the secondary crusher feed conveyor will detect any remaining tramp metal in the secondary crusher feed and will, when activated, stop the conveyor so that tramp metal can be manually removed by an operator. A weightometer will indicate the secondary crusher feed tonnage.

The secondary crusher feed conveyor will discharge to the secondary crusher feed bin. Ore will be drawn from the bin by the secondary crusher feeder and will discharge to the secondary cone crusher. Secondary crushed ore will report to the crusher discharge conveyor (along with primary crushed ore) and will discharge to the crushing screen feed conveyor.

Crushing screen undersize, with a top size of 35 mm, will report to the crushed ore conveyor and will be transferred to the crushed ore stockpile. A weightometer will indicate the crushed ore stockpile feed tonnage.

A dust collector will reduce dust emissions at the primary and secondary crusher discharges and water sprays will be used for dust suppression in the screening area.

Crushed Ore Stockpile, HPGR Crushing and Wet Screening (Fresh Ore)

The crushed ore conveyor will discharge to the crushed ore stockpile. Secondary crushed ore will be withdrawn from the stockpile at a controlled rate by variable speed feeders and will discharge to the stockpile reclaim conveyor. A weightometer will indicate the reclaimed ore tonnage. Reclaim tunnel ventilation and dust collection will be provided.

The stockpile reclaim conveyor will discharge to the HPGR feed conveyor, where it will combine with recycled milling screen oversize, and will be transferred to the HPGR feed bin. A weightometer will indicate the HPGR feed tonnage. A magnet mounted above the HPGR feed conveyor will remove tramp metal.

HPGR feed will be drawn from the feed bin by the HPGR feeder and will discharge to the HPGR. A metal detector located above the HPGR feeder will detect any tramp metal in the HPGR feed and will activate a diverter gate to prevent steel from reporting to the HPGR. This diverter gate will also allow HPGR feed to bypass the HPGR if required. Ore will be crushed in the HPGR and will discharge to the HPGR product conveyor.

The HPGR product conveyor will discharge to the HPGR product surge bin with excess HPGR product overflowing from the surge bin to a stockpile. When the HPGR is offline, HPGR product can be reclaimed by FEL from the stockpile into the HPGR product reclaim hopper onto the milling screen feed conveyor to maintain feed to the milling circuit.

HPGR product will be drawn from the surge bin via the milling screen feed conveyor and will discharge to the pulping box. A weightometer will indicate the milling screen feed tonnage. Water will be added to de-agglomerate the HPGR crushed ore with the pulped slurry discharging to the wet milling screen.

Milling screen oversize will discharge to the milling screen oversize conveyor and will report to the HPGR for further crushing, via the HPGR feed conveyor. When the HPGR is offline, the milling screen oversize can be bypassed to the oversize stockpile conveyor which will transfer the oversize to the oversize stockpile, adjacent to the HPGR product reclaim hopper. Oversize material can be reclaimed into the circuit by FEL as required.

Milling screen undersize slurry will report to the grind and classification circuit.

17.2.3 Grinding and Classification Circuit

Crushed ore will be milled to achieve the required grind size for effective gold recovery. The grinding circuit will consist of a ball mill in closed circuit with a cluster of classification hydrocyclones (cyclones).

When treating fresh ore, the ball mill will be reverse fed with milling screen undersize slurry reporting the mill discharge / cyclone feed hopper and cyclone underflow reporting to the ball mill feed. When treating oxide ore, the ball mill will be direct fed with primary crushed oxide ore reporting to the ball mill feed. Water will be added to the ball mill feed as required to achieve the required milling density.

The ball mill will be equipped with a variable speed drive to assist with managing variations in feed hardness. When treating oxide ore, the ball mill will be operated at a lower speed and reduced ball charge to minimise overgrinding of the low competency oxide ore. When treating fresh ore, the ball mill speed and ball charge will be increased to mill the competent fresh ore.

Steel grinding media (balls) will be added to the ball mill feed hopper as required using a hoist and a ball kibble. Balls will be loaded into the kibble which will be hoisted to the ball charging level and discharged into the ball mill.

Lime slurry, used for pH control in the leach circuit, will be added to the ball mill feed box from the lime ringmain. The rate of lime addition will be ratioed to the new feed tonnage rate or added to achieve the target leach pH.

The ball mill will discharge via a trommel to the mill discharge hopper and for fresh ore, will combine with the milling screen undersize slurry. Trommel oversize (worn steel grinding media and small pebbles) will report to the scats bunker. Dilution water will be added to the mill discharge hopper and the slurry will be pumped to the cyclones.

Cyclone underflow (coarse material) will report to the cyclone underflow splitter box where a portion of the slurry will be split to the gravity circuit and the remaining slurry will report to the ball mill feed box. Cyclone overflow (fine material) will gravitate to the trash screen.

17.2.4 Gravity Circuit

The gravity circuit will consist of two parallel trains each containing a feed scalping screen and a centrifugal concentrator. An intensive leach reactor (ILR) will process the combined gravity concentrates from the two trains.

A portion of milling circuit cyclone underflow will report, via the cyclone underflow splitter box, to the gravity circuit. The gravity feed slurry will be split to feed the two gravity scalping screens with combined screen oversize reporting to the ball mill feed. The screen undersize from each screen will feed a centrifugal concentrator to remove coarse gold from the milling circuit recirculating stream. The tails slurry from the centrifugal concentrators will gravitate to the mill discharge hopper. The concentrators will be operated on a semi-batch basis with periodic discharge of the gravity concentrates to the ILR.

The ILR will be located within a secure enclosure in the milling area and will process the gravity concentrate once per day under intensive cyanidation conditions. Caustic and sodium cyanide solutions from the respective storage tanks will be metered into the ILR and hydrogen peroxide will be metered into the leach to provide the required oxygen demand. Flocculant will be used to settle the leach solids with the resulting gold-rich leach solution pumped to the gravity pregnant solution tank. ILR tails will be pumped to the mill discharge hopper for further processing.

17.2.5 Trash Screening and Pre-Leach Thickening

Milled slurry will gravitate from the cyclone overflow to the trash screen where any grit, wood chips and plastics will be removed. Trash (screen oversize) material will report to the trash bunker. Trash screen underflow will gravitate to the thickener.

The thickener feed slurry will be mixed with flocculant to facilitate effective settling. Thickener underflow slurry will be pumped to the leach feed distribution box via an automatic two stage leach feed sampler. Thickener overflow will gravitate to the milling water tank for distribution as dilution water to the milling circuit. Process water will be added to the milling water tank as required to make up the milling water requirements.

17.2.6 Leaching

The leaching circuit will consist of six tanks in series operating in a one pre-aeration / five leach tank configuration. The leach tanks will be interconnected with launders and slurry will flow by gravity through the tank train. Each tank will be fitted with an agitator to ensure the tank is well mixed. All tanks will be fitted with bypass facilities to allow any tank to be removed from service.

Pre-leach thickener underflow will be pumped to the leach feed box and diluted with process water to the required leach solids concentration. Lime slurry will be added to the leach feed box as required to ensure that the slurry pH is suitable for cyanidation. Sodium cyanide solution will be metered from the cyanide ring main into the first leach tank, with further addition points located down the leach train for use as required. Low pressure air will be added to the leach tanks, via air spargers at the bottom of each tank, to provide oxygen to the leach. Slurry from the last leach tank will gravitate to the adsorption circuit.

17.2.7 Adsorption

A Pumpcell® CIP plant will be used to recover the solution gold from the leached slurry onto activated carbon. The CIP circuit will consist of seven tanks with each tank fitted with an agitator / intertank screen. The CIP circuit will operate as a carousel, with carbon retained within the tanks and the slurry feed advanced counter current to the carbon adsorption stages. The CIP circuit will operate on a daily cycle with loaded carbon removed from the selected tank and further processed to recover the gold.

Leached slurry will gravitate to the CIP feed launder and will be directed to the currently selected first (head) tank in the CIP train. Slurry will flow from the head tank through to the last tank and exit the circuit via the discharge manifold. Once the target gold on carbon loadings have been achieved in the head tank, the tank will be isolated and feed slurry will be directed to the next tank in the carousel sequence. The contents of the head tank will be pumped to the loaded carbon recovery screen, via the loaded carbon recovery pump, to separate the loaded carbon from the slurry. The screened slurry will be returned to the CIP feed launder and the loaded carbon will report to the acid wash column. This operation will take about one hour.

Regenerated carbon will be pumped from the carbon transfer hopper to the empty isolated CIP tank over a 30 minute period. The tank feed valving will be changed to allow the tank to be brought back on line as the new last tank in the carousel sequence.

CIP tails slurry from the last CIP tank will discharge to the CIP tails hopper and will be pumped to the carbon safety screen via an automatic two stage CIP tails sampler.

17.2.8 Elution Circuit and Goldroom Operations

The following operations will be carried out in the elution and goldroom areas:

- Acid washing of carbon.
- Stripping of gold from loaded carbon using the split AARL method.
- Regeneration of barren carbon.
- Electrowinning of gold from pregnant solution.
- Smelting of electrowinning products.

The stripping and goldroom areas will operate seven days per week, if required, with the majority of loaded carbon preparation and stripping occurring during day shift. The split AARL stripping circuit will be automated and will contain a separate acid wash and an elution column.

Acid Wash

Loaded carbon will be received into the acid wash column from the loaded carbon screen. During acid washing, a dilute solution of hydrochloric acid will be pumped into the bottom of the column to remove contaminants, predominantly carbonates, from the carbon. After the soak period has elapsed, the loaded carbon will be rinsed with water with dilute acid and rinse water pumped to the tailings hopper for disposal. Washed carbon from the acid wash column will be hydraulically transferred from the acid wash column to the elution column and the water will be drained out.

Pre-soak and Elution

The split AARL elution process will be used to recover gold from the loaded carbon in the elution column.

Strip solution will be pumped from the lean eluate tank through the inline heat transfer system (heat exchangers and strip solution heater) and injected into the base of the elution column. Caustic and sodium cyanide solutions will be pumped from the respective storage tanks and injected into the suction line of the strip solution pump. The loaded carbon will be pre-soaked in the cyanide / caustic solution to prepare the carbon for gold desorption.

The carbon will then be eluted by hot strip solution which will pass out of the circuit to either of the two pregnant solution tanks initially and then to the lean eluate tank. Outgoing strip solution will pass through the recovery heat exchangers to heat the incoming strip solution.

Once desorption is complete, the carbon is cooled to <95°C prior to transfer to carbon regeneration.

Electrowinning and Goldroom

Once the elution cycle is completed, recovery of gold by electrowinning will proceed. Pregnant solution will be pumped from either of the pregnant solution tanks through the parallel electrowinning cells. Direct current will be passed through stainless steel anodes and stainless steel mesh cathodes within the electrowinning cells and electrolytic action will cause the gold in solution to form a gold rich sludge on the cathodes.

Electrowinning will take approximately 4 to 10 hours. Solution discharging from the electrowinning cell will return by gravity to the pregnant solution tank. The system will be configured to allow single or multiple pass electrowinning. Electrowinning will continue until the solution exiting the electrowinning cells is depleted of gold. The barren electrolyte will be pumped to the leach circuit feed.

An overhead crane within the goldroom will be provided to assist with handling of cathodes and anodes. The gold sludge will be removed from the cathodes by washing with high pressure water jets. The resulting slurry will be filtered in a vacuum pan filter and the solids then dried in an oven. The sludge will then be direct smelted with fluxes in a diesel-fired furnace to produce doré bars. Slag from smelting operations will be returned to the milling circuit.

Fume extraction equipment will be provided to remove fume gases from the electrowinning cells, drying oven and barring furnace. In addition to this, fresh air fans will be provided to ensure there is adequate ventilation inside the gold room. A wet scrubber on the barring furnace fume extraction will be used to recover any gold particles and dust.

The gravity pregnant solution tank and electrowinning cell will be located adjacent to the elution pregnant solution tanks and cells. Pregnant solution will be pumped through the electrowinning cell with electrowon gold plating on the cathodes. The gold from the gravity electrowinning cell cathodes will be treated separately to the CIP electrowinning cathodes to assist in metallurgical accounting. The barren electrolyte will be pumped to the leach circuit feed.

Carbon Regeneration

The carbon regeneration circuit will consist of a dewatering screen, diesel fired regeneration kiln, quench tank, carbon sizing screen and carbon transfer hopper.

Barren carbon will be hydraulically transferred from the elution column to the carbon dewatering screen. The dewatering screen will remove the majority of contained water prior to entering the feed hopper of the regeneration kiln.

In the regeneration kiln feed hopper, any residual and interstitial water will be drained from the carbon before it enters the kiln. In the kiln, the carbon will be heated to 650 - 750°C to allow effective reactivation to occur. Reactivated carbon from the kiln will discharge to the quench tank where it will be quenched with raw water. The quenched carbon will be pumped to the carbon sizing screen where fine carbon and the majority of contained water will be removed. The carbon will discharge to the carbon transfer hopper. The water and carbon fines will report to the area sump and will be recycled to the quench tank or pumped to the tails hopper for disposal. This stage occurs over about 20 hours.

When required, the reactivated barren carbon will be pumped from the carbon transfer hopper to an empty CIP tank. This stage occurs over about 30 minutes.

17.2.9 Carbon Safety Screen and Tailings Wash Thickening

CIP tails will be pumped to the carbon safety screen to recover any carbon escaping from worn screens or overflowing tanks. Screen oversize (recovered carbon) will report to the carbon bin for subsequent return to the CIP circuit and screen underflow will gravitate to the tails wash thickener.

The tails wash thickener will be used to thicken and wash the plant tails in order to reduce the cyanide in the plant tails and to recover free cyanide and gold in solution. The thickener underflow can be diluted with raw water and/or decant return to further reduce the cyanide level in the plant tails.

Carbon safety screen underflow and the majority of the wash water (decant return / raw water makeup) will report to the thickener feed and will be mixed with flocculant to facilitate effective settling. Thickener overflow, containing the majority of the cyanide and soluble gold, will overflow to the process water tank for distribution as dilution water to the leach circuit and as make-up to milling water. Thickener underflow will report to the tails hopper

17.2.10 Tailings Disposal

Tails thickener underflow will be combined with any other waste streams in the tails hopper. As required, the plant tails will be diluted with the remaining wash water to achieve the cyanide discharge target, prior to pumping to the tails storage facility (TSF) via either or both of the parallel tails pipelines.

Tailings will be deposited into the TSF using established discharge and decant methods. Supernatant water (decant return) will be directed to a pond and will be pumped to the plant process for re-use in tails wash thickening / dilution.

17.2.11 Reagents

Reagent Storage

Reagents will be received on site either in bulk or in shipping containers, with a minimum of six weeks capacity stored on site to ensure that supply interruptions due to port, transport or weather delays do not restrict production.

Quicklime

Slaked lime slurry will be prepared in a vendor supplied slaking plant for use in the process. The lime slaking plant will consist of a lime silo, feeder, slaking mill, mill discharge hopper and pump.

Quicklime will be delivered to the site as a dry powder in bulk tankers. The bulk tanker will be off-loaded pneumatically directly into the lime silo and will be metered into the slaking mill along with process water. The slaked lime slurry will discharge from the mill to the discharge hopper and will be pumped, via the lime slaking screen, to the slaked lime storage tank.

Lime slurry will be pumped from the lime storage tank through a ringmain to feed the ball mill and the leach circuit as required for leach circuit pH control.

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.

Quicklime will also be delivered to site in bulk bags which can be used if there are delays in the bulk tanker deliveries. The bulk bags will be hoisted and unloaded via a bag breaker into the lime transfer hopper and the quicklime will be pneumatically transferred to the lime storage silo.

Cyanide

Sodium cyanide will be delivered as dry briquettes in bulk bags. The cyanide bags will be added to the mixing tank via a bag breaker and dissolved in process water to achieve the required solution strength.

The cyanide solution will be transferred to the storage tank for use in the process. Cyanide will be reticulated to the leach circuit via a ring main and dosed to the leach tanks as required. A dedicated pump will meter the cyanide solution to the elution circuit and ILR as required.

Caustic

Caustic soda (sodium hydroxide) will be delivered to site as dry 'pearl' pellets in bulk bags. The bulk bags will be added to the mixing tank via a bag breaker on the receiving hopper. The pellets will be discharged into the mixing tank via a screw feeder to prevent splash back from the tank and dissolved in filtered water to the required solution strength.

The caustic dosing pump will meter the caustic solution to the elution circuit and ILR as required.

Hydrochloric Acid

Concentrated hydrochloric acid will be delivered to site in intermediate bulk containers (IBCs). The concentrated hydrochloric acid will be pumped into the acid mixing tank where it will be diluted with filtered water to achieve the required acid wash solution concentration. The dilute acid solution will be pumped to the acid wash column 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 CIP inventory. This addition point will allow any fine carbon particles to be removed on the carbon sizing screen prior to entering the CIP tanks.

Grinding Media

Grinding balls will be delivered to site in drums or bulk bags. The balls will be loaded into the ball mill feed box via a ball charging kibble as required to achieve the target power draw.

Flocculant

Flocculant will be delivered to site in bulk bags as a dry powder and will be added to the flocculant plant feed hopper. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant with raw water and transfer the mixed flocculant to an aging / storage tank after each mixing cycle is complete.

Dedicated pumps will meter the flocculant solution to the pre-leach thickener and tails wash thickener as required. A fill point off the flocculant storage tank will be used to periodically fill an IBC with mixed flocculant for use in the ILR.

Hydrogen Peroxide

Hydrogen peroxide (50% w/w) will be delivered to site in IBCs. The hydrogen peroxide will be used in the ILR.

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 gold sludge prior to smelting.

Diesel

Diesel will be delivered to site by bulk tankers and transferred to the bulk storage tanks. The diesel will be used in the mine, the process plant, for emergency / backup power generation and to refuel site vehicles.

Diesel will be pumped from the bulk storage facility to the plant day tank for use in the elution heater, smelting furnace and carbon regeneration kiln. A diesel unloading point will also be provided adjacent to the plant day tank for manual unloading from a diesel tanker.

17.2.12 Services

Raw Water

Raw water for the project will be sourced from a water harvest dam (WHD) designed to collect rain water during the wet season. During and immediately after the wet season, water will be pumped from the WHD to the water storage dam (WSD). Water will be pumped from the WSD to the plant raw water tank and the camp raw water tank.

The plant raw water tank will have sufficient capacity to minimise the impact of short term supply interruptions. The raw water pumps will distribute raw water to the plant and MSA. Raw water make-up to the plant will be via the tails wash thickener and tails dilution circuit with overflow from the raw water tank used to make-up any additional process water requirements.

Fire Water

Fire water for the process plant and MSA will be drawn from the raw water tank. Suctions for other water services fed from the raw water tank will be at a higher level to ensure a fire water reserve always remains in the raw water tank.

The fire water pumping system will contain:

- an electric jockey pump to maintain fire ring main pressure
- an electric fire water delivery pump to supply fire water at the required pressure and flowrate
- a diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.

Fire hydrants and hose reels will be placed throughout the process plant, fuel storage, plant offices and MSA 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. Process water will be pumped to the mill water tank in order to maintain the tank level. The mill water pumps will pump to the milling circuit.

Process Water

Plant process water will consist of tails wash thickener overflow, decant return water pumped from the TSF and raw water tank overflow. The process water pumps will reticulate process water to the process plant.

Antiscalant will be added as required to condition the water and reduce fouling of the pipelines, spray nozzles and screen decks.

Fluidising Water

Fluidising water will be provided by a dedicated fluidising water pump per centrifugal concentrator. The pumps will draw water from the process water tank via inlet filters for use in the centrifugal concentrators.

Filtered Water

Filtered water for the process plant will be produced by treating raw water in the filtered water treatment plant. The treatment plant will consist of clarification through flocculant addition, sand filtration, carbon filtration and biocide dosing. Filtered water will report to the filtered water tank and will be distributed to the plant by the filtered water pumps for use as required in elution and reagent mixing.

Gland Water

Water from the filtered water storage tank will be distributed as gland service water using gland water pumps.

Potable Water

Potable water will be generated by treatment of filtered water in the plant potable water treatment plant. The water treatment facility will include micro filtration, ultra-violet sterilisation and chlorination. Potable water will be stored in the plant potable water tank and will be reticulated to the site ablutions, safety showers and other plant potable water outlets, including the MSA. An ultra-violet sterilisation unit will be installed on the outgoing potable water distribution header.

Cooling Water

Cooled water for the ball mill will be provided 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.

Leach, Plant and Instrument Air

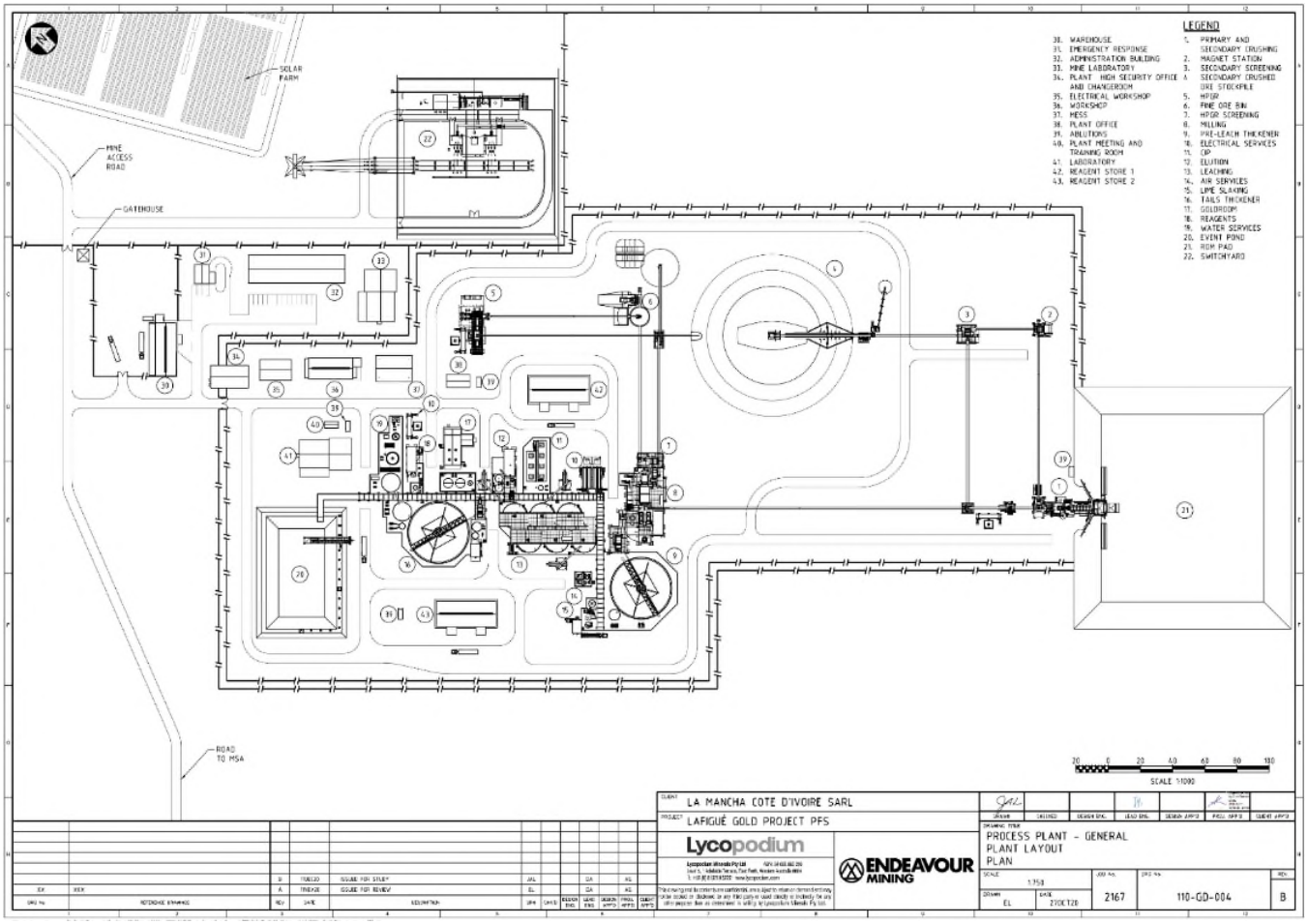
Low pressure (LP) air will be supplied using air compressors with pressure let down to achieve the required LP air. LP air will be reticulated to the leach tanks and injected into the slurry via spargers in each tank.

Plant and instrument air will be supplied from air compressors. The air will be filtered and dried before distribution to separate area specific air receivers which will supply the plant. A dedicated plant air compressor and receiver will be located in the crushing area.

17.3 Plant Area Design

The civil, mechanical and electrical design of the plant facilities is based on industry standard practice and Lycopodium's extensive experience in gold plant design and project implementation in West Africa and elsewhere. An overall General Arrangement drawing is provided in Figure 17.3.1.

Figure 17.3.1 Overall General Arrangement



The plant site location is primarily selected so that the ROM pad location, feeding ore to the plant, is located as close as practicable to the open cut mining area, while considering blast exclusion zones. A siting study was undertaken for Endeavour by Knight Piésold.

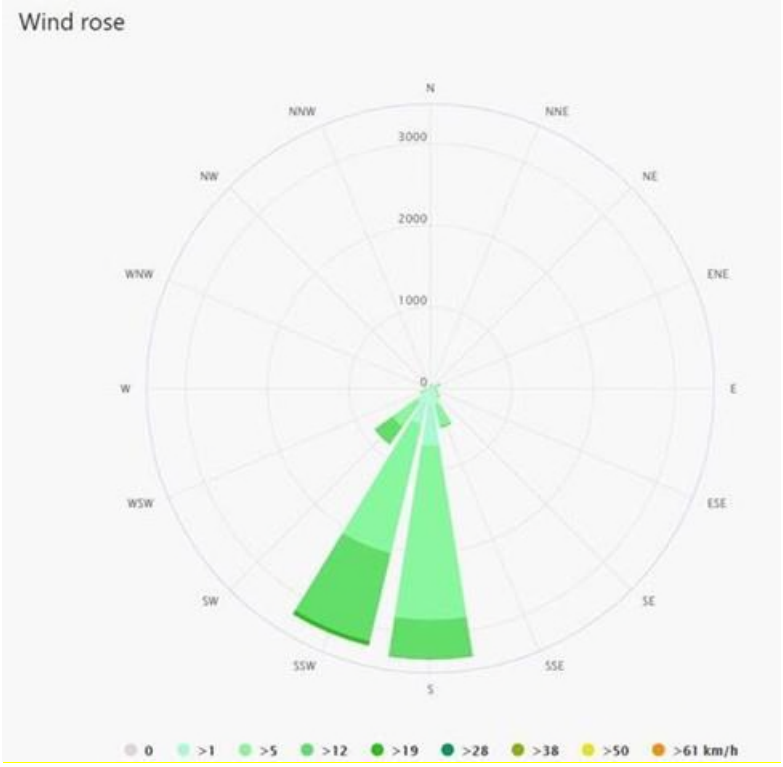
The alignment of the plant is reflective of the suitability of the contours and required foot print area for founding the heaviest loaded equipment or plant on cut, rather than fill. The TSF is located southwest of the process plant and the natural ground level slopes towards this location.

The final location of the plant site may be optimised further when detailed geotechnical assessments are completed during the FS phase.

The layout of the plant roads and equipment considers the need for ease of access to perform maintenance activities. Utilisation of mobile cranes for maintenance activities has been maximised.

The initial weather data analysis indicates that the wind will blow across the plant, typically from NNE to SSW, as shown in the wind rose in Figure 17.3.2.

Figure 17.3.2 Wind Rose



The mining activities and ROM pad are southeast of the plant site and this is considered to be advantageous from a dust control perspective.

17.4 Electrical Design

17.4.1 Installed Load and Maximum Demand

The plant site-wide electrical power requirements for infrastructure, mining and processing were calculated on the basis of preliminary equipment sizing. The installed load and maximum demand for the Project is summarised below.

Table 17.4.1 Installed Load and Maximum Demand

Installed Load		Maximum Demand		Average Demand	
(MW)	(MVA @ 0.85pf)	(MW)	(MVA @ 0.85pf)	(MW)	(MVA @ 0.85pf)
23.4	26.2	16.5	18.6	15.2	17.9

The maximum demand is based on an average load factor of 80% for most areas. The 2,200 kW HPGR and 7,900 kW ball mill represents approximately 43% of the total load giving an overall average load factor of 71%. Further details of the plant electrical requirements are included in the electrical load list available on request.

17.4.2 Grid Power Supply

ECG completed a study of options available to supply the power requirements for the Project. The options assessed included the following:

- Grid power from the CIE supply system 225 kV bus at Dabakala Substation.
- HFO/LNG on site power station to be fully purchased by Endeavour.
- HFO/LNG on site power station, to be built by an Independent Power Provider (IPP) with an option to transfer the asset to Endeavour following an initial 5 years of operation.
- A combination Grid and 7 MW solar hybrid option.

Based on the cost and risk evaluation presented in the ECG report, the decision was taken to implement the hybrid grid/solar option for the PFS. The hybrid selection has the potential to reduce the overall unit power cost from US$ 0.1117/kwh to US$ 0.0900/kwh, at an initial capital cost of approximately US$ 7.0M.

The power supply will be sourced from the grid. La Société des Energies de Côte d'Ivoire (CI-E) owns the National Interconnected Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government.

The preferred option is to connect to the CIE electricity grid at Dabakala Substation and to run a 225 kV transmission line to the mine site at Lafigué.

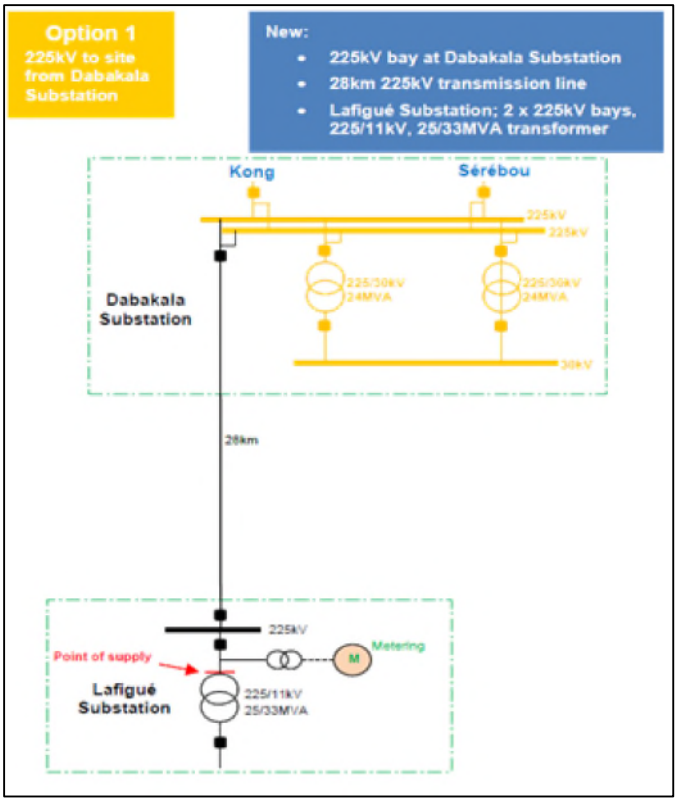
The Plant is estimated to have a connected load of approximately 23.4 MW with a Maximum Demand of 16.6 MW and an expected energy consumption of 133 GWh/y.

The scope of works for the 225 kV power supply to the Project includes the modifications necessary in the electricity grid network and the infrastructure required at the plant site with the following battery limits:

- The CIE supply system 225 kV bus at Dabakala Substation.
- The outgoing 11 kV feeder in the CIE Lafigué Substation.
- The services connection point at the Lafigué Substation.
- The LV power connection point at the Lafigué Substation.

The Project involves the extension of the Dabakala Substation by extending the existing 225 kV bus, adding a 225 kV transmission line feeder bay, construction of 28 km of 225 kV single circuit lattice tower transmission line, and constructing a substation at Lafigué site. The Lafigué Substation would be owned and operated by CIE and the Project would take a 225 kV tariff metered feeder, install a 225/11 kV transformer in their substation and take an 11 kV feeder to the plant main 11 kV switchboard. Figure 17.4.1 shows this grid connection.

Figure 17.4.1 CIE Grid Connection



The point of supply and point of change of ownership will be the primary 225 kV terminals of the 225/11 kV step down transformer at Lafigué substation near the Lafigué site. This will be the point of the tariff metering.

The transmission line alignment as it approaches and enters the Project site is shown in Figure 17.4.2.

Figure 17.4.2 Project Site Transmission Line Alignment



The transmission line route to site has been discussed with CIE and a suggested route plotted with reference to google earth topography. Figure 17.4.2 shows the intended route for the 225 kV transmission line to site.

17.5 Control Systems

17.5.1 General

The general control philosophy for the plant will be one with a high level of automation and remote control facilities. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room will house two PC based operator interface terminals (OIT). Two additional servers will also be located here to act as the control system supervisory control and data acquisition (SCADA) servers in a redundant configuration. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

The process control system that will be used for the plant will be a programmable logic controller (PLC) and SCADA based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment.

Vendor supplied packages will use vendor standard control systems throughout the project. Vendor packages will generally have limited interfaces with the PCS such that control and set-point changes may have to be adjusted locally. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

17.5.2 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 the extended cable run lengths back to the switchroom. These remote I/O modules are electrical field enclosures housing power supplies, network equipment, marshalling terminals and PLC remote I/O nodes. 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.

17.5.3 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.

The OITs will allow drives to be selected to Auto, Local, Remote or Out-of-Service modes via the drive control popup. Statutory interlocks such as emergency stops and thermal protection will be hardwired and will apply in all modes of operation. All PLC generated process interlocks will apply in Auto and Remote modes. Process interlocks will be disabled or bypassed in Local mode except for critical interlocks such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start pushbutton. Remote selection will allow the equipment to be started from the control room via the drive control popup. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

17.5.4 Control Loops

Regulatory control loops will be provided for all key process circuits to provide optimal functionality without regular operator intervention.

There will be two modes for loop controlled set points available in the OIT. These are 'Loop Auto Mode' and 'Loop Manual Mode'. In Loop Auto Mode (analogous to cascade control), the set point will be predominantly controlled by the applicable 'master' PID loop (e.g. for thickener underflow pumping control the bed pressure PID controller output will supply a set point for the thickener underflow flow control loop which ultimately controls the speed of the thickener underflow pump). In Loop Manual Mode, set point may be entered manually from the loop set point pop up in the OIT.

Where required, analogue set points from the PCS to vendor supplied control panels can be done either via the OIT or via vendor control panels.

17.6 Metallurgical Accounting

Some provision will be made to assist in the metallurgical accounting during operation. Weightometers will be located on conveyors throughout the plant:

- Crusher Discharge Conveyor - primary crushed ore (new plant feed), located between the primary crusher discharge and the secondary crusher discharge.
- Oxide Feed Conveyor – primary crushed oxide ore fed directly to the ball mill.
- Secondary Crushing Feed Conveyor - crushing screen oversize material reporting to the secondary crusher.
- Crushed Ore Conveyor – crushed ore (crushing screen undersize) reporting to the crushed ore stockpile.
- The crushing screen feed tonnage can be estimated from the addition of the secondary crushing feed conveyor tonnage and crushed ore conveyor tonnage.
- HPGR Feed Conveyor 1 - crushed ore reclaimed from the crushed ore stockpile reporting to HPGR crushing circuit.
- HPGR Feed Conveyor 2- HPGR feed (combined stockpile reclaim and milling screen oversize) located after the milling screen oversize conveyor transfer to the HPGR feed conveyor.
- Milling Screen Feed Conveyor - HPGR product from the HPGR surge bin reporting to the milling screen.

- The milling screen oversize tonnage can be estimated from the difference between HPGR Feed Conveyor 1 tonnage and HPGR Feed Conveyor 2 tonnage.

- The milling circuit feed tonnage can be estimated from the difference between milling screen feed tonnage and milling screen oversize tonnage.

Automated stream samplers on the leach feed and adsorption tails streams will ensure 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 as a cross check on the milling circuit feed tonnage determined from the crushing circuit weightometers. In conjunction with the gold assays for the leach feed and adsorption tails samples, the mass flow measurements will allow the gold recovered in the leach / CIP to be calculated.

A dedicated electrowinning cell will be provided for recovery of the gold leached by intensive cyanidation of 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.

Water supplied and used in the various areas will be continuously monitored.

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/min) and accumulated (m^3) using flow meters.

18.0 PROJECT INFRASTRUCTURE

18.1 Overall Site Development

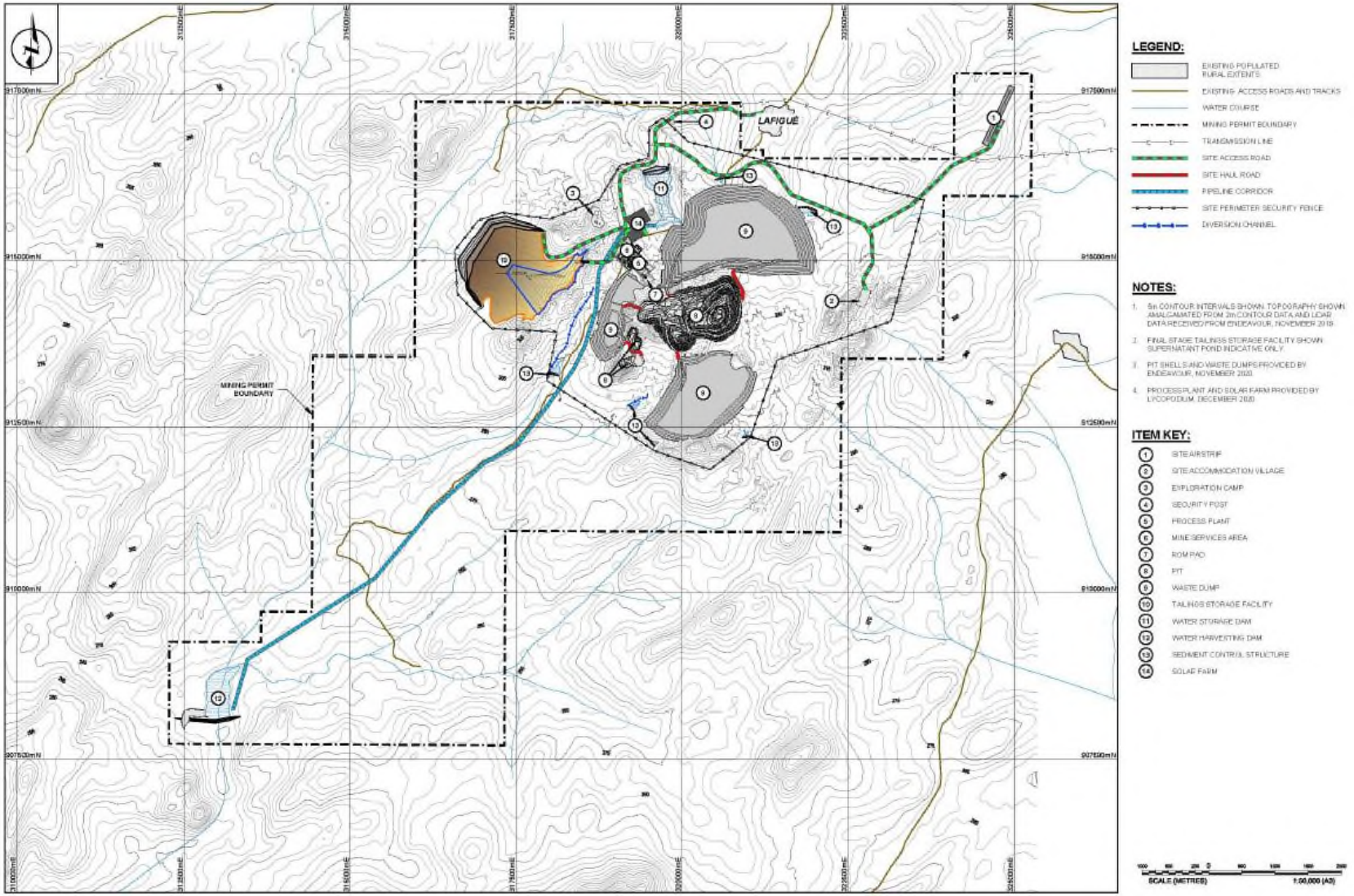
The overall site development plan is shown in Figure 18.1.1. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, airstrip, mine services area, mine open pit and mine waste dump.

The process plant is located immediately north of the open pit, with the tailings storage facility (TSF) to the west.

The main access road approaches the site from the north and the layout provides easy access for personnel and material movements.

The site as a whole will be fenced to clearly delineate the mine area, prevent animal access and deter access by unauthorised persons. Road access into the fenced area will be through a manned checkpoint. Security fencing will surround the accommodation camp and general site infrastructure. Monitored high security fencing will surround the process plant.

Figure 18.1.1 Lafigué Project Layout



18.2 Site Access

18.2.1 Main Access Road

An upgrade to the existing road / track from Bounadougou on the B412 has been allowed for in the estimate. This upgraded all-weather unsealed road will track south-east for approximately 11 km before turning due south for a further 4 km to the village of Lafigué. It is understood that the access road can become inundated in the wet season and a full survey of the route will be undertaken by Endeavour prior to the FS.

From Lafigué, a new all-weather unsealed access road will be constructed for 4.5 km to the plant site. This road will be extended 4.4 km to the site accommodation and the airstrip.

18.2.2 Site Access Road

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.

The key design parameters for the site access road are summarised in Table 18.2.1.

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 Average Recurrence Interval (ARI)
Pavement	200 mm laterite gravel wearing course

18.2.3 Minor Roads and Tracks

A number of new tracks will be constructed to access infrastructure such as the water harvesting dam (WHD). The access tracks will be cleared and graded natural earth tracks. Exact routes will be determined during construction of the Project to best fit local terrain and vegetation density.

18.2.4 Haul Roads

Mine haul roads connect the open pits, waste dumps and mine services area. The total length of the haul roads outside of the proposed waste dump extents is 1.3 km. The haul roads will be constructed progressively in line with the staged pit and waste dump development.

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.

Knight Piésold have provided design parameters for the haul roads as summarised in Table 18.2.2.

Table 18.2.2 Haul Road Design Parameters

Design	
Road Cross Section	Running Width – 30 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 Average Recurrence Interval (ARI)
Pavement	200 mm laterite gravel wearing course (to be upgraded by the mining fleet when competent rock available)

18.3 Airstrip

Knight Piésold completed a PFS level design for the non-instrument design site. The airstrip design was completed using a Pilatus PC-12 as the design aircraft, with the key design parameters summarised in Table 18.3.1.

Table 18.3.1 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

18.4 Water Supply

Based on currently available information potential yields from fractured rock aquifer are expected to be low. Work is ongoing to explore the potential for underground water sources to supply and/or supplement the Project requirements.

Project water requirements will be met from a Water Harvesting Dam (WHD) as the primary water collection source. The planned structure will be designed and capable of storing up to 500,000 m^3 of water at the maximum operating level. The design intent of the WHD is that the reservoir is frequently pumped to a Water Storage Dam (WSD) during each wet season, with a view to filling the WSD reservoir to its maximum storage level prior to each dry season. The WHD has a catchment area of 40,593 ha, and when the pond volume is at maximum level the reservoir surface area will be 25 ha.

The WSD is the primary storage pond for clean process water on site and is able to store up to 1,100,000 m^3 of water at its maximum operating level. The WSD has a catchment area of 219 ha, and when the pond volume is at maximum level the reservoir surface area will be 20 ha.

The WHD and WSD embankments will comprise a central low permeability core (Zone A), with outer structural zones (Zone C). A chimney drain (Zone F1) and embankment finger drains are included within the embankment cross section. The upstream embankment face will be lined with textured HDPE geomembrane liner for erosion protection. Topsoil will be spread over the downstream embankment face, which will be vegetated with local grasses. Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow pits to form the low permeability core of the embankment.
- Zone C material shall be won from borrow to form the outer structural zones.
- Zone F1 shall be clean sand / gravel drainage material supplied to a stockpile adjacent to the works.

18.5 Tailings Storage Facility

18.5.1 Design Summary

The tailings storage facility (TSF) has been designed by Knight Piésold with an initial starter cell of 4.8 Mt (18 month capacity) with a final capacity of 32.8 Mt at an average annual throughput rate of 3.0 Mt/y. 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.

The TSF has been designed to ANCOLD guidelines. An ANCOLD consequence category of 'High B' was determined on the basis of a potential PAR (Population at Risk) in the range of '≥10 to <100' and a Severity Level of 'MAJOR'. An ANCOLD environmental spill consequence category of 'Significant' was determined on the basis of a potential PAR in the range of '<1' and a Severity Level of 'Major'.

The TSF will comprise a high-density polyethylene (HDPE) lined cross-valley storage facility formed by multi-zoned earth fill embankments, having a total footprint area (including the basin area) of approximately 114 ha for the Stage 1 TSF increasing to 200 ha for the final TSF. The TSF is designed to accommodate a total of 32.8 Mt of tailings. Downstream raise construction methods will be utilised for all TSF embankment lifts.

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. 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 series of decant towers located within the eastern valley within the TSF basin. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

The TSF design criteria are summarised in Table 18.5.1.

Table 18.5.1 Tailings Storage Facility Design Criteria and Specifications

Design Standards	
TSF Consequence Category	High B
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	 32.8 Mt of dry tails. 4.5 Mt of dry tails (18 months initial capacity).
Production Rate	3.0 Mt/a.
Slurry Characteristics • Beach Slope • Density - Stage 1 - Final	 100H:1V 1.19 t/m^3 (from density modelling). 1.40 t/m^3 (from density modelling).

18.5.2 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 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 would be approximately 1.35 t/m^3.

The TSF incorporates sufficient measures to prevent the loss of tailings (and supernatant liquor) from the facility based on the expected tailings geochemistry.

18.5.3 Monitoring

A total of two 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.

18.5.4 Closure Summary

At the end of the TSF operation, the downstream faces of the embankment 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 and 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.

18.6 Surface Water Management

The management of water is a critical aspect of the design for Lafigué. In order to understand the flow of water around the site, a water balance model was developed. The water balance modelling included the TSF, WHD, WSD and process plant, 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.

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.
- All make-up water requirements can be provided by the WSD reservoir, supplemented by the WHD for design dry conditions. It is necessary that the WHD and WSD are completed early to allow a full wet season for filling prior to commissioning.
- A WSD storage capacity of 1,100,000 m^3 is required to provide sufficient make-up water, supplemented by an abstraction rate of 210 L/s from the WHD.
- A WHD capacity of 500,000 m^3 is required to reduce the risk of shortfalls under design dry conditions.

18.7 Power Supply

The Project will implement a hybrid grid / solar option. The grid connection will be achieved via a 28 km Overhead Power Line (OHPL) from site to the La Société des Energies de Côte d'Ivoire (CI-E) supply system 225 kV bus at Dabakala Substation.

CI-E owns the National Interconnected Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government.

The Plant is estimated to have a connected load of approximately 23.4 MW with a Maximum Demand of 16.6 MW and an expected energy consumption of 133 GWh/y.

The grid supply will be supplemented with a 7 MW solar PV located on the Project site.

18.8 Fuel Supply

A vendor supplied fuel storage and pumping system will be supplied as part of the fuel supply contract. The fuel farm will include an integrated fire control system. A fully bunded storage tank will be allowed to service the process heating requirements.

18.9 Sewage Management

A packaged treatment plant will process daily sewage from up to 200 personnel at the accommodation village site. Effluent will discharge to a leach drain system. A separate, similar treatment plant will be located near the process plant.

18.10 Explosives 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 facility will be fully fenced and will include monitored electronic 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.

18.11 Accommodation

An accommodation camp will be located approximately 2 km east of the plant site and approximately 4 km by road. The accommodation camp facilities will include the following:

- Accommodation units with en-suites for 200 staff.
- Dry mess with food storage and preparation, kitchen and dining facilities.
- Kitchen / dry mess toilets.
- One catering staff change room.
- Recreational facilities including a gymnasium, bar, TV area and store room.
- Laundry building.
- Gatehouse.

A separate 60 bed security barracks will be built and provide housing for Gendarmes when required and will include basic mess facilities. Services such as power, water and sewage handling will be supplied from the process plant.

18.12 Process Plant Facilities

Site buildings will be a mixture of prefabricated steel frame and modular industrial type structures, as well as of blockwork construction where appropriate.

The following buildings have been allowed for:

- Main administration office - blockwork construction.
- Clinic / First Aid and emergency response building - blockwork construction.

- Security gatehouse and change room - blockwork construction.
- Plant meeting and training room - prefabricated modular type building.
- Plant change room and ablutions - prefabricated with toilets / showers.
- Plant offices and control room - blockwork construction.
- Plant kitchen and diner - prefabricated with benches and sink.
- Electrical switchrooms, of prefabricated construction, mounted on plinths for bottom cable entry.
- Reagent stores - two of steel framed with concrete floor.
- Plant warehouse - steel framed with concrete floor.
- Plant workshop - steel framed with concrete floor.
- Electrical workshop - steel framed with concrete floor.

In addition, provision has been made for two laboratories. The first will be located inside the plant security fence and will analyse process plant samples only. The second will be located outside the fence and will be used for exploration and mine grade control samples using fire assay.

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 a general downward trend from 1980 to 2000 where gold traded as low as 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 the gold has remained in a 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's strategy to outsource mining activities to contractors and in all instances ensure that the mining operation has the option to 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 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 Introduction

The following section includes discussion and comment on the environmental studies, permitting and Social or community impact aspects relating to the Lafigué Gold Project.

The Mining Code of Côte d'Ivoire provides in its Article 28 of Chapter II on the Exploitation License that *"the allocation of an Exploitation License must result from a technical and economic feasibility study and must be the subject of an environmental and social impact study (ESIA)"*.

In order to comply with this provision of the Mining Code and in application of the Environmental Code, Endeavour Mining has engaged the services of an environmental consulting company approved by the National Environment Agency (ANDE) – Cabinet ENVAL Côte d'Ivoire to carry out the ESIA of the Lafigué area, located in the Dabakala Department.

These studies began with the exhaustive inventory of the physical, human and social environment of the area covering the Exploration License as well as the future area of the Mining Permit to be requested, considering the standards of the Environmental Code and of the Performance Standards of the International Finance Corporation (IFC).

These studies were carried out covering the two main seasons: the dry season and the rainy season, with consultants experienced in the different fields covered by the study.

The social component of the study was undertaken by Trust International who developed the stakeholder engagement plan in order to identify the actors involved in, and likely to contribute to the future Project.

The ESIA study was carried out in two stages:

- Initial development of field inventories.
- Conducting studies to understand the Project's impacts on the physical and human environment, and develop mitigation strategies.

The study area covered the administrative Department of Hambol in the Centre-East of Côte d'Ivoire, the capital of which is Katiola, with the Departments capital of Dabakala to which the village of Lafigué is attached.

20.2 Baseline Studies

The analysis of the initial state of the environment of the Project allows the development of a baseline record of the physical, biological and socio-economic environment, prior to the development of the Project.

The definition of the baseline level of the environment is based on literature information and on the collection of field data according to the technical parameters of the Project.

The collection of field data was carried out between 16 November 2019 and 16 August 2020 in order to consider the two seasons (dry and rainy seasons) according to the schedule below depending on the environmental parameters:

- Field recognition visit: from 16 to 20 November 2019.
- Project's kick off visit: 19 to 22 February 2020.
- Noise measurement campaign: from 23 to 27 June 2020.
- Air quality analysis campaign: from 23 to 27 June 2020.
- Water analysis campaign: from 10 to 26 May 2020.
- Socio-economic survey: from 20 February to 24 March 24, 2020.
- Public consultations: from 20 February to 24 March 2020 and from 11 to 13 June 2020.
- Hydro-biological study: from 19 to 26 April 2020 and from 10 to 16 August 2020.
- Terrestrial fauna study: from 19 to 26 April 2020 and from 10 to 16 August 2020.
- Bird species study: from 19 to 26 April 2020 and from 10 to 16 August 2020.
- Botanical study: from 19 to 26 April 2020 and from 10 to 16 August 2020.

20.3 ESIA and Environmental Permits

The environmental and social impact study (ESIA) for the Project license in the Lafigué area will involve several stages from the development of inventories to the issuance of the environmental decree of exploitation.

After collecting field data from January to the end of August 2020, Endeavour provided the resultant data to ENVAL in September 2020, along with information on proposed infrastructure. ENVAL's goal was to assess the impacts of the future mining activity on the physical and human environment.

A Draft ESIA was developed and submitted for review to Endeavour and to the National Environment Agency (ANDE) on 5 October 2020.

A public survey was held in the Dabakala Department during the second half of November 2020, with a large consultation open to elected officials, customary authorities, administrative authorities, and the local populations concerned. This public survey was held over a ten-days period and was finalised with an official report giving the opinions of the populations on the Project.

During this public survey, the Prefect of the Department of Dabakala appointed an investigating commissioner who is responsible for collecting the opinions and observations from the surrounding populations during the period. The commissioner's role is to draft the public survey report which will officialise the population's opinion on the Project.

A major technical review meeting for the ESIA was held in Abidjan in January 2021 to present to an inter-ministerial commission the issues of the ESIA and the measures put forward to take into account the environmental, social, legal, and regulatory provisions in effect. This meeting, organised in the presence of the experts from the different ministries concerned by the ESIA, will lead to a recommendation report, and ultimately to the further approval of the ESIA. At the end of this technical review, an environmental authorisation order will be signed by the Minister of the Environment.

This environmental authorization order, granted to Endeavour, will be attached to the Mining License application file for the Project, which is currently planned to be submitted at the beginning of February 2021, and can be presented to the Ministry in charge of Mines to obtain the Lafigué Exploitation Permit.

20.4 Social and Environmental Management Plan

The Environmental and Social Management Plan (ESMP) allows the implementation and monitoring of ESIA measures to remove, reduce and possibly compensate for the consequences of the Project on the environment.

It reflects Endeavour's commitment to universal precautionary principles and constitutes proof of the consideration of the demands from the Ivorian authorities and international funders.

For the environmental aspects, the ESMPs are as follows:

- Air quality management plan.
- Biodiversity management plan.
- Noise management plan.
- Water management plan.

- Waste management plan.
- Traffic and transport management plan.
- Hazardous substances management plan.
- Prevention and intervention plan in case of emergency.

20.4.1 Biodiversity Management Plan

With regard to the management of biodiversity, the plan aims to protect natural environments, terrestrial and aquatic flora and fauna, to limit disturbances of natural habitat resulting from the activities of the Project and to protect endangered and protected species, critical habitats and areas housing certain plant species listed in the IUCN Red List as threatened nationally and internationally.

Biodiversity management must consider bypassing sensitive environments and sites, minimising the physical footprint of the Project in sensitive and natural habitats, and promote works and activities in areas with a lower ecological value.

20.4.2 Water Management Plan

The water management plan will help to limit the impact on surface water, soil and groundwater (anti-erosion measures and good practices, management of hazardous products, effluents and waste, etc.) by mining activities and, to limit earthworks activity to the dry season period and in hydromorphic zones.

The water management plan will cover the protection of soil, surface water and groundwater and will include measures to monitor water consumption, reduce the impact of drainage water coming from working areas. This plan will also deal with waste water and the appropriate method of treatment of the effluent before releasing into the environment.

The water management plan will cover spill prevention measures and treatment measures in the event of a spill to limit the potential impacts on soil and water (especially in case of hydrocarbon spill).

The integrated water management plan will aim at reducing water consumption of the site, develop a wastewater treatment system, with the goal to monitor the quality of the water discharged to ensure compliance with applicable standards.

20.4.3 Hazardous Substances Management Plan

The hazardous substances management plan will:

- help prevent the infiltration of pollutants into surface and groundwater
- promote the substitution of hazardous products by an equivalent but less dangerous products as far as possible
- promote the installation of refuelling station with an impoundment system and spill recovery
- ensure that a proper maintenance system of fuel tanks is in place
- avoid the installation of underground tank for the storage of hydrocarbons or other dangerous products
- ensure that a regular inspection schedule of hazardous products network (supply pipes, connection pipes), is done by qualified employees and that a log of inspection reports is available
- ensure that an adequate training program is offered to employees, in order to maintain good practices in terms of storage and handling of hazardous products in order to prevent the risks associated.

20.4.4 Air Quality Management Plan

The air quality management plan covers the management of dust induced by mining activities and construction work and the management of atmospheric emissions from potential diesel power generators.

The air quality management plan involves the establishment of a program to measure the air quality in the initial state during the pre-construction and construction phases, in order to obtain the baseline data of the air quality in the area over a period of at least one year (to characterise the potential seasonal variability).

To prevent excessive dust emission, mitigation measures will be put in place, such as use of dust suppression systems at the processing plant, watering the haulage roads, limiting the trucks speed at sensitive areas (to avoid excessive dust emissions in those areas), and ensure that all equipment, vehicles and machinery are kept in good working order.

20.4.5 Noise Management Plan

The purpose of the noise management plan is to integrate into the ESMP the noise management that is part of the HSE regulations and guidelines. It concerns the noise generated by all construction and operation activities of the Project, in particular at the places closest to the villages.

The plan will identify the Project activities and places in which noise-related impacts are to be anticipated, and identify what are the sensitive receptors and areas where noise must be particularly controlled. Noise management will also involve keeping equipment in good condition and will define a clear mechanism for handling complaints about noise.

20.4.6 Waste Management Plan

The waste management plan is an operational process to be implemented and continuously improved on the basis of experience feedback. This continuous improvement will not be limited to the evaluation of waste treatment and means of disposal, but will also focus on the use of technical solutions for reducing waste generation.

The waste management system will be updated in order to identify the consumption of products, to have traceability of the waste disposal process and to identify overconsumption. The plan will aim to quantify all the waste will be generated, disposed and/or eliminated (type and volumes), and will establish targets for reducing the quantities of waste production, that is accordance with health and safety rules and plan for the management of hazardous substances and the prevention of spills.

Dedicated waste storage sites will be built at designed areas to prevent liquid waste from seeping into the ground, these sites will be fenced, well maintained, clean, with waste separated by type and risk classification, in order to limit pollution risks, proliferation of vermin.

When waste is sent off-site, suitable transport vehicles will be used (possibly using a service provider) in order to comply with the regulation and ensure the load is safe and correctly labelled.

20.4.7 Traffic and Transport Management Plan

The transport management plan describes the principles of the procedure for road and river transport, during the construction and operation phase and relates to the transport of construction materials and equipment, transport of personnel, and transport of consumables.

The standards applicable to the transport of freight and personnel will be respected, included maximum load, compliance with speed limits on roads and tracks, special measure in place for oversized loads (escort vehicles leading and tailing the convoy, coordination with local authorities to agree on the routes to be taken and when).

The personnel will be trained on a regular basis in road safety, and traffic code in the country.

In the event of a traffic accident involving one of the Project vehicles, the manager responsible for supervising the activity will inform the emergency services as soon as possible. It is expected that outside of public roads, the emergency vehicle dedicated to the Project will be mobilised to the accident area, while on the public road the public emergency services will be involved. The details of the incident will be recorded in a specific incident report.

20.4.8 Hazardous Substances Management and Spill Response Plan

The hazardous substances management and spill response plan aims at reducing the impacts of any accidental spills of hazardous products into the environment, in the event of a leak or breach from the storage area.

A spill management plan and a fire prevention management plan will be defined. These plans will organise a systematic, rapid and effective response to any type of emergency, fire, explosion, accident or spill of water contaminated by hydrocarbons or any other hazardous product.

These management plans will define how to contain as fast as possible any spill and rapid clean-up of any contaminated area. The plan will define the roles and responsibilities of employees and contractors that are part of emergency response team.

Part of the management plan will include personnel training and awareness, and will specify the requirements in terms of periodic exercises, as well as frequency of checks and maintenance activities of emergency response team.

20.4.9 Emergency Prevention and Response Plan

The emergency prevention and response plan will define the response and communication procedures to be followed in the event of an emergency or natural disaster. It will outline the on-site intervention process related to both construction and operating activities (road accident, explosion, fires, medical emergencies, etc.).

It is designed to reduce employee exposure to risks and injuries and limit potential impacts on the environment and the community in emergency situations.

The prevention plan will include all possible emergency situations such as fires or explosions, medical emergencies, the transport of dangerous products, climatic phenomena, natural disasters, social and political tensions.

This plan will also address community emergency preparedness and will be disclosed in a culturally appropriate manner to all communities in the social area of influence of the Project.

20.5 Social Management Plan

The specific social management plans proposed for the Project are the following:

- Livelihoods restoration plan.
- Working conditions and information management plan.
- Local hiring and procurement plan.
- Stakeholder engagement plan.
- Grievance management plan.
- Health and safety management plan.
- Local assets management plan.

20.5.1 Livelihoods Restoration Plan

The Project is likely to generate some economic displacement rather than physical displacement of camps or villages. Some land is used as a means of subsistence by local communities as agricultural land, pastures or for hunting and gathering of forest products.

The risks associated with economic displacement are mainly loss of revenue and impoverishment, social distancing due to loss of cultural identity and changes in family structure.

In this context, a Livelihood Restoration Plan will be developed to ensure the restoration of the livelihoods of displaced households. The plan will include land and crop compensation and additional measures necessary to enable the restoration of the lost economic activity.

20.5.2 Working Conditions and Information Management Plan

This plan will detail the measures put in place to ensure the working conditions are in accordance with local regulations and international standards. The objective of this plan is to ensure fair treatment of workers, to fight against discrimination, and to protect the workers against the use of forced and child labour, and promote healthy and safe working and housing conditions for workers.

The plan guarantees good working condition for the employees and the subcontracting companies, by including in the terms and conditions the standards to be respected and by providing for corrective measures in the event of non-compliance with the standards in place.

20.5.3 Local Hiring and Procurement Plan

This plan refers to the principles of corporate social responsibility to which Endeavour adheres and intends to contribute to local sustainable development through its activities.

The objectives of this plan are to optimize the positive social impacts of the Project through the implementation of a local procurement policy, a preferential hiring procedure in local communities for unskilled jobs and a preference for the use of local businesses. Endeavour's hiring policy will promote local employment and local or national procurement. Monitoring indicators will be developed in this regard.

20.5.4 Stakeholder Engagement Plan

The stakeholder engagement plan was developed as part of the Project by ENVAL and by Trust International, in strict compliance with Endeavour's policy on community engagement.

This plan describes the consultations and the results of the consultations held as part of the Project's ESIA. It is used to structure the communication and consultation activities carried out and to plan the consultations to be carried out within the framework of the Project. Its objective is to ensure a continuous and transparent dialogue with the stakeholders of the Project during its various phases of development.

The main objectives are to promote and maintain open and respectful dialogue between the stakeholders and the Project, to identify the stakeholders, their interests, concerns and influences in relation to Project activities.

The goal of the plan is also to provide to the stakeholders information on the development of the Project and access to information and, depending on the potential impacts of the Project, to give stakeholders the opportunity to communicate their opinions and concerns through consultations and then preventing conflicts and developing a good relationship with the various stakeholders.

20.5.5 Grievance Management Plan

The grievance management plan is a good international practice in order to ensure that projects take into account the population's complaints, and that the population has easy and free access to file a complaint.

This plan includes information on the means provided to the population to file their complaints and on the procedures and deadlines for their follow-up and resolution. Good communication to the population, on a regular basis, will ensure its functionality.

20.5.6 Health and Safety Management Plan

This plan will include the procedures, infrastructure and means implemented to ensure the hygiene, health and safety of workers and the local population during the construction and operation phase.

The health and safety management system ensures the prevention of accidents and incidents, and includes the assessment of the various health and safety risks to which workers are exposed according to the type of activity.

This plan will also deal with the impact on local communities of the risks associated with the addition of a relatively significant construction workforce over a short period of time (construction activities, exposure to diseases from construction workers, increased traffic). The health and safety management plan will include an analysis of these different risks and detail the mitigation and management measures planned.

The Project is also committed to promoting the prevention of communicable diseases such as HIV and malaria.

20.5.7 Local Assets Management Plan

This plan will be developed with the objective of protecting sites and objects of international or local cultural importance in accordance with national regulations and in accordance with Endeavour's cultural heritage management policy.

The plan will include a geo-referenced inventory and mapping of cultural heritage sites and the procedures and measures to be implemented to ensure the protection or preservation of these sites and objects of cultural importance.

20.6 Potential Impacts and Mitigation Measures

Several sectors will be impacted by the Project and a series of measures to mitigate the impacts will be implemented.

20.6.1 Control of Air Emissions and Dust

The construction and operation activities will have a significant impact on air quality through an increase in the dust emissions and the concentration of atmospheric pollutants (vehicle exhaust gases) on the Project site. Farmers surrounding the site could thus be affected by dust and fumes from machinery and mining activities.

This will involve a strict control of blasting operations, limiting the speed and density of traffic on unsealed tracks, promote the use of water truck to eliminate dust on the access tracks to the site and haulage track.

20.6.2 Noise and Vibration Control

Mining work may generate noise from machinery, vehicles during earthworks and blasting operations. The noise emissions could lead to inconvenience and noise pollution for the neighbouring populations, especially in Lafigué village.

Better control of blasting operations and the provision of equipment and installations with noise suppression systems will be the appropriate means to limit noise pollution for the Project area. Personnel will be positioned at sensitive areas of the site prior to the blasting activities and a building survey will be undertaken prior to the start of operation to monitor potential vibration damage to nearby structures.

20.6.3 Conservation of Water Quality and the Aquatic Environment

Surface water and groundwater will be impacted by the Lafigué mining operations and appropriate measures will be put in place to mitigate the impacts.

The extraction of ore from the pits and its processing at the process plant can negatively impact surface water resources. Water from the tailings pond, accidental oil or chemical spills and muddy run-off water can create disturbances to the hydrographic network and surface water.

Similarly, mine water, waste water, and the drawdown of water tables for dewatering are important factors that can degrade groundwater.

Practical measures will be taken to address these risks of pollution of surface water and groundwater. Most of the earthworks will be done during the dry season. Wastewater will be treated and recycled in appropriate structures. Process water will not be released into the environment. The tailings pond will be sealed with high density polyethylene (HDPE) liner to prevent any infiltration of tailings water into the ground.

20.6.4 Impact on Flora and Fauna

The earthworks and mining activities will impact the vegetation cover and the flora and fauna. The trees that need to be cut down will be made available to the local populations for their usage.

The extractive activities of open-pit mining will lead to a significant modification of the landscape with a significant visual impact and modification of the relief. The plant cover will be modified and over time will give rise to an industrialised environment, but a progressive rehabilitation plan during the entire life of the Project will be put in place, with revegetation of the mined-out areas.

The noise and vibrations of the machines, the blasting and crushing activities are factors of acceleration in the destruction of wildlife habitats and will cause an added disturbance and therefore a forced migration of the fauna present on site.

Measures to limit these impacts will involve the strict control of clearing sites during construction, and only the necessary sites will be cleared. The top soil will be stripped and stored for re-use during future site rehabilitation work.

Hunting of animals will be strictly prohibited on the site as well as all actions tending to disturb the life of the animals.

20.6.5 Employee Health, Safety and Security

During the construction and operation, employees and communities will be exposed to various disruptions with the significant in-migration of workers coming from various locations, increased noise, increased atmospheric impacts, the appearance of the COVID-19 pandemic and the strong presence of illegal artisanal miners. These factors can affect the health, safety and security of the workers.

Appropriate measures will have to be taken to deal with these events. Written procedures for occupational safety measures will be essential to guarantee a safe and healthy working environment.

The construction of a medical clinic with the appropriate medical equipment and trained personnel will provide the staff with good medical care and urgent treatment of trauma cases when needed.

The fencing of the site's facilities, and the presence of a private security company on top of gendarmes are urgent actions to be taken at the start of the Project, after the eviction of the artisanal miners with the help of gendarmes.

20.6.6 Cyanide Control

Cyanide is an important chemical in the gold processing industry. The supply, transport and use of cyanide at mine sites are strongly regulated by the international cyanide code to comply with international good practices

The risks of accidents linked to cyanide (intoxication, allergy, burns by inhalation, or ingestion) must be reduced by a proper training program and strict control of its handling, throughout the implementation of rigorous procedures for transportation and use in the operation.

20.6.7 Tailings Storage

The tailings storage facility (TSF) will be sized and constructed in stages to contain all tailings for the entire duration of operation according to international standards. The important point is the protection against infiltration by lining the facility with HDPE liner and a dam control system with piezometers.

An audit control system by independent engineers and a detailed closure plan will be put in place to guarantee the long-term stability of the structure. Refer to Section 7.1 for more details on the TSF, including design, monitoring and closure.

20.6.8 Identification and Rectification of Adverse Socio-Economic Impacts

The future Project is an opportunity that will bring positive benefits to the Dabakala Department and its implementation must take into account all identified negative impacts that may affect its success.

Issues such as the massive arrival of labour from other regions of the country, access to land, and traffic rights, must be addressed prior to the start of the construction of the Project through an awareness campaign and well-established administrative procedures to reduce any negative impact.

As a social partner, Endeavour must maintain a permanent dialogue with the socio-economic actors of the region and all the stakeholders involved in the Project in order to manage the concerns and questions as they arise.

20.6.9 Social and Community Impacts

The populations of the Project area are mainly native Tagbana, Djimini, Djamala, Mangoro and Malinké who coexist with communities of non-native nationals, essentially coming from countries such as Burkina Faso, Mali, Guinea, Benin and with a strong presence of illegal miners of various origins. The populations are grouped into eight villages around the Project area.

The Project covers the area between the Bonieredougou area and the capital of the Dabakala Department. The main economic activities are cashew cultivation, food crops and illegal gold mining.

The exploration work carried out by Endeavour in the area for several years has helped develop good relations with the communities living in the Project area. The populations welcomed the Project and hope it will result in benefits in local infrastructure, job creation and that local development will change the face of their villages and their region.

The mine site is approximately 3 km from the main village of Lafigué which will not need any resettlement of houses or public infrastructures. No other village will be impacted under this Project. The lands will be subject to customary rights, in agreement with the customary authorities, landowners, administrative authorities, executives and elected officials according to a protocol negotiated by Endeavour with the communities. The farms which will be affected will be compensated, according to the official scales of the Ministry of Agriculture.

In terms of employment, unskilled jobs will primarily be reserved for local populations. A robust and proactive training plan should be put in place to allow young people from the local communities to join teams of skilled workers or specialised agents during construction and mining operations.

An infrastructure development program will strengthen health infrastructure and increase access to drinking water in the locality.

Information, education and awareness campaigns carried out upstream of the Project will help reduce the risk of disease (HIV and malaria).

The solid expertise of Endeavour and of the Project team in the construction of the process plant are important values and resources that will ensure greater integration of young people from the region into the Project during both the construction phase and mining operations.

21.0 CAPITAL AND OPERATING COSTS

21.1 Operating Costs

The operating cost estimates have been prepared by contributors as follows:

- Lycopodium - Processing costs.
- Endeavour/Lycopodium - Site general and administration costs.
- Endeavour/Snowden - Mining costs

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).

The operating cost estimates are presented at summary level with detailed operating cost estimates available on request. Estimates are based on the design throughput rate of 3.0 Mtpa fresh ore.

21.1.1 Mining Operating Costs

Mining operating costs were built-up on the assumption that all mining would be under a Contractor Mining basis. Endeavour calculated mining costs based on haulage paths provided by Snowden. Table 21.1.1 provides the Life of Mine Operating Cost breakdown by activity to include ore and waste splits.

Table 21.1.1 Life of Mine Operating Costs (US$ million)

Cost Area (US$M)	Ore	Waste	Total	% Total
Load	15.9	92.2	108.1	10.2%
Haul	39.0	371.6	410.6	38.6%
Drill and blast	30.0	236.4	266.4	25.0%
Grade control	41.8		41.8	3.9%
LT stockpile	26.0		26.0	2.4%
Mine admin.	6.2	64.7	70.9	6.7%
Ancillary	19.8	108.4	128.2	12.1%
RoM	11.9		11.9	1.1%
Total	**190.7**	**873.2**	**1,063.9**	

21.1.2 Process Plant Operating Costs

The process plant operating costs have been compiled with input from Endeavour and from a variety of sources, including the following:

- Labour pay rates and manning as advised by Endeavour.
- Power cost as advised by ECG Engineering.
- Consumable prices from supplier budget quotations, Endeavour advice or the Lycopodium database.
- Modelling by Orway Mineral Consultants (OMC) for crushing and grinding energy and consumables, based on Lafigué ore characteristics determined in the metallurgical testwork or benchmarked ore characteristics from the OMC database.
- Reagent consumptions and gold extractions based on metallurgical testwork results.
- First principle estimates based on typical operating data / standard industry practice.

The following exchange rates have been used for the preparation of the operating cost estimate:

- USD1.00 = AUD1.408 (Australian Dollar).
- USD1.00 = EUR0.845 € (Euro).

The process plant operating cost estimate by cost centre is summarised in Table 21.1.2.

The relative proportions of each operating cost centre for the processing and process plant (processing + G&A) operating costs are shown in Figure 21.1.1.

The 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.

Table 21.1.2 Process Plant Operating Cost Estimate Summary (US$, 4Q20, ±25%)

COST CENTRE	Fresh		Oxide / Transition		LOM Blend	
Proportion of LOM	**92%**		**8%**		**100%**	
Plant Feed t/y	**3,000,000**		**3,500,000**		**3,039,614**	
	US$/y	**US$/t**	**US$/y**	**US$/t**	**US$/y**	**US$/t**
Power	9,845,239	3.28	6,292,359	1.80	9,563,750	3.15
Operating Consumables	9,224,661	3.07	9,745,105	2.78	9,265,895	3.05
Maintenance	3,473,286	1.16	2,635,940	0.75	3,406,944	1.12
Contract Laboratory	1,342,574	0.45	1,342,574	0.38	1,342,574	0.44
Process & Maintenance Labour	6,134,550	2.04	6,134,550	1.75	6,134,550	2.02
Total Processing	**30,020,309**	**10.01**	**26,150,529**	**7.47**	**29,713,713**	**9.78**
Fixed Component US$/y	13,394,496		11,936,603		13,278,990	
Variable Component US$/t		5.54		4.06		5.41
Administration Labour	9,748,187	3.25	9,748,187	2.79	9,748,187	3.21
General & Administration Costs	8,828,943	2.94	8,828,943	2.52	8,828,943	2.90
Total General & Administration (G&A)	**18,577,130**	**6.19**	**18,577,130**	**5.31**	**18,577,130**	**6.11**
Total Process Plant	**48,597,439**	**16.20**	**44,727,658**	**12.78**	**48,290,842**	**15.89**
Fixed Component US$/y	31,971,626		30,513,732		31,856,119	
Variable Component US$/t		5.54		4.06		5.41

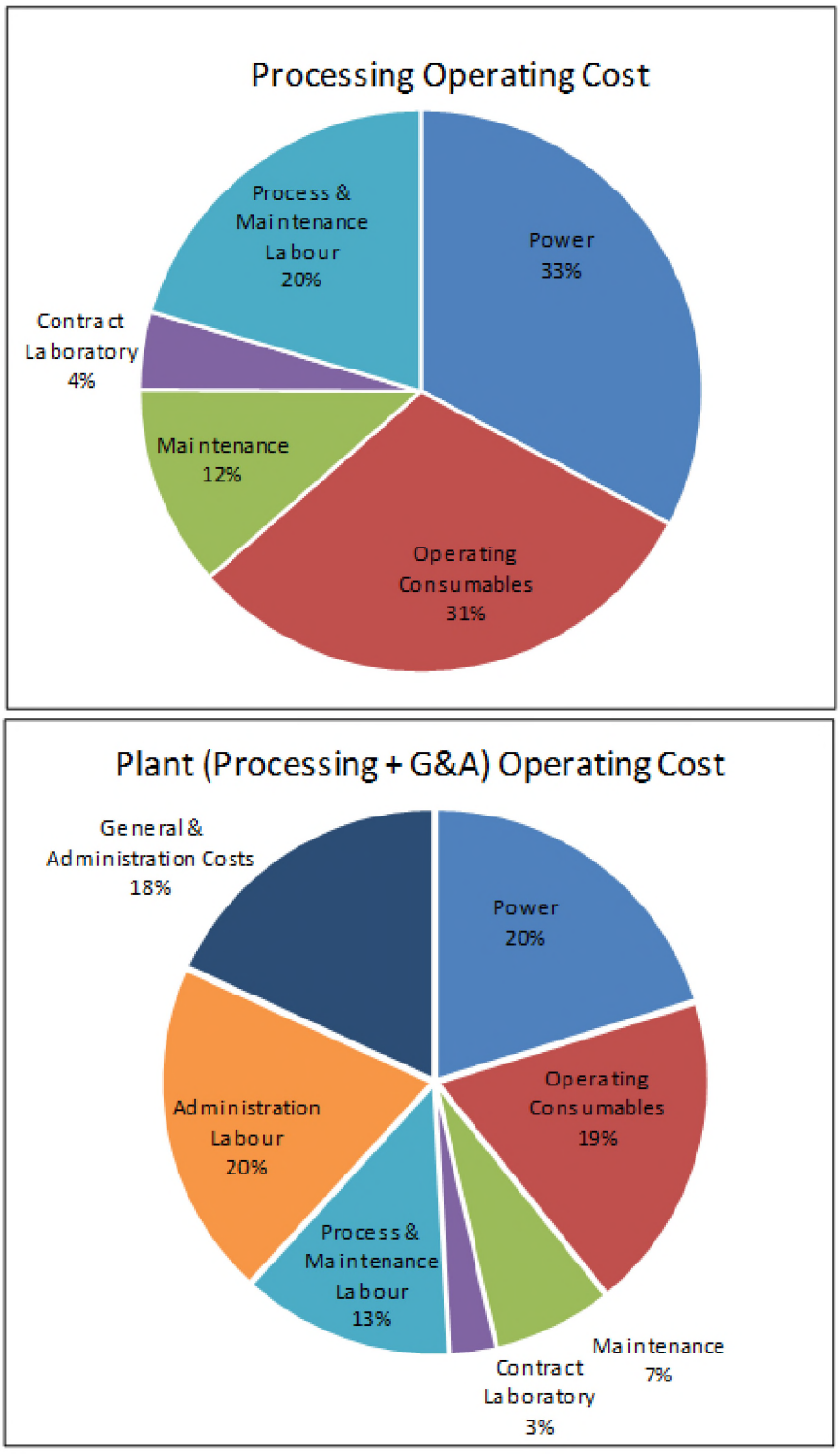
Exclusions: All Mining Costs except MSA power and water, grade control assay and selected mining personnel G&A costs

Summary Battery Limits

The battery limits for the processing operating costs are as follows:

- Ore in the ROM bin.
- Tailings discharge from the tails pipeline to the tailings storage facility (TSF).
- Gold bullion in plant goldroom safe.

Figure 21.1.1 Lafigué Operating Cost Breakdown



Qualifications

The process plant 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 the capital cost estimate).
- Gold refining and bullion transport and in-transit security of gold from site (included in the general overheads operating cost).
- Tailings storage, rehabilitation or closure costs (included in the capital cost estimate).
- Water harvest dam and water storage dam maintenance (included in sustaining capital).

The process plant 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 in the plant laboratory as well as the normal process plant 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).
- The power cost estimate has been based on hybrid grid / solar power at a unit cost of US$0.090/kWh.
- The plant maintenance cost allowance has been factored from the capital supply cost using factors from the Lycopodium
- The labour rates, manning levels and rosters used to determine the labour operating cost estimate were agreed with Endeavour and are based on other Endeavour operations and benchmarking within Côte d'Ivoire.

21.2 Capital Cost Estimate

21.2.1 Overview and Summary

The Project capital cost estimate (CCE) was compiled by Lycopodium and is presented here in summary format. Additional detail for items costed is available on request. The CCE reflects the Project scope as described in this report. Inputs have been provided 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, Owners costs and HV power supply.

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 based on rates obtained at the end of January 2020:

- 1 AUD = 0.7100 USD
- 1 EUR = 1.1833 USD
- 1 GBP = 1.3097 USD
- 1 ZAR = 0.061 USD

A summary of Project capital costs is presented in Table 1.14.1.

Table 21.2.1 Capital Cost Estimate Summary (US$, 3Q20, +20/-10%)

Main Area	Capital (US$M)
Treatment Plant	74.4
Reagents & Plant Services	14.4
Infrastructure	70.4
Mining	50.3
Construction Distributables	23.5
Subtotal	**233.0**
Management Costs	24.5
Owners Project Costs	34.2
Working Capital	4.8
Subtotal	**296.5**
*Contingency	37.7
DFS costs	3.8
Project Total	**338.0**

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

21.2.2 Qualifications

The estimate is subject to the following qualifications:

- The capital cost estimate has been based on project implementation using an EPCM contracting strategy.
- Mining contractor mobilisation and establishment costs provided by Endeavour.
- Overhead power line and grid connection cost provided by ECG.
- An allowance for communications capital costs has been included - fibre installation, tower, etc.
- 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.
- A temporary 200 bed construction camp will be provided for the project. Owner's team, EPCM personnel and construction management will be accommodated in the mine village.
- 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.

- Owner's 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.
- 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 overall contingency for the Project is 12.2%.

21.2.3 Exclusions

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.

21.2.4 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.

21.2.5 Preproduction Costs

Preproduction costs that include first fills, opening stocks, preproduction labour and vendor representative costs have been included in the estimate.

21.2.6 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. Sustaining capital costs have not been quantified at this study stage.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

A financial model has been built to include the relevant PFS results in order to estimate and evaluate project cash flows and economic viability. The evaluation method considers 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 Basis for Selected Evaluation Throughput

The PFS design criteria, equipment list and the mine plan have been based on a throughput of 3.0 Mtpa of fresh ore. At the request of Endeavour, the 100th percentile (hardest) comminution parameters were used for the modelling and selection of crushing and grinding equipment to provide a conservative comminution circuit design. Grinding circuit design typically utilises the 85th percentile comminution parameters, so a high level exercise was undertaken during the PFS to determine the throughput achievable utilising the selected major comminution equipment with the relaxed comminution parameters. Additionally, a high level assessment of all major plant equipment (e.g. pumps, motors, etc.) was carried out and equipment that was required to be upgraded to support the higher throughput rate was identified.

The high level evaluation concluded that:

- the major comminution equipment (crusher, mills) is capable of processing ore at 3.4 Mtpa if the 85th percentile comminution parameters are utilised
- the downstream process plant may be capable of processing 3.4 Mtpa if selected equipment is upgraded to maintain the required engineering design basis and design margins.

Endeavour has assumed that additional capacity above the warranted design nameplate capacity of 3.0 Mtpa on primary ore will be available consistently without upgrading of any equipment selected for the 3.0 Mtpa plant.

On this basis Endeavour has developed an alternate mine plan, with annual throughput ranging from 3.3 Mtpa to 3.5 Mtpa, with a blend of oxide to fresh ores.

This mine plan and throughput rate has been used as the basis for the economic evaluation included in this section. The following Figure 22.2.1 and Figure 22.2.2 show the alternate mine plan and alternate production schedule of this scenario.

Figure 22.2.1 Alternate Mine Plan

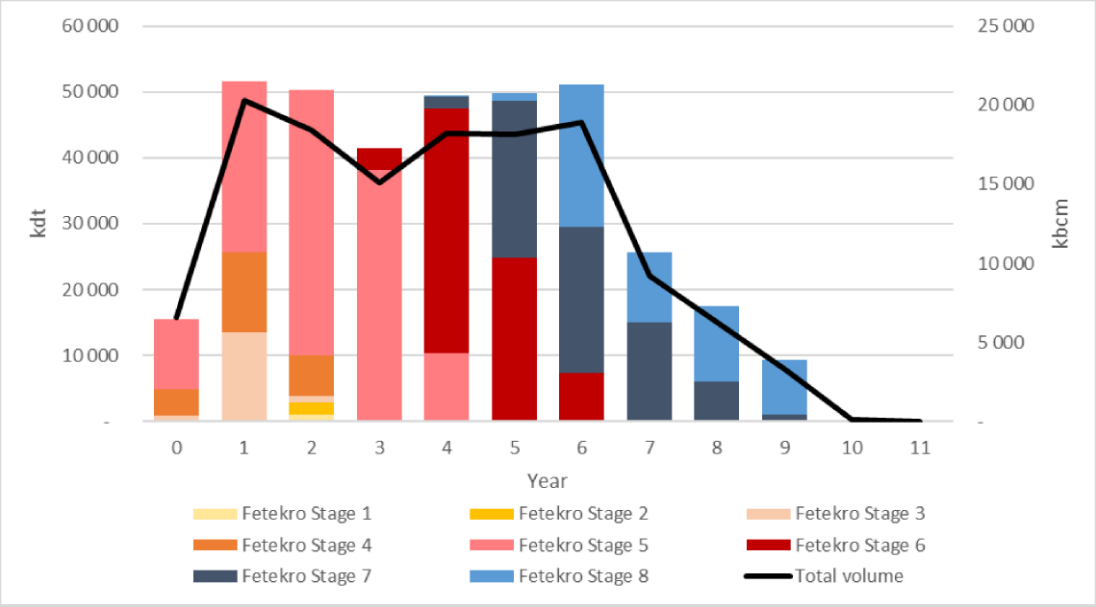
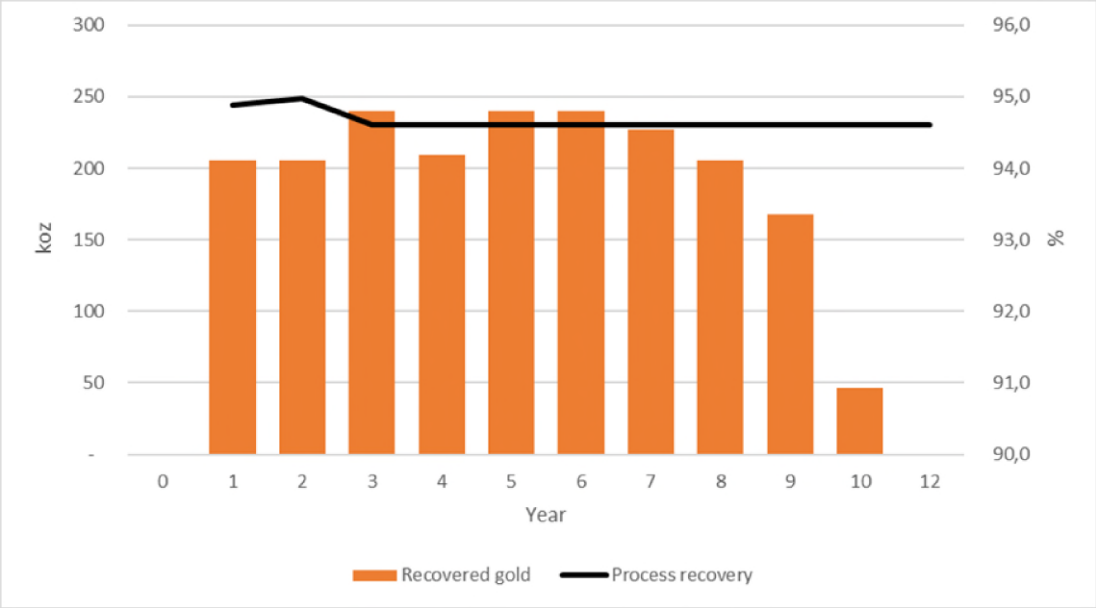


Figure 22.2.2 Alternate Production Schedule



22.3 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 $663 million and the IRR is 38%. The life of mine average cash cost per ounce is $684.

Project gold production is 1,985 koz over the life of the project using a metallurgical recovery of 94.7%.

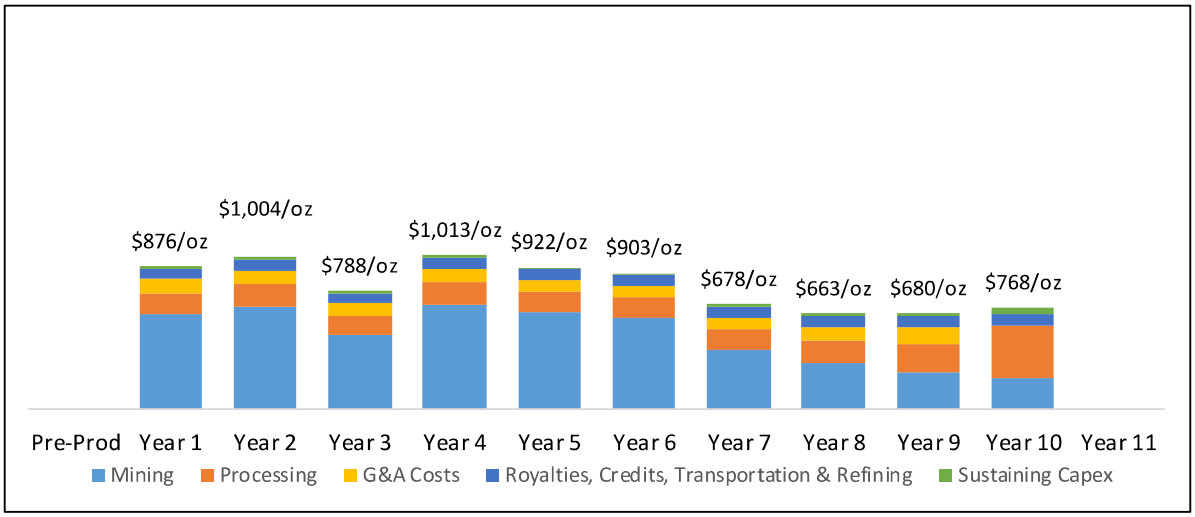
Table 22.3.1 presents a summary of the production information on which the cash flow model is based and the key project financial measures.

Figure 22.3.1 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 AISC per ounce is $838.

Table 22.3.1 Summary of Financial Analysis Results

Financial Summary	Units	Value
LOM tonnage ore processed	kt	31,897
LOM strip ratio	w:o	10.34
LOM feed grade processed	Au g/t	2.05
LOM gold recovery	%	94.67%
LOM gold production	oz	1,985,443
Upfront capital cost	US$ (M)	338
Pre-Tax:		
Internal rate of return	%	38%
NAV - 5% discount rate	US$ (M)	663
Post Tax:		
Internal rate of return	%	33%
Payback period (Pre-Tax)	Years	2.58
Payback period (Post-Tax)	Years	2.66
NAV - 5% discount rate	US$ (M)	479

Figure 22.3.1 AISC Breakdown ($/oz sold)



22.4 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 Lafigué 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.
- Côte d'Ivoire government royalties. The cash flow analysis excludes any effects due to inflation.

- A gold price of $1,500/oz.
- Construction completed over an 18 month period shown in the 'Pre-production' period.
- The financial assessment has been undertaken in United States Dollars (US$).

22.4.1 Basis of Estimate

- Annual tonnage, strip ratio and head grade have been based on the mining schedule as discussed in Section 16.
- The mining schedule and contractor operating cost estimate used as basis for the annualised production and cost model is presented in Section 16 and Section 22.
- The processing and administration costs are based on the operating cost estimate discussed in Section 21.
- The overall gold recovery figures utilise a flat 94.7% recovery based on the metallurgical testwork. Metallurgical recovery is discussed in Section 13.
- The capital cost estimate used as the basis for the cash flow model is discussed in Section 21. Project capital expenditure is split over two years.

22.4.2 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 5% of the material moved does not require blasting, 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 Lafigué. project was very similar.
- Crusher feed costs and stockpile rehandle costs are averaged and applied to 100% of ore tonnes.

22.4.3 Sustaining Capital

A nominal allowance has been made for sustaining capital to include for:

- Stage 2 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 mining fleet replacement will not be required with contract mining operations.

A provision for waste capitalisation has been made on a $/t basis for waste material that is in excess of the life of mine average strip ratio in any given quarterly period.

22.4.4 Depreciation

Provision has been made for depreciation using a straight-line method for a period of five years for all capital.

22.4.5 Company Tax

Corporate tax rate of 25% of gross profit has been used.

22.4.6 Refining Costs

- A typical refinery gold payable rate of 99.95% and an additional $3.00/oz freight plus insurance charge has been used based on historical costs.

22.4.7 Silver Credits

- No silver credits have been included in the financial model.

22.4.8 Royalties

- Gold production is subject to a government royalty which is based on a percentage of the spot gold price (4.0% based on $1,500/oz) increasing with price bracket shown in Table 22.4.1, plus 0.5% of revenue paid to a community development fund.

Table 22.4.1 Côte d'Ivoire Sliding-Scale Royalty

Gold Price Range	Royalty %
$0 - $1,000	3.0%
$1,000 - $1,300	3.5%
$1,300 - $1,600	4.0%
$1,600 +	5.0%

22.4.9 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 capital 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.

22.4.10 Closure Costs

- In the economic model, demolition and closure activities occur after mining and processing activities have ceased, with an allowance of US$35M.

22.4.11 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.

22.5 Cash Flow Analysis

The following table provides the details of the cash flows anticipated over the entire life of the mine.

Table 22.5.1 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	361,808,020	15,544,655	51,608,578	50,242,545	41,411,686	49,287,222	49,775,532	51,166,016	25,678,142	17,549,335	9,299,359	244,951	--	--
Waste	t	329,910,642	15,210,418	47,076,900	46,316,533	37,522,007	46,844,878	45,669,475	48,008,903	21,145,223	14,498,538	7,495,709	122,058	--	--
Ore	t	31,897,379	334,237	4,531,678	3,926,012	3,889,679	2,442,344	4,106,057	3,157,113	4,532,920	3,050,797	1,803,650	122,893	--	--
Grade	g/t	2.05	1.53	1.76	1.58	2.13	2.56	2.06	2.49	2.16	1.64	2.48	3.1	--	--
Contained Gold	**oz**	**2,097,262**	**16,489**	**256,753**	**199,499**	**266,877**	**201,252**	**271,845**	**253,047**	**315,025**	**160,633**	**143,593**	**12,249**	**--**	**--**
Strip Ratio	**Waste:Ore**	**10.34**	**45.51**	**10.39**	**11.8**	**9.65**	**19.18**	**11.12**	**15.21**	**4.66**	**4.75**	**4.16**	**0.99**	**0**	**0**
Processing Total															
Ore Processed	t	31,897,379	--	3,423,582	3,529,939	3,300,000	3,318,541	3,300,000	3,300,000	3,300,000	3,300,000	3,302,053	1,823,264	--	--
Ore Grade	g/t	2.05	--	1.96	1.9	2.39	2.07	2.39	2.39	2.26	2.04	1.67	0.84	--	--
Contained Gold	oz	2,097,262	--	216,063	215,856	253,700	221,207	253,700	253,700	239,668	216,702	177,357	49,309	--	--
Recovery	%	95%	0%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	0%	0%
Recovered Gold	**oz**	**1,985,443**	**--**	**205,000**	**205,000**	**240,000**	**209,287**	**240,000**	**240,000**	**226,726**	**205,000**	**167,782**	**46,648**	**--**	**--**
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**	**$2,977**	**--**	**$307**	**$307**	**$360**	**$314**	**$360**	**$360**	**$340**	**$307**	**$252**	**$70**	**--**	**--**
Less: Cash Operating Costs															
Mining	US$M	-$1,023	--	-$127	-$138	-$117	-$144	-$154	-$145	-$87	-$62	-$40	-$9	--	--
Processing	US$M	-$295	--	-$30	-$31	-$31	-$31	-$31	-$31	-$31	-$31	-$31	-$16	--	--
General & Administrative	US$M	-$167	--	-$19	-$19	-$19	-$19	-$19	-$19	-$19	-$19	-$19	--	--	--
Other (royalties, inventory movement, etc.)	US$M	-$7	--	-$1	$1	-$18	$49	-$3	$26	-$12	-$21	-$20	-$8	--	--
Operating EBITDA	**US$M**	**$1,484**	**--**	**$131**	**$121**	**$175**	**$169**	**$153**	**$191**	**$191**	**$175**	**$142**	**$36**	**--**	**--**
Less: Sustaining Capital	US$M	-$171	--	-$3	-$19	-$4	-$67	-$14	-$48	-$4	-$4	-$5	-$2	--	--
(Including Sustaining Exploration)	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--
All-In-Sustaining Costs	US$M	-$1,663	--	-$179	-$206	-$189	-$212	-$221	-$217	-$154	-$136	-$114	-$36	--	--
Sustaining Margin	**US$M**	**$1,314**	**--**	**$128**	**$102**	**$171**	**$102**	**$139**	**$143**	**$186**	**$171**	**$138**	**$3**	**--**	**--**
Less: Non Sustaining Capital	US$M	-$35	--	--	--	--	--	--	--	--	--	--	-$13	-$18	-$4
(Including Closure Costs)	US$M	-$35	--	--	--	--	--	--	--	--	--	--	-$13	-$18	-$4
Mine Direct Cash Flow	**US$M**	**$1,279**	**--**	**$128**	**$102**	**$171**	**$102**	**$139**	**$143**	**$186**	**$171**	**$138**	**$21**	**-$18**	**-$4**
Less: Working Capital Movement	US$M	--	--	$7	$2	-$1	$3	-$3	$0	-$10	$3	$6	-$7	--	--
Less: Taxes	US$M	($258)	--	--	-$4	-$13	-$26	-$28	-$31	-$41	-$41	-$37	-$31	-$7	--
Less: Lease Principal Repayment	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mine Free Cash Flow	**US$M**	**$1,021**	**--**	**$135**	**$103**	**$154**	**$93**	**$97**	**$119**	**$125**	**$134**	**$106**	**-$16**	**-$25**	**-$4**
Less: Growth Projects	US$M	-$338	-$338	--	--	--	--	--	--	--	--	--	--	--	--
Less: Other outflows	US$M	$0	$0	--	--	--	--	--	--	--	--	--	--	--	--
Net Cash Flow	**US$M**	**$683**	**-$338**	**$135**	**$103**	**$154**	**$93**	**$97**	**$119**	**$125**	**$134**	**$106**	**-$16**	**-$25**	**-$4**
Project NAV (Post Tax)	**US$M**	**$479**													
Project NAV (Pre Tax)	**US$M**	**$663**													
Project IRR (Post Tax)	**%**	**33%**													
Project IRR (Pre Tax)	**%**	**38%**													

22.6 Sensitivity Analysis

22.6.1 Gold Price

Table 22.6.1 shows the impact on the pre-tax NPV @ 5% and pre-tax IRR of a change in the gold price.

Table 22.6.1 Pre-Tax NPV and Pre-Tax IRR Gold Price Sensitivity

Gold Price ($/oz)	NPV ($M)			IRR (%)
	0%	5%	10%	
$1,000	12	-68	-125	1%
$1,250	479	299	172	21%
$1,500	940	663	466	38%
$1,750	1,379	1,009	747	52%
$2,000	1,848	1,379	1,045	67%

22.6.2 Operating Cost

Table 22.6.2 shows the impact on the pre-tax NPV @ 5% and pre-tax IRR of a change in the operating costs

Table 22.6.2 Pre-tax NPV and Pre-Tax IRR Operating Cost Sensitivity @ $1,500 Gold Price

Opex Change (%)	NPV ($M)			IRR (%)
	0%	5%	10%	
-15%	1,164	841	612	46%
-7.5%	1,052	752	539	42%
0%	940	663	466	38%
7.5%	828	574	394	34%
15%	717	485	321	29%

22.6.3 Capital Costs

Table 22.6.3 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 22.6.3 Pre-tax NPV and Pre-Tax IRR Capital Cost Sensitivity @ $1,500 Gold Price

Capex Change (%)	NPV ($M)			IRR (%)
	0%	5%	10%	
-15%	997	718	521	45%
-7.5%	968	691	494	41%
0%	940	663	466	38%
7.5%	912	636	439	35%
15%	884	608	412	32%

23.0 ADJACENT PROPERTIES

The QP is not aware of any relevant work on properties immediately adjacent to the Fetekro Property.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

24.1.1 Implementation Strategy

The cost estimates have been complied on the basis that Endeavour will adopt an EPCM approach, in which a small Owner's team will manage several specialist engineers and consultants, who will carry out the detailed engineering, procure the equipment and fabricated items and manage the installation contracts.

The EPCM engineer will be appointed on a schedule of rates basis, this arrangement permitting Endeavour to have maximum input to the quality of engineering and equipment procured for the project with a minimum Owner's team and securing the services of specialist designers and project management expertise.

The Endeavour Owner's team will need to be progressively expanded to widen its skills and knowledge base to meet the needs of the project. An Endeavour project team will manage both the onshore and offshore activities of the principal EPCM contractor and specialist subcontractors as well as providing specialist technical input into aspects of the project design.

Key onshore mine operations roles will be filled early to contribute to mine design and manage the early mining development works on site. An Endeavour on site management, administration and services department will manage environment and community issues and prepare the site and local communities for the coming influx of construction and subsequently operations personnel.

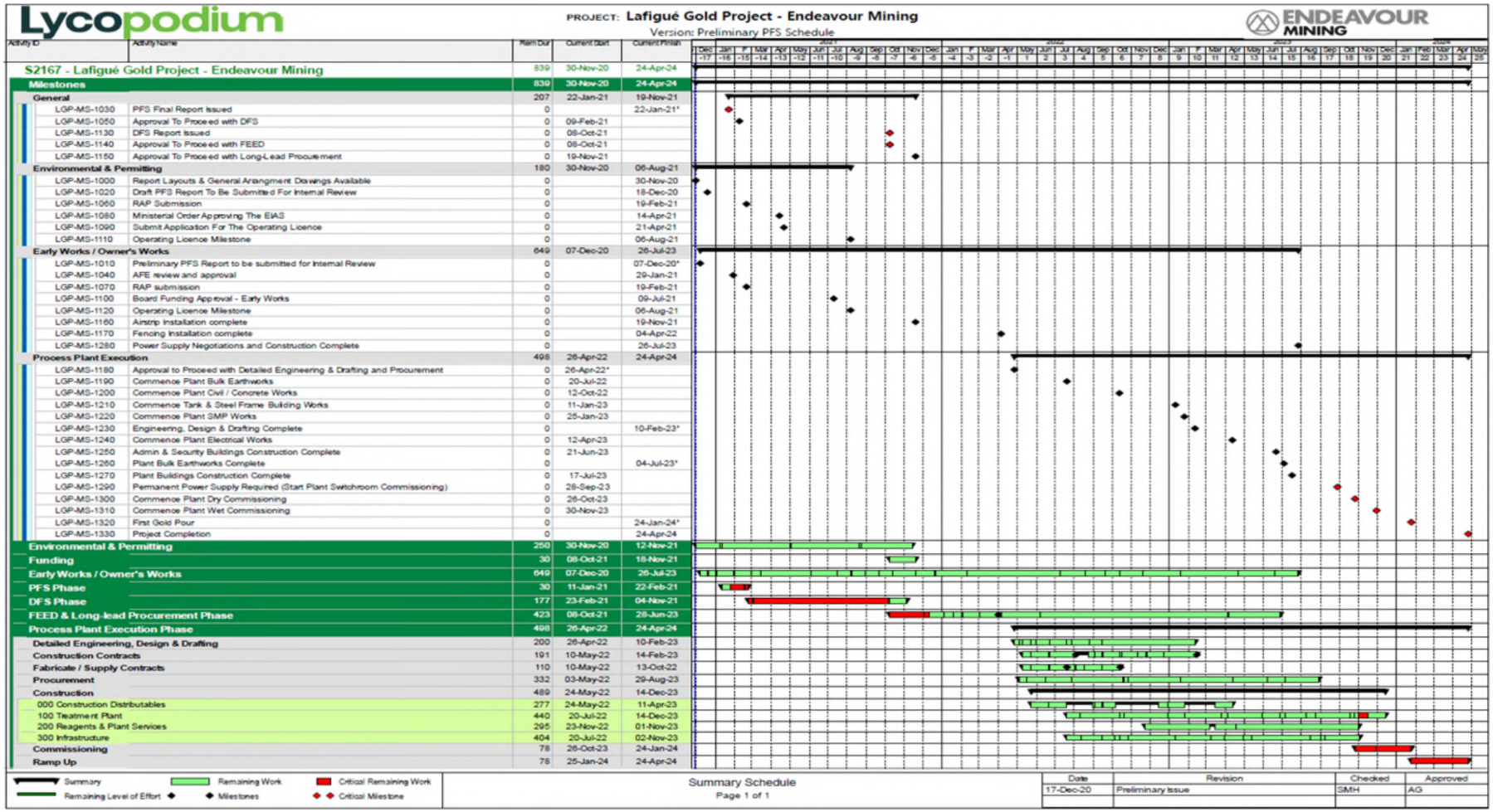
24.1.2 Implementation Schedule

A preliminary schedule has been prepared based on input from Endeavour and Lycopodium experience, which provides an overview of the major activities and their timing. A summary schedule is shown in Figure 24.1.1.

A Front End Engineering Design (FEED) will be utilised and instigate, subject to Board approval, approximately 6 months into the final Feasibility Study (FS) phase. This can be used to shorten the overall project schedule with minimum capital commitment ahead of the approval to proceed decision.

During this phase (normally 3-4 months), preliminary engineering is completed, focussing on finalisation of flowsheets, process design criteria and the mechanical equipment list to 'issued for design'. This is then used to finalising scopes for early works, prepare and issue specifications and tendering for long-lead items.

Figure 24.1.1 Summary Overall Project Implementation Schedule



24.1.3 HSEC Management

The management of health and safety as well as the environment and local community (HSEC) are critical for any successful project development. The EPCM Engineer will develop a project management plan to address these issues, based on their standard procedures, project construction experience and the risks and hazards identified during the feasibility and design stages, all through the project construction to commissioning and hand-over.

All aspects of the Project will be required to comply with the relevant local legislation and current international industrial practice. The Project will target zero harm.

24.1.4 Logistics

Equipment will be sourced from vendors on the basis of price and delivery; such vendors typically fabricate components in many different countries. Fabricated steelwork and platework will potentially be sourced in African and Asia, again on the basis of price, delivery and quality, . The EPCM engineer will engage a logistics and transport coordinator to consolidate freight items where possible and facilitate clearance through customs and transport to site. All items will be transported to site by truck. Oversize loads will receive special attention and escorts as applicable.

24.1.5 Training

Endeavour is an operating company with well-established training procedures. The EPCM engineer will assist Endeavour to train its operations staff during the commissioning period and thereafter as needed, to ensure a smooth transition from construction to operations

24.1.6 Operations

Following successful commissioning of the facilities the project will be handed over to the operations team at Lafigué. A gradual handover of facilities during commissioning will allow Endeavour staff to operate and manage the mine and process plant at nameplate capacity.

24.2 Other

There is no other relevant data or information to the Project that has not already been disclosed in this Report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Overview

The Lafigué Gold Project PFS has identified, defined and costed a relatively low technical risk conventional open cut min with a 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 Côte d'Ivoire and surrounding countries have varying levels of established mining infrastructure.

Sovereign risk is understood by Endeavour having considerable experience in existing operations in Côte d'Ivoire and other West African countries.

Residual environmental and social issues are low and manageable; however, the current large number of artisanal miners in the Mine Project area will require careful management in future development stages.

At a PFS level of study development, the Project exhibits good techno-economics and should be advanced to the next project stage gate, namely a DFS. This conclusion is based on the gold price used in this study, and the level of technical and cost development undertaken, which is, commensurate with the requirements of a PFS.

25.2 Mineral Resource

The Mineral Resource model on which the Project is based is considered to be robust, with the key outcomes being:

- The Lafigué mineralisation shows very good continuity over most of the domains. The drill spacing of 40 m to 50 m over the majority of the resource area allows for classification into the Indicated and Inferred Mineral Resource categories.
- The July 2020 Lafigué Mineral Resource is reported above a grade cut-off of 0.50 g/t Au and inside a US$1,500/oz Au 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 information and data in the Endeavour exploration database, and the QPs opinion and experience. The drilling techniques, survey, sampling, sample preparation, analytical techniques and database management/validation conform to industry standards. The database represents an accurate record of the drilling undertaken in the Project.
- Geological continuity is understood with reasonable confidence. The classification of the Mineral Resource reflects the level of confidence in the MRE.

- The estimation technique and parameters used are appropriate for the style of mineralisation.
- It is the Qualified Person's opinion that the Lafigué Gold Project Mineral Resource documented in this Report meets NI 43-101 requirements.

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 31.9 Mt at 2.0 Au g/t with a strip ratio of 10.3:1.
- Annual gold production is above 200 koz for the first eight years of processing.
- Eleven 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 2020 Mineral Resource, it is expected that the metallurgical and comminution characteristics of this material will be similar to those Lafigué ores previously tested.
- The flowsheet is based on unit operations that are well proven within industry.
- The process plant will process fresh and oxide ores and is expected to operate on either 100% fresh ore (possibly with a small portion of oxide) or 100% oxide ore.
- High recoveries are achievable from all major ore types that are planned to be processed.

25.5 Risks and Opportunities

25.5.1 Introduction

A number of potential risks and opportunities have been identified via a formal risk workshop undertaken by Endeavour and external consultants facilitated by Lycopodium during the PFS. A risk register has been developed. Identified and recorded risks and opportunities have been rated in terms of likelihood and consequence. This has allowed further control actions to be considered to arrive at a residual risk rating.

The register focused on the technical scope of the PFS. The workshop relied on the experience of the team participants to identify risks which could affect the Project outcomes. These risks were then considered in terms of their impact or consequence on one or more categories such as health and safety, environment or cost.

The following sections summarise the key risks and opportunities which will be progressively addressed and managed as the Project progresses.

25.5.2 Risks

In total, 15 risks were initially ranked as high; however, with mitigation the review team considered that the mitigating actions proposed would reduce all the ratings to medium. After mitigation measures, 26 risks where ranked medium and 64 low.

Of the risks initially rated as high, the following are most notable in terms of their risk of having the potential to delay the Project commencement or impact on operations:

- The presence of reportedly 30,000 illegal artisanal miners on site, and the difficulty and time potentially required to remove them.
- Water balance and potential supply options currently indicating that water requirements will need to be satisfied predominantly via water harvesting of rain in wet seasons and storage.
- Community disturbance and social unrest due to construction activities.
- Time required for permitting and construction of the overhead power line (OHPL) to site delaying availability for commissioning.

These key risks are mitigated by:

- The existing and ongoing issues with artisanal miners is currently being addressed within the boundaries of mining and other relevant laws, in consultation with relevant Government authorities. Plans are being developed, once removal is successfully completed, to fence the Project site and use a trained security presence to assist in maintaining control.
- Ongoing evaluation of water supply options and investigation of potential ground water supplies will continue through the FS phase. The construction of the water harvest and storage dams is considered a priority for early capture and storage of water well ahead of construction.
- Continuing engagement of local communities and stakeholders is a priority of Endeavour. All attempts will be made to engage, train and employ local people to improve the overall standard of living. Construction crews will be housed either on-site, or offsite in larger centres to reduce the potential disturbance locally.
- Power security will be prioritised both in early planning and in seeking approvals to construct an overhead power line (OHPL), the potential installation of full back up 20 MW diesel power supply early in construction, supplemented with a 7 MW solar farm.

A number of other risks were identified that could affect the Project costs, schedule, revenue or community. Courses of action were also recommended to mitigate the effect of these risks on the overall Project including the following:

- The potential for residual degradation to the environment from the current artisanal mining activities resultant from the use of medium sized machinery, and the use of reagents and chemicals including mercury.
- The risk to local communities and persons attempting to enter and pass through the site.
- The impact of viscous ores on throughput, particularly during the first year of operation.
- Lack of skilled personnel regionally, impacting ramp up and operation.

These key risks are mitigated by the following:

- Residual contamination of the site by artisanal miners is best remediated in the first instance with their removal; however, plans to understand the residual impacts and processes to remedy any environmental impact will be instigated.
- Accidental injury or fatality potential will be controlled by fencing the Project site and ongoing engagement of local peoples to understand the risks. Where necessary, Endeavour will establish safe access ways for transport.

25.5.3 Opportunities

Notable opportunities identified that will be assessed and pursued as the Project progresses include:

- Further drilling and modelling work to upgrade Inferred Mineral Resources to Indicated Mineral Resources.
- Ongoing engagement with Government to understand potential contributions to capital items including the 15 km access road and site power supply, both of which have the potential to benefit the local community.
- Potential to eliminate the requirement for full back-up diesel power.
- Potential to further optimise the mining waste dump locations. With clearance from exploration and redesign of the pit exits, the south dump and to a lesser extent the west dump could be expanded allowing for a size reduction of the north dump which is at a higher elevation.

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 Environmental and Social

In compliance with Ivorian regulations and IFC performance standards, an Environmental and Social Impact Assessment (ESIA) has been completed. Two major phases remain for the ESIA validation, which include the public inquiry and the validation workshop, as discussed in Section 20.

Key work and actions required include:

- Collection of water, noise and dust monitoring baseline data. A preliminary budget allowance of US$117,500 should be allocated to this activity.
- Installation of a weather station in the new exploration camp. A preliminary budget allowance of US$12,000 should be allocated to this activity.
- Conduct a detailed inventory of all land and crop assets (planned for January 2021) to understand compensation requirements and develop and implement a livelihood restoration programme (LRP). A preliminary budget allowance of US$85,000 should be allocated to this activity.
- Develop and implement community development programmes to build and subsequently reinforce Endeavours' social license to operate. A preliminary budget allowance of US$100,000 should be allocated to this activity.

26.2 Mineral Resource

- A structural study is required to improve the understanding of mineralisation controls (budget allowed for in the costs below).
- The future exploration campaign should test the junction of Lafigué Centre and Lafigué North. A preliminary budget of US$3,200,000 should be allowed for this activity.
- Update the Mineral Resource using an independent consultant to validate the Resource. A preliminary budget of US$83,000 should be allowed for this activity.
- Review the need for additional drilling to provide a Measured Mineral Resource category in the initial mining phase areas.

26.3 Pit Geotechnical

A preliminary budget allowance of US$360,000 should be allocated for the activities listed below.

- Plan and complete additional geotechnical investigations to increase the level of confidence and reliability of current geotechnical assessments, particularly in relation to the larger Mineral Resource identified in 2020.
- Geotechnical data obtained from future drilling should be cross-checked with findings and assumptions presented in this PFS report.

26.4 Mining

As part of the forward work programme for mining, a series of recommendations have been made below. These recommendations/activities either form part of the Feasibility Study or are costed as part of the other recommendations listed herein.

- Increase the confidence (grade and tonnage) of the Mineral Resource by completing a drilling programme to increase a portion of the Lafigué deposit that will be mined in the early part of the mine life from Indicated to Measured category.
- Establish early engagement with potential Mining Contractors experienced in West African operations to obtain a firm budget price estimate for the open pit mining operations.
- Review pit shell designs (in particular the orientation of Stage 5) to potentially reduce the stripping ratio in the first years of operation, and delay the stripping the ridge.
- The current resource model is estimated on a minimum block size of 2.5 mE x 2.5 mN x 1.25 mH. The block model size is suitable for the evaluation of the underground potential. Future expansions to the resource model should be cognisant of the requirement for the models use in underground and open pit evaluations.
- Further review of the potential for in-pit backfill should be evaluated as part of the FS as it has not been included in the PFS study.
- Produce a short-term mine schedule to firm up the early production profile.
- Investigate a redesign of waste dumps using LIDAR topography instead of regional topography which appears to be approximately five metres higher.
- While groundwater is not expected to be significant enough to impact on mine operations or pit wall stability, it is recommended that this assumption be confirmed with a detailed hydrogeological study.

26.5 Metallurgical and Process Plant Selection

While the metallurgical testwork conducted to date is sufficient to support the PFS design, additional work has been recommended for the FS. A budget allowance of US$140,000 is recommended for the test work outlined below.

- Testwork to cover ore zones added in the updated 2020 Mineral Resource and to confirm whether the data used in the PFS is representative of the expanded resource.
- The addition of a HPGR as opposed to the previous Scoping Study designed SABC circuit requires additional testwork prior to the FS, to confirm design criteria assumptions made in the PFS.
- Undertake additional viscosity and thickening testwork to understand the implications of intended processing strategy for separate treatment campaigns for oxide and fresh ores, and the associated implication for plant throughput and design.

26.6 Infrastructure

26.6.1 Geotechnical Investigation

Conduct a detailed site geotechnical investigation for the PFS plant site, TSF and other infrastructure areas prior to commencement of detailed design. A preliminary budget allowance of US$50,000 should be allocated to this activity.

26.6.2 Topographical Survey

Conduct a detailed topographical survey of the site infrastructure locations, in particular the TSF and WHD footprints. A budget of US$100,000 should be allowed for this activity.

26.6.3 Water Balance

Conduct a review of water balance modelling to consider relocating the WHD further upstream to reduce the distance from the WSD, thus reducing the pumping/pipeline costs. This would be offset by the reduced catchment, thus will require optimisation. This recommendation falls within the scope of the FS budget.

26.6.4 Access Road

Conduct a detailed road route survey to assist with design of 15 km section of the road to be upgraded from Bounadougou on the B412 to site. A budget of US$140,000 should be allowed for this activity.

26.6.5 Grid Power

Continue discussions with power network provider and regulator to understand requirements for design and in particular seek agreement for early commencement of OHPL route development to site. A budget of US$50,000 should be allowed for this activity.

26.7 Project Advancement

Based on the positive outcome of the PFS, it is recommended that the Lafigué Project be advanced to the next stage gate, namely a Definitive Feasibility Study (DFS). The DFS will be carried out in accordance with industry norms, best practice and/or standards with respect to the level of technical and cost development expected. For the DFS, a budget of US$3.8 M has been allowed for the engagement of consultants and peripheral project development activities and resources.

27.0 REFERENCES

27.1 References

The following references were used in the preparation of this report:

1. ANCOLD, "Guidelines on Tailings Dams, Planning, Design, Construction, Operation and Closure", July 2019.

2. ANCOLD, "Guidelines for Design of Dams for Earthquakes", 1998.

3. Civil Aviation Safety Authority (Australian Government), Manual of Standards Part 139 – Aerodromes Version 1.11, November 2013.

4. International Civil Aviation Organization (ICAO), Aerodrome Design Manual, 2006.

5. Rico M., Benito G., and Díez-Herrero A., "Floods from tailings dam failures (Science Direct: Journal of Hazardous Materials)", 2007.

28.0 CERTIFICATES OF QUALIFIED PERSONS