NI 43-101 Technical Report

Timok Project

Pre-Feasibility Study

Zagubica, Serbia

Prepared by:

Philip de Weerdt, P. Eng, PMP, MBA
DRA Americas Inc.

Schadrac Ibrango, P.Geo., Ph.D., MBA
DRA Americas Inc.

Daniel Gagnon, P.Eng.
DRA Americas Inc.

Volodymyr Liskovych, PhD, P. Eng.
DRA Americas Inc.

Claude Bisaillon, P. Eng.
DRA Americas Inc.

Ross Overall BSc (Hons), CSci, MIMMM.
Dundee Precious Metals

Kevin Leahy BSc (Hons), PhD, CGeol
ERM

David Ritchie, P. Eng.
SLR

Luis Vasquez, M.Sc., P. Eng.
SLR

Galen White, FAusIMM
CSA Global (UK)

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This Technical Report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable laws. Forward-looking statements are based upon the responsible Qualified Person's (QP) opinion at the time they are made but, in most cases, involve significant risks and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this Report, there may be other factors that could cause events or results not be as anticipated, estimated or projected. There can be no assurance that forward-looking information in this Report will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements or information, Accordingly, readers should not place undue reliance on forward-looking information. Forward-looking information is made as of the effective date of this Technical Report, and none of the QPs assume any obligation to update or revise it to reflect new events or circumstances, unless otherwise required by applicable laws.

This report contains certain non-GAAP (Generally Accepted Accounting Principles) measures such as cash cost and ASIC. All-in sustaining cost per ounce for the Project represents mining, processing, site general and administrative costs ("G&A"), water treatment costs, royalties, treatment and refining charges and sustaining capital, divided by payable gold ounces, and excludes corporate G&A. Such measures have non-standardized meaning under GAAP and may not be comparable to similar measures used by other issuers. See DPM's latest Management's Discussion and Analysis available on DPM's website (www.dundeeprecious.com) and on SEDAR (www.sedar.com) for additional general information about non-GAAP measures reported by DPM.

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1 SUMMARY

This report summarises the results of the Pre-Feasibility Study (PFS) technical report (Report) completed by DRA Americas Inc. (DRA) on behalf of Dundee Precious Metals Inc. (DPM) for the Timok gold project (the Project).

1.1 Property Description and Location

The Timok property is located in the eastern part of the Republic of Serbia, approximately 270 km southeast of its capital, Belgrade. Its northern boundary is positioned about 25 km south from the Danube River and the Project area extends 24 km southwards to a point approximately 14 km west and southwest of Bor at its southern boundary. The main deposits on the Project are located approximately 25 km northwest of the town of Bor, Serbia. Bor is a historical centre for copper mining and smelting in Serbia.

The Timok property comprises three (3) exploration licences (Potaj Čuka Tisnica, Umka and Bigar Istok) covering an aggregate area of 131.21 km. The Bigar Hill, Korkan, Korkan West, and Kraku Pester Deposits, which are the subject of this Report, are located within the boundary of the Potaj Čuka Tisnica exploration licence.

The exploration licences for the Project are held by Avala Resources d.o.o., a Serbian registered, wholly owned subsidiary of DPM, following the amalgamation of a wholly owned subsidiary of DPM with Avala Resources Ltd. in April 2016.

The Potaj Čuka Tisnica and Bigar Istok exploration licences were renewed (second renewal) in July 2019 and are valid until July 2021. The Umka exploration licence was renewed in August 2019 for a further three years (first renewal).

There are no other known agreements or encumbrances on the properties. DPM operates with the permission of the Serbian Ministry of Mining and Energy (MoM&E), in conjunction with the Ministry of Environmental Protection, and the Ministry of Culture and the Media of the Republic of Serbia.

DPM does not currently own the surface rights to any of the land parcels located on the exploration licences. To gain access to the land to conduct exploration activities, land access agreements have been negotiated with the local landowners, in the case of privately held land, or with the state, in the case of state land. These land access agreements follow Serbian legislative requirements in terms of proscribed compensation for access and land disturbance, etc. The land access agreements are recorded in a master register to document and maintain transparency in negotiating and maintaining land access compensation.

DPM intends to acquire land through a voluntary willing-buyer willing-seller approach. In the case that this does not occur, there are mechanisms under the Law on Expropriation that will allow the government to acquire immoveable properties for projects demonstrated to be within the public interest.

1.2 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The property is accessible by regional asphalt roads between Bor, Žagubica, Krepoljin, and Zlot, and well-developed unpaved forestry roads. The area is also linked via Bor to Zaječar and Paraćin and via Žagubica to Požarevac (and further to Belgrade). There is a railroad from Bor to Belgrade through Požarevac.

The town of Bor is connected by rail to Belgrade (via Požarevac). This same rail network is part of European Transportation Corridor 10, which extends southwards through the Republic of North Macedonia to Greece and the Mediterranean, and also eastwards through Bulgaria to ports on the Black Sea (and further on to Turkey). Bor is accessible via the national highway grid (Paraćin turnpike), leading to paved roads through Boljevac to Bor.

The property area is characterised by moderate continental climate, with some influence of high mountainous climate. Winters are long and cold, with abundant snow cover, and summers are usually hot. First seasonal frosts occur in October and the last frosts are in April. Site elevations vary between 600 and 830 m. The estimates used to characterise the climate at the Project site have been obtained from long-term observations from the Crni Vrh weather station, located 13 km to the south at an elevation of 1,037 m. Data from this station is considered representative of the upper end of the site elevation range. Site estimates are that the coldest month is January, with an average temperature of -3.4°C, and the hottest month is July, with an average temperature of +17.2°C.

The annual precipitation is in the range of 500 mm to 1,130 mm, with the mean annual precipitation estimated to be 770 mm. The mean monthly precipitation at the Project area is estimated to vary from about 48 mm in both January and February to about 85 mm in both May and June.

Terrain in the Project area comprises steep, narrow valleys and rounded hilltops, ranging from about 500 masl in valleys in the northern part of the Project, to 944 masl at Coka Rakita, the highest peak in the region directly, south of the Project. Other high peaks are Coka Berbjesce (817 masl), Strez (731 masl), and Coka-Unuk (741 masl).

1.3 History

The Timok region has a long history of exploration and mining, dating back to Roman times. Key periods include:

Previous exploration at the Project, undertaken from 2007 to 2009, has been summarised by Coffey Mining (2010). DPM is not aware of any exploration for gold taking place within the Project area prior to 2007. Prior to DPM's involvement from July 2010, there are no historical Mineral Resource and Mineral Reserve estimates for the Project.

In July 2010, Avala Resources Ltd. acquired Avala Resources d.o.o. (formerly named Dundee Plemeniti Metali d.o.o.) from DPM through a reverse takeover transaction, in which DPM retained a 51% share.

Extensive soil sampling and surface trenching programs were carried out during the 2007 to 2009 period. Four (4) (581.7 m) diamond core drill-holes and 152 trenches (28,014.6 m for 14,138 samples) were completed on the Project, though much of this was outside the four (4) deposits that are the subject of this Report (Bigar Hill, Korkan and Korkan West, and Kraku Pester).

Avala then focused exploration drilling campaigns from 2010 to 2013 on the Potaj Čuka Tisnica licence to outline mineralisation on the Bigar Hill, Korkan, Kraku Pester and Umka areas. The drilling that relates to Bigar Hill, Korkan and Kraku Pester is covered in more detail in Section 10. Along with drilling, from 2010 onwards, outcrop, soil and trench sampling were conducted. After 2014, several exploration trenches, channels and drill-holes were completed on wide-spaced grids on areas peripheral to the mineralised prospects.

After Avala was fully re-acquired by DPM in 2016, a near-resource target generation exercise was undertaken, which led to the discovery of the Korkan West Deposit during winter 2016/2017.

1.4 Geological Setting and Mineralisation

The Project is located within the north-western part of the Timok Magmatic Complex (TMC) in eastern Serbia. The TMC is part of the greater Alpine-Balkan-Carpathian-Dinaride metallogenic-geodynamic (ABCD) province (Figure 7.1), which, in turn, is part of the Tethyan (or Alpine-Himalayan) orogenic system that extends from Western Europe to South-East Asia. The orogen resulted from the convergence and collision of the Indian, Arabian, and African plates with Eurasia, initially in the Cretaceous and continuing today.

The structural complexity and present-day asymmetric lozenge-shaped geometry of the TMC area resulted from oroclinal bending during post-collision tectonics throughout the Tertiary. This has led to tectonic modifications of lithological contacts, including those that represent syn-depositional features, beds, or faults. The extent of deformation is commonly difficult to assess due to variable responses of different rock types to the same deformation event. Much of the deformation has been absorbed by argillaceous horizons due to their ability to accommodate shearing and shortening, whereas sandstone beds have resisted much of the deformation. Similarly, competent massive limestone units forming the base of the sequence exhibit minor deformation and much of this is expressed as fracturing near the contact with the overlying clastic sedimentary rocks.

1.5 Exploration

Intensive exploration at the Project commenced in July 2010 following the acquisition of the projects by Avala Resources Ltd. A systematic exploration approach has been undertaken with the assembly of the following data sets over the whole Project area: topography, geological mapping, rock chip sampling, trenching, channeling, and stream sediment geochemistry. Stream sediment sampling was previously completed over the entire Project area, at a nominal density of one sample per square kilometre.

Anomalous areas were followed up by rock chip sampling, mapping, and soil sampling, on a first-pass 400 m x 50 m grid in some of the anomalous areas and, very locally, subsequently with 100 m x 50 m grid sampling.

Trenching was used as a follow-up strategy to explore areas with anomalous soil geochemistry and to assist in defining key geological relationships due to the limited outcrop in the Project areas. There was a high success rate in intersecting sediment-hosted gold mineralisation by drilling near extensive and well mineralised trench intercepts.

Channel samples are routinely taken on road cuttings or where outcrop exists. Channels are typically cut using a hammer and chisel, which allows sufficient penetration to excavate a channel approximately 100 mm high and 30 mm deep. Samples are caught into a chip tray which is cleaned at the end of every interval.

Geophysical survey works within Timok belt started in 2006, by means of a heliborne VTEM survey, covering the original exploration license areas of the Project. Additionally, Induced Polarization surveys have been used since from the commencement of exploration works at Timok. Profiling arrays (Dipole-Dipole) with variable dipole spacing, depending on the target in question.

1.6 Drilling

Avala has employed a combination of diamond drilling and RC drilling across the Bigar Hill, Korkan, Korkan West and Kraku Pester exploration areas, and diamond drilling at Umka.

The majority of drill holes at the Bigar Hill project are orientated at an azimuth of 270° and inclined approximately 60° to the west. In the case of Kraku Pester, drill holes are mostly inclined at 60° to the east to intersect gently west-dipping mineralisation. At Korkan and Korkan West, however, the orientation of the mineralisation is much more variable than at either Bigar Hill or Kraku Pester, and this is reflected in the greater range of drill hole orientations.

Table 1.1 – Summary of Exploration Drilling, Channel Sampling and GC Drilling, (as at May 29, 2020)

Deposit	Diamond Drilling		RC		RC Pre-Collar/ Diamond Tail		Geotechnical/ Hydrogeological		Metallurgical		Total	
	Drill Holes	m	Drill Holes	m	Drill Holes	m	Drill Holes	m	Drill Holes	m	Drill Holes	m
Bigar Hill	129	32,157	333	71,287	30	3,423	5	1,134	14	1,879	511	109,880
Korkan (including Korkan West)	281	65,614	295	49,804	9	1,237	7	850	22	2,099	614	119,604
Kraku Pester (inc. Kraku Pester South)	51	12,478	94	14,962	7	960	-	-	-	-	152	28,400
Total	**461**	**110,249**	**722**	**136,053**	**46**	**5,620**	**12**	**1,984**	**36**	**3,978**	**1,277**	**257,884**

1.7 Mineral Processing and Metallurgical Testing

Metallurgical testwork for the Timok deposits Bigar Hill, Korkan, and Korkan West included the 2011-2013 testwork program and the 2018-2020 testwork program. Sulphides were the focus of the 2011-2013 testwork program. The 2018-2020 testwork program included tests on oxide, transitional, and sulphide material. Testwork included comminution tests, open and closed cycle flotation tests, coarse bottle roll and column leach tests, whole ore leach tests (CIL), and mineralogical characterisation. Planning and supervision of the testwork as well as review of the results was carried out by DPM. DRA provided input during review of testwork results produced in 2020. A summary of the testwork was prepared by DPM, and interpretation of the results and development of the process design basis has been conducted by DRA.

Testwork on the Project is considered representative of the ore to be processed pursuant to this PFS, according to PFS mine schedule. Testing provides suitable coverage for the primary ore types, namely Bigar Hill, Korkan, and Korkan West oxide ores, and Bigar Hill and Korkan transitional ores.

From the completed metallurgical testwork programs, the oxide and transitional ore types were shown to have amenability to conventional heap leaching technology for the recovery of contained gold. The crushing, heap leach, and adsorption-desorption-recovery (ADR) process facilities have been designed to process ore from the Bigar Hill, Korkan, and Korkan West deposits. Agglomeration has been included on an as required basis, but a greater understanding of clay characteristics and deportment is required. To date, testwork on the sulphide material has not demonstrated an economically viable process flowsheet. Further geometallurgical investigations are planned.

The results of the various metallurgical programs allowed for definition of a single-stage irrigation schedule, with a design leach gold recovery of 82.6%. A discount lab-to field factor of 2% was applied for the oxide ore, and a 5% discount lab-to field factor was applied to the transitional ore lab column leach recovery results.

The target 100% passing (P_{100}) crush size of 25 mm for all ore types is based on testwork results and DRA experience.

In the QP's opinion:

- Metallurgical testwork completed to date has been appropriate to evaluate and develop appropriate process routes and metallurgical assumptions for the various Timok mineral resource estimates.
- Metallurgical testing data supports the metal recovery assumptions contained in the LOM plans and metal recovery schedules.
- Samples selected to prepare the metallurgical composites are considered to be representative and reflects future feedstock to the process plant.
- The QP is not aware of any processing factors or deleterious elements that could have a significant impact on potential economic extraction.

1.8 Mineral Resource Estimate

DRA completed a Mineral Resource Estimate (MRE) update for the Bigar Hill (BH), Korkan (KO) and Korkan West (KW) gold deposits of the Project, located in Serbia. No estimation update was made for the Kraku Pester deposit. The previous MRE for Kraku Pester performed in 2018 by CSA Global remains current and has not been updated.

The MRE in this Report follows exploration work completed since the last MRE. Additional information consisted of a further of two hundred eighteen (218) additional drill holes distributed on all the three (3) deposits. The previous 2018 CSA MRE had an effective date of May 15, 2018. Since the previous estimate, the additional drillholes have a cumulative length of 27,394.1 m.

The current MRE is based on an updated drill hole database which encompasses a total of 1,360 drill holes and trenches with a cumulative length of 259,400.3 m. The database received from DPM also encompasses drilling done on the Kraku Pester deposit although no mineral resource estimate update has been performed for this deposit. The date of receipt of the final drill hole database from DPM was May 29, 2020 which can be considered as the effective date of the MRE.

The following sections describe the methodology, parameters and key assumptions used in the preparation of the updated MRE. The MRE is based on integrations using integrated geological and assay information recorded RC drilling, diamond core and trenches logging and assaying. DPM geologists prepared lithology, mineralisation and weathering interpretations using the Leapfrog software package. DRA reviewed these models and found them suitable for the use in the MRE.

The estimation work was completed using the HxGN MinePlan 3D (previously Minesight®) and Datamine Studio software package by DRA and Datamine Consultants South Africa. The MRE was performed in two steps with an initial estimate performed using Ordinary Kriging (OK) interpolation into a sub-blocked model, with parent blocks of 20 m×20 m×10 m and sub-blocks of 5 m × 5 m × 5 m, within defined estimation zone domains (ESTZON). A second step was achieved in Datamine Studio using the Localized Uniform Conditioning (LUC) approach for support change and localisation from the parent block size to Selected Mining Unit (SMU) size which is 5 m × 5 m × 5 m.

Under the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards, Mineral Resources should have a reasonable prospect of eventual economic extraction. In order to determine the mineralisation zones that can be potentially mined economically, an optimised pit shell was developed using the Lerchs-Grossmann (LG) Algorithm implemented in GEOVIA Whittle® software.

The MRE may be affected by various factors inherent to mineral properties. In particular, it is unknown if DPM can obtain all required surface rights, governmental approvals and permits for the possible development and operation of the Project.

The MRE was performed by Dr. Schadrac Ibrango, P.Geo., Ph.D., MBA, senior consulting geologist of DRA Global with assistance of Datamine for parts related to UC/LUC. Dr. Ibrango is a Qualified Person (QP) as defined by NI 43-101 and independent from DPM. Mineral Resources have been classified using the definitions in CIM Definition Standards for Mineral Resources and Mineral Reserves, May 10, 2014 (CIM Definition Standards). The criteria used by the QP for classifying the estimated Mineral Resources are based on confidence and continuity of geology and grades. The CIM Definition Standards for Mineral Resource classification are provided in Section 14.1 and a summary of the estimated Mineral Resources is provided in Table 1.2. Risks as related to Mineral Resource estimates are discussed in Section 25.2.

Table 1.2 – Consolidated Mineral Resource Estimate – Effective May 29, 2020

Deposit	Ore Type	Indicated Mineral Resource			Inferred Mineral Resource		
		Tonne (Mt)	Au		Tonne (Mt)	Au	
			(g/t)	k oz		(g/t)	k oz
Bigar Hill	Oxide	5.2	0.79	132	0.0	0.69	0
	Transitional	6.2	0.94	186	0.0	0.94	1
	Sulphide	10.9	1.63	572	0.6	1.79	36
	Sub-total	*22.3*	*1.24*	*890*	*0.7*	*1.73*	*37*
Korkan	Oxide	2.2	0.70	49	0.0	0.42	0
	Transitional	2.0	0.78	50	0.1	0.54	1
	Sulphide	3.4	1.89	206	0.0	1.17	0
	Sub-total	*7.6*	*1.25*	*305*	*0.1*	*0.56*	*2*
Korkan West	Oxide	0.0	0.74	1			
	Transitional	0.0	0.64	0	0.0	0.15	0
	Sulphide	0.0	1.12	0			
	Sub-total	*0.1*	*0.71*	*2*	*0.0*	*0.15*	*0*
Kraku Pester	Oxide	0.7	0.95	22	0.1	1.3	5
	Transitional	0.1	0.95	4	0.0	1.2	0
	Sulphide	1.5	2.01	95	0.0	1.8	0
	Sub-total	*2.3*	*1.61*	*122*	*0.1*	*1.3*	*6*
Total Oxide		*8.2*	*0.78*	*205*	*0.2*	*1.1*	*5*
Total Transitional		*8.3*	*0.90*	*241*	*0.1*	*0.7*	*3*
Total Sulphide		*15.8*	*1.72*	*873*	*0.6*	*1.8*	*37*
Grand Total		**32.3**	**1.27**	**1,319**	**0.9**	**1.5**	**45**

Footnotes
1. The effective date of the MRE for Bigar Hill, Korkan and Korkan West is May 29, 2020
2. The effective date of the MRE for the Kraku Pester Estimate is May 15, 2018
3. Mineral Resources are reported in accordance with CIM Definition Standards
4. A cut-off of 0.19 g/t Au for the Oxide material, 0.216 g/t Au for the Transitional material, and 0.571 g/t Au for the Sulphide material is applied at Bigar Hill, Korkan and Korkan West.
5. A cut-off of 0.35 g/t Au for the Oxide material, 0.40 g/t Au for the Transitional material, and 1.05 g/t Au for the Sulphide material is applied at Kraku Pester.
6. Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.
7. Due to rounding, some columns or rows may not compute exactly as shown.
8. The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.
9. The Mineral Resources are stated as in-situ gold ounces.
10. The models are reported above surfaces based on conceptual $1,400 gold price pit shells to support assumptions relating to reasonable prospects of eventual economic extraction.
11. 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.
12. Mineral Resources are reported exclusive of Mineral Reserves

All blocks coded within the mineralised domains were estimated and classified. However, only the blocks located inside the resource pit shell were considered as Mineral Resources since they demonstrate a "reasonable prospect of eventual economic extraction" as defined in the CIM Definition Standards. The CIM requirement implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade.

The operating parameters for the optimisations and cut-off grade estimates were based on reasonable technical (maximum pit slopes, gold recovery) and economical (gold price, unit mining and processing costs) parameters. Unlike pit design shells for reserves, inferred category Mineral Resources were included.

The Reader should note that the overall slope angle parameters used for the three (3) deposits vary by lithology and type of oxidation. The overall slope angles are detailed in Section 16.3

The Mineral Resources are constrained within pit shells based on the parameters presented in Table 1.3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1.3 – Parameters Used for Mineral Resource Constraining Pit Shell

Description / Parameter	Unit	Bigar Hill	Korkan	Korkan West
Gold Price	$/oz	1,400		
Overall Slope angle - Oxide	degree	Refer to Section 16.3		
Overall Slope angle - Transitional and Sulphide	degree	Refer to Section 16.3		
Mining Recovery	%	95	95	95
Mining Dilution	%	0	0	0
Mining Cost - Waste	$/t mined	1.71	1.84	1.96
Mining Cost - Oxide and Transitional	$/t	2.07	2.46	2.15
Mining Cost - Sulphide	$/t	2.16	2.45	1.67
Gold Process Recovery - Oxide	%	85	85	85
Gold Process Recovery - Transitional	%	74.52	74.52	74.52
Gold Process Recovery - Sulphide	%	89.90	89.90	89.90
Process Cost - Oxide and Transitional	$/t processed	5.24	5.24	5.24
Process Cost - Sulphide	$/t processed	19.10	19.10	19.10

Description / Parameter	Unit	Bigar Hill	Korkan	Korkan West
G&A Cost - Oxide and Transitional	$/t processed	1.60	1.60	1.60
G&A Cost - Sulphide	$/t processed	2.67	2.67	2.67
Metal Payable - Oxide, Transitional and Sulphide	%	99		
Royalty	%	5		
Rehabilitation Cost	$/t waste	0.0207		
Process Rate - Heap Leach (Oxide and Transitional)	Mtpa	2.5		
Process Rate - Sulphide Concentrator	Mtpa	1.5		
Grams in a Troy Ounce	g/oz	31.1035		
Discount Rate	%	10		

1.9 Mineral Reserves Estimate

Mineral Reserves for Timok were estimated using GEOVIA Whittle to determine the economic pit limits for the Bigar Hill, Korkan West, and Korkan pits. Only Measured and Indicated Mineral Resource categories and only oxide and transitional minerals were considered for the Mineral Reserves. A standard open pit truck and shovel operation was assumed, with a 2.5 Million tonne per annum (Mtpa) ore production rate. An optimised pit shell containing 17.6 Mt of ore at an average grade of 1.16 g/t of gold was selected. No Mineral Reserves have been estimated for the Kraku Pester prospect since this prospect is comprised predominantly of primary sulphide mineralisation.

From the optimised pit shell, operational pits were designed using Datamine Studio. Haul roads and ramps were designed with an overall width of 15 m.

The Mineral Reserves are estimated at 19.2 Mt of probable reserves grading 1.07 g/t. To access the Mineral Reserves, a total of 48.3 Mt of waste will need to be removed, resulting in an overall 2.52 stripping ratio. Table 1.4 presents a summary of the Mineral Reserves estimate for the Project.

Table 1.4 – Mineral Reserves Estimate by Pit – Effective May 29, 2020

Deposit	Ore Type	Probable Reserves Tonnes (Mt)	Au Grade (g/t)	Mined Au ounces (k oz.)	Strip Ratio (Waste/Ore)
Bigar Hill	Oxide	8.8	1.19	334	**2.85**
	Transitional	1.9	1.09	67	
	Sub total	**10.7**	**1.17**	**401**	
Korkan	Oxide	3.4	0.90	97	**2.69**
	Transitional	1.2	1.02	39	
	Sub total	**4.6**	**0.93**	**137**	
Korkan West	Oxide	3.7	0.99	118	**1.42**
	Transitional	0.3	0.74	6	
	Sub total	**4.0**	**0.97**	**124**	
Total	Oxide	15.8	1.08	549	**2.52**
	Transitional	3.4	1.04	110	
	Total	**19.2**	**1.07**	**662**	

Footnotes:

1. The effective date of the Mineral Reserve Estimate is May 29, 2020.
2. Mineral Reserves are reported in accordance with CIM guidelines.
3. A marginal cut-off of 0.21 g/t Au for the Oxide material, and 0.24 g/t for the Transitional material is applied at all deposits.
4. Mineral Reserves were estimated at a gold price of $1,250 per oz and include modifying factors related to mining cost, and dilution and recovery, process recoveries and costs, G&A, royalties, and rehabilitation costs.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows may not compute exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher. All figures are in metric tonnes.

1.9.1 GEOTECHNICAL PARAMETERS

SRK Consulting (Canada) Inc. (SRK) has provided open pit slope design parameters for the three (3) proposed open pits of the Project: BH, KO, KW. The design parameters are based on geotechnical site investigations, available local and regional geological data, and well-established geotechnical design methods used to estimate the Project design pit slope angles. SRK conducted the work in four (4) phases over the period of January 2019 to July 2020.

SRK has identified geotechnical rock mass units associated with the primary rock and alteration types, based on the results of the site investigation by SRK, the University of Belgrade Faculty of Mining and Geology and geological interpretations by Dundee. Major geological structures (faults and foliation) have been included in the geotechnical slope stability analyses for each pit. Slope stability analyses were conducted using industry standard limit-equilibrium software, finite element analysis software, and in-house proprietary SRK tools.

A field program to characterise the hydrogeological conditions was conducted as part of the investigations by the University of Belgrade (Faculty of Mining and Geology, Department of Hydrogeology). SRK completed hydrogeological studies for each of the proposed pits, and numerical simulations of pit dewatering/depressurization have been carried out. SRK interpreted hydrostratigraphic units, estimated hydraulic conductivity.

1.9.2 OPEN PIT MINING

Conventional open pit mining with rigid body mining trucks, hydraulic excavators and wheel loaders was chosen for the Project. The material extracted from the Project's three (3) pits: BH, KW, and KO, will be loaded into trucks by hydraulic excavators and transported to its destination. Each pit is mined in three phases and the Project is mined over a 7-year mine life, with one (1) year of pre-production and an additional year at the end of the mine life for stockpile reclamation.

Ore material, either oxide or transitional material, will be sent to the crusher or an ore stockpile; marginal material will be stockpiled to be reclaimed in the operation's last year. Reclaimed stockpile material will be loaded onto trucks by wheel loaders and sent to the crusher. After the crusher, the ore material will be sent to the leach pad. Waste material will be sent to waste piles located near each pit, and sulphide material above the sulphide cut-off grade will be stockpiled near the waste piles for potential future reclamation.

The mine will be operated by an owner-operated fleet of 60 T trucks and related equipment seven (7) days a week, 24 hours a day in two (2) 12-hour shifts. The operation considers two weeks of adverse weather conditions, therefore running 350 days a year.

The mine plan is based on an ore production rate of 2.5 Mtpa sent to the leach pad. The oxide ore has a higher recovery rate at the leach pad than does the transitional ore and was therefore favoured in the optimization process. A total of 67.6 Mt of material is extracted over the Project's life-of-mine. The annual mine production schedule, by pit is presented in

Table 1.5.

The operation will require 90 employees during pre-production and 183 during operation.

Table 1.5 – Mine Production Schedule (by Pit)

Pit	Ore/Waste Tonnage by Pit (kt)								Total
	Pre-Prod	Y1	Y2	Y3	Y4	Y5	Y6	Y7	
BH Ore	44	1,390	1,644	2,375	1,736	2,082	1,339	72	10,683
KO Ore	0	0	0	0	801	657	1,547	1,562	4,567
KW Ore	29	1,349	1,842	0	0	529	216	0	3,964
Total Ore	74	**2,739**	**3,486**	**2,375**	**2,537**	**3,267**	**3,102**	**1,634**	**19,214**
Au Grade (g/t)	0.90	1.19	1.00	0.81	1.37	1.12	1.01	0.98	1.07
BH Waste	2,178	5,277	5,442	7,625	5,189	2,777	1,806	132	30,426
KO Waste	0	0	0	0	2,274	2,677	4,897	2,459	12,307
KW Waste	1,082	1,985	1,072	0	0	1,280	195	0	5,612
Total Waste	**3,260**	**7,261**	**6,514**	**7,625**	**7,463**	**6,733**	**6,898**	**2,591**	**48,345**
Total Material	**3,333**	**10,000**	**10,000**	**10,000**	**10,000**	**10,000**	**10,000**	**4,226**	**67,559**
Strip Ratio	44.2	2.7	1.9	3.2	2.9	2.1	2.2	1.6	2.5

1.10 Recovery Methods

Oxide and transitional ores from the BH, KO, and KW deposits will be processed using conventional heap leach technology, adopting a Heap Leach Fill (HLF) design. The design stacking rate is based on processing 2.5 Mtpa, at an average gold grade of 1.07 g/t, and an overall discounted gold recovery of 84.9% and 71.8% for the oxide and transitional ore types respectively. The design criteria were developed based upon the testwork results described in Section 13.

The HLF pad will be constructed in two (2) separate phases: 9.9 million tonnes in Phase 1 with an addition of 9.3 million tonnes during Phase 2, totalling to 19.2 million tonnes of ore processed during 8 years of mine life. Phasing of the Project includes the earthworks and pad construction and lining and reduces initial capital expenditure.

The process plant has been designed based on the mineral processing test work results received.

The concentrating steps can be summarised as follows:

- Three (3) stage primary crushing;
- Agglomeration circuit;
- Trucking to the HLF pad;
- Leach pad;
- Irrigation and solution recovery system;
- Adsorption, desorption, and recovery plant (ADR).

The process flowsheet is based on proven technology operated by numerous gold operations around the world. Key process plant design criteria were derived from the metallurgical test work programs conducted and have allowed for the sizing of major equipment items. Metal production estimates were prepared based on metallurgical test results, and the mine plan developed for the PFS. The plant layout and mechanical equipment selected ensure access for proper constructability, operability, and maintainability.

The heap leach operation was designed to crush and stack 2.5 Mtpa of ore to a crush size of 100% passing 25 mm. The mineral processing facility is designed to operate 365 d/a. The crushing plant and agglomeration circuit have been specified with an operating availability of 75%, equivalent to 18 h/d of operation. The leaching and ADR plant have been designed for an availability of 95%, equivalent to 22.8 h/d.

1.11 Project Infrastructure

1.11.1 Site Layout and Process Element Description

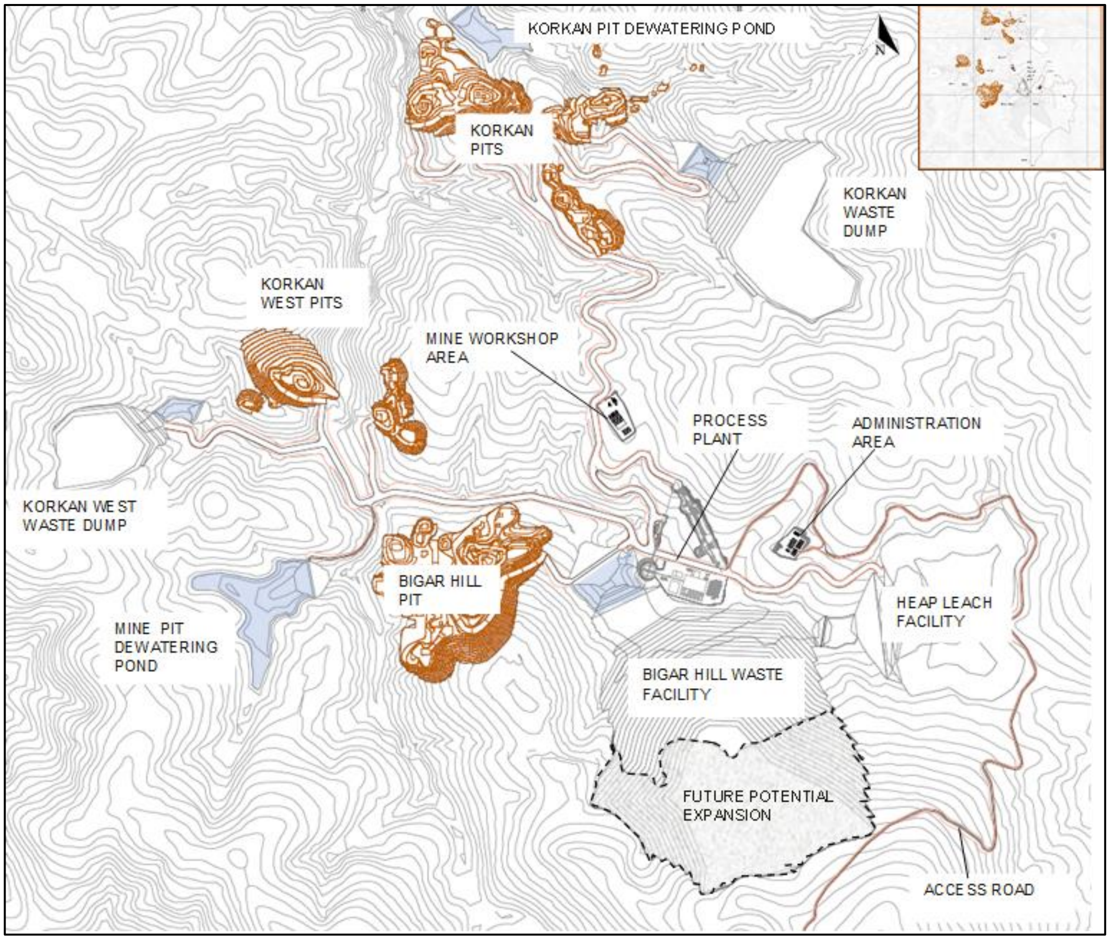
Project infrastructure elements were arranged to optimise the use of available space near the designated mine pits and nearby waste rock dumps. The three waste rock dumps each have a contact water pond for capturing and treatment of contact water. Waste rock facility capacities are aligned with current pit waste storage requirements. Minimal waste haulage is anticipated between facilities.

Pit dewatering is undertaken using two (2) additional unlined ponds with contact water from the Korkan West and Bigar Hill pits contained in the southern dewatering pond. The northern Korkan pits will have a separate pit dewatering pond. It is assumed that pit dewatering contact water does not require chemical water treatment, only turbidity treatment via settling, prior to environmental discharge.

As the main ore recipient facility, the heap leach facility location was first determined following a trade-off process. The heap leach crushing facility and ADR processing plant are located near the heap leach facility to ensure operational optimisation. Space allocation for a potential future sulphide processing facility is reserved in the vicinity of the ADR processing pant. Process plant support services are located adjacent to the ADR plant.

Support infrastructure is located on two (2) main terraces namely administrative and mining terrace areas. The mining terrace houses the mine workshops, fuel storage and mine administration buildings. The administration terrace houses the management offices, security buildings, substation, and on-site clinic.

Figure 1.1 – Site Layout



1.11.2 Heap Leach Facility and Processing

The 36-hectare valley fill Heap Leach Facility (HLF) is sized to accommodate 20 Mt of oxide and transitional ore. It is designed with an internal process pond, and external event pond. The internal pond allows for better liquor temperature control and environmental safety but is more expensive compared to conventional external pond facilities. The HLF is double geomembrane lined underneath the internal pond area with the second geomembrane acting as a leak detection and prevention layer. The HLF is designed to comply with the International Cyanide Management Code (ICMC) of which DPM is a signatory. Construction of the HLF will occur in two (2) phases with the first reaching the extents of the internal process pond. The second expansion will occur during the third year of operation to allow for the heap expansion and growth in height. Construction will require mine waste rock to construct the initial embankment.

The heap leach ore crushing facility consists of 3 stages of crushing with a design throughput rate of 2.5 Mtpa. Final product size from the crushing plant will be 100% passing 25 mm. Ore clay content is yet to be defined. The crushing circuit design allows for an agglomeration circuit that will involve the addition of cement. Crushed ore is loaded into a loading bin from where it is batch dumped into a haul truck for delivery to the HLF.

The ADR facility is a closed building housing the main processing equipment and columns. Gold refining and security rooms are incorporated within the building. Reagent off loading, storage and mixing forms part of the ADR building. Cyanide mixing facilities are sufficiently isolated to ensure safety.

1.11.3 WASTE FACILITIES

The three waste dump facilities were designed as lined facilities to understand the associated costs. Waste rock facility lining is required to protect ground water from heavy metal leaching and acid rock drainage. Ongoing waste rock sample test work indicates that less than 10% of sampled waste rock is potentially acid generating. Acid generating waste will require lined containment and associated water run-off will require chemical treatment prior to environmental discharge. The Bigar Hill waste facility will be partially lined to accommodate potentially acid generating waste rock up to 7 Mt. The Bigar Hill facility is expandable to accommodate a potential future sulphide processing facility which is not addressed as part of this PFS. The rest of Bigar Hill, Korkan and Korkan West waste facilities are planned as conventional unlined facilities.

1.11.4 HAUL ROADS AND GEOTECHNICAL

A 5.4 km, 6 m wide site access road connects to the main site administration area with the national road 105. 9 km of site haul roads have been designed to connect the multiple pits, waste dumps and processing facilities. The haul roads are double-laned with an 18 m wide road surface to accommodate 90-ton Cat77 haul trucks and will be constructed from mine waste rock and excavated material.

An opportunity exists to further optimises the haul road construction during the next project phase through the maximising of excavation instead of engineered filling during construction, and the narrowing of the haul road should smaller haul trucks, either owner fleet or contractor mining, be viable.

Geotechnical investigations are recommended prior to commencement of the next project phase. An initial geotechnical drilling and test pit plan has been compiled that aligns with the current project element locations. Initial geotechnical feedback will confirm the suitability of the chosen project element locations.

1.11.5 Electrical Supply

Two (2) transmission lines, operated by the national power utility, Elektromreza Srbije, are located approximately 5 km south of the Project; one transmission line at 110 kV and the other transmission at 35 kV. Power system studies using ETAP software were conducted to evaluate capacities of both transmission line voltages to supply the Project.

The power studies determined that the existing 35 kV transmission line does not have sufficient capacity to supply the Project, whereas the existing 110 kV transmission line may be adequate to supply the Project, provided the existing power grid has spare capacity to supply the Project.

For the purposes of this study, it is assumed that electrical power for the Project will be provided by Elektromreza Srbije, via existing 110 kV transmission line 122ab connected to the Project's main substation. Elektromreza Srbije will need to be engaged in the next project phase to confirm and secure the Project's impact on the existing 110 kV power grid.

In the next phase of the Project, power system parameters including power supply reliability and power factor correction will need to be confirmed with the national power supply provider. In addition, power connection costs along with energy tariffs and power demand rates will also need to be confirmed. Initial tariffs rates were obtained from Elektromreza Srbije and will require confirmation in the next study phase.

The main substation has been designed with allowances for future addition of a concentrator along with future backup redundancy for both heap leach and future concentrator facilities.

1.11.6 Site Facilities and Services

Site facilities are located on two main terraced areas. The administration area houses site offices, main substation, clinic room and laboratory. The mine terrace area houses mining related facilities such as the workshop, fuel bay and mine offices.

Site wide services include raw water supply, potable water supply, compressed air, fire water systems, sewage treatment, water treatment and waste disposal.

Process raw water supply will be collected from the Bigar Hill waste facility pond water where a positive water balance is expected. Potable water supply will be from locally located boreholes that will be identified following currently planned hydrogeological studies. A potable water treatment plant will be present on site. Distribution of potable and fire water will be via buried services from the main ADR processing plant terrace to the administrative and mining terraces. Fire water will consist of a buried service ring main with protruding hydrants and connected fire hoses within insulated buildings. Sewage treatment will be housed on the ADR plant terrace with septic tanks located at the terraces; permanent sewage connections will be investigated during the next phase of the Project. Non-mineral wastes, including hazardous and chemical waste will be disposed of off-site at designated government facilities. Suitable third-party waste carriers and treatment/disposal sites will be identified and the details of the approach for storage, transportation, treatment, and disposal of each waste stream will be set out in the Project waste management plan still to be developed during the next phase.

1.11.7 Site Water Management and Treatment

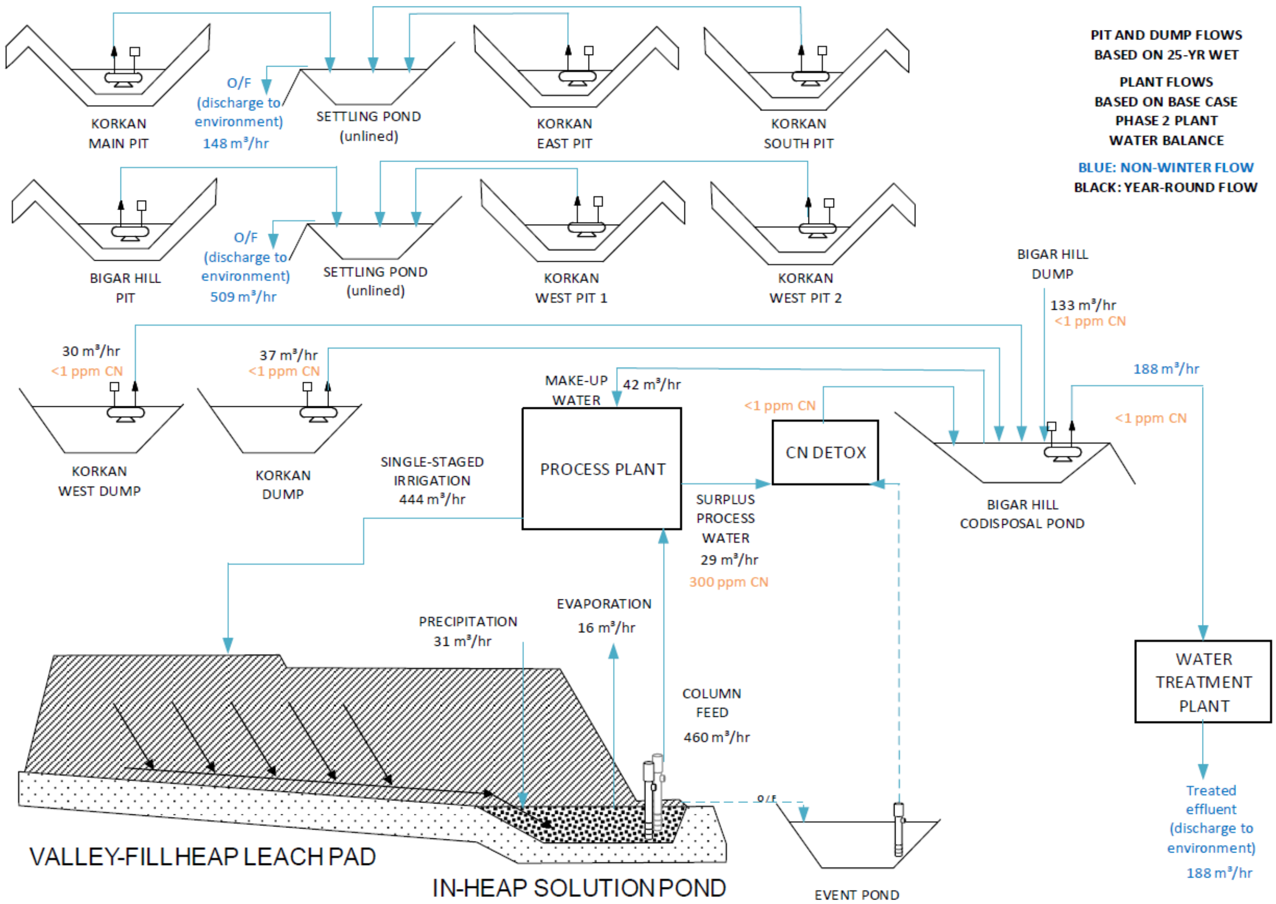
The site wide water management system has been devised to minimise the Project's impact on surface as well as ground water systems associated with the local area. A site-specific hydrogeological study has not yet been undertaken. Experienced based assumptions and interpretations have been made to approximate the water management system flows.

The water management system of the Project includes the following main components:

- Non-contact water diversion ditches – minimising area of influence and associated surface water volume treatment through the construction of diversion ditches to avoid ingress of non-contact water onto exposed mine infrastructure areas;
- Contact water collection ditches – where surface water has come into contact with infrastructure or disturbed areas, such water is collected and channeled to collection ponds for analysis and appropriate treatment;
- Contact water management ponds for the HLF, waste rock dumps (and potential tailings co-disposal) – collection of contact water for appropriate treatment. The ponds also provide opportunity for water storage and discharge monitoring;
- Sedimentation ponds for management of open pit dewatering – surface and pit ingress water is collected in the pits and pumped to sedimentation ponds where the water is provided with an opportunity to settle out solids prior to environmental discharge.

The site-wide water balance simulations were developed for average annual precipitation conditions as well as dry and wet annual precipitation conditions considering three (3) return periods: 10 years, 25 years, and 100 years. The modelling results show that even during dry years (i.e., annual precipitation below average), there is a water surplus that must be discharged to the receiving environment. The analysis indicates that surface runoff collection can meet the water demand for the Project without the need to withdraw water from natural streams or aquifers through groundwater wells, even under dry annual precipitation conditions. It is envisaged that potable water source will continue to be drawn from a suitable well to ensure uncontaminated water source for human consumption.

Figure 1.2 – Timok Project PFS Water Flow and Treatment Chart



The site water flow requires two (2) water treatment plants:

- a cyanide detoxification plant to neutralise surplus cyanide containing process water from the cyanide heap leach circuit. The cyanide detoxification plant makes use of Caro's acid, a mixture of sulphuric acid and concentrated hydrogen peroxide (50% strength), as an oxidizing detoxification agent, for treatment of the concentrated surplus of process water (300 ppm CN). Each treated process water batch is sampled and tested prior to release into the contact water pond storage facility.
- a contact water treatment plant for settling and clarifying contact water prior to environmental discharge. The effluent treatment plant makes use of coagulation, flocculation, and decantation steps for metals treatment and pH correction. Utilising the site water balance and a design factor of 20%, the effluent water treatment capacity has been determined as 227 m³/hr, for treatment during the non-winter months.

1.12 Environmental Studies, Permitting and Social or Community Impact

The Project has undertaken permitting efforts, environmental studies, and community engagement since 2007. Initial environmental and social baseline work commenced in 2012 and continues to the present day.

The proposed Project is located in Serbia, and as such will be permitted to operate and regulated by Serbian authorities to Serbian standards. Mining structures are permitted under the Law on Mining and Geological Explorations whilst supporting and auxiliary structures such as roads and administrative buildings are separately permitted under the Law on Planning and Construction. There is a range of other approvals and permissions required under ministries including the Ministry of Agriculture, Forestry and Water Management and Ministry of Environmental Protection. However, the permitting system is undergoing change as part of Serbia's planned accession to the European Union (EU).

DPM has sought to minimise permitting risks by engaging with regulators and aligning the Project with EU requirements and good international practice such as the performance requirements of the European Bank for Reconstruction and Development (EBRD) (European Bank for Reconstruction and Development (EBRD), n.d.), Equator Principles (The Equator Principles, 2020) and World Bank Group Environmental, Health and Safety Guidelines (International Finance Corporation World Bank Group, 2007).

Key environmental and social risks are similar to those associated with other gold mining projects and as a result measures are required to safeguard rivers, groundwater, biodiversity, the local community, and local heritage and mitigate permanent effects.

A Mine Water Management Plan (Section 18.15) has been developed for the PFS design outlined above, which defines the hydrological parameters to support engineering design, estimates the open pit dewatering rates, and defines the site water management plan, water management closure concepts, site-wide water balance modelling, sizing of water management ponds and ditches, pump sizing, and construction material estimates. Contact and non-contact waters will be managed separately, with diversions around most of the infrastructure directed to the Jagnilo River. During operations, rivers will be affected by dewatering, diversions and discharges, and permanent infrastructure will overlie several kilometres of upper headwaters reach of the Jagnilo tributaries.

Process water supply will be from the Bigar Hill process pond which is primarily fed by contact water run-off from the Bigar Hill waste rock dump and from the Korkan and Korkan West ponds (themselves fed by waste rock dump contact water run-off). Excess water in the Bigar Hill process pond will be treated and monitored, prior to discharge to the Valja Saka, at an industrial water treatment plant to be located by the north-western end of the pond. Process water balance calculations and diagrams have been developed (DRA, September 2020) to model water needs in baseline conditions, in unusually dry conditions and in unusually wet conditions (24 hr 1 in 100-year storm event). Spillways on the dams will prevent over-topping and will discharge to the Jagnilo River or its tributaries, although the impoundments have been designed to accommodate extreme (24hr 1 in 100-year) rainfall events.

The design will have liners, drains or seals preventing infiltration installed on the process plant area, the Valley Heap Leach (VHL) pad and all three (3) ponds serving the waste rock dumps. Waste rock dumps will only be lined in areas designated for rock-types identified as potentially acid generating, currently anticipated to be an area or cell within the Bigar Hill waste rock dump, which will drain to the lined Bigar Hill process pond, where water treatment is available.

Whilst the transportation and use of cyanide in the heap leach process presents potential risks to surface and groundwater quality, DPM is a signatory to the International Cyanide Management Code (International Cyanide Management Code, n.d.) , which provides standards of practice for protection of communities and the environment during transportation of cyanide and specific usage requirements on handling, storage, operation, disposal and decommissioning. These standards and guidance have been used in the development of the design options at this PFS stage.

The Project will result in loss of habitat, including several kilometres of riverine habitat, which will have an impact on biodiversity. Stone crayfish are present in the Project area and are sensitive to low flows, high water velocities and poor water quality. The Project may affect these and other protected species.

The Project's social licence to operate is of critical importance. There are risks associated with acquiring land, for which careful planning will be required. Seasonal farming, hunting and tourist amenity will also be lost during operations and certain cultural heritage buildings and features will be affected by the Project. DPM has built a good relationship with the local community over many years. The use of cyanide lixiviant, short life of mine and expectations around employment may be key issues amongst stakeholders. The key risks and mitigations for these different aspects that will be more completely covered in the Project ESIA and Livelihood Restoration Plan to be undertaken in the near future.

An outline Mine Closure Plan (MCP) (ERM, 2020) has been drafted as part of the PFS. The Project's approach to closure will be to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use. The current MCP closure vision is to restore the site to pre-mining land use and status and will include revegetating using compatible species. The heap leach and waste rock dumps will remain after site closure as low, unsaturated tabular landforms, which will be placed over a sealed base and will ultimately be capped with low permeability covers. Monitoring of groundwater and surface water will be undertaken through-out the LOM.

The Project currently holds exploration licences and is in its final exploration licence extension period. It has prepared an environmental permitting strategy (ERM, 2020) setting out the principal permits and approvals required for further exploration, construction, operation and closure of the Project. Key permits are summarised in Figure 20.2. A spatial plan will be required for the areas of Potaj Čuka and Bigar Istok to authorise a change in land use to mining. The process of development of a plan is lengthy, taking of the order of 18-24 months and up to 33 months in the case of a recent mining project in Serbia of similar scale.

An Environmental Impact Assessment (EIA) will be required. There are risks to the projects associated with permitting delays. Such delays be caused by potential changes to Serbian regulations to align with EU Law, regulator delay, public challenge to the Spatial Plan or EIA and administrative appeals. Ongoing engagement with regulators will be need.

1.13 Capital Cost Estimate

1.13.1 Capital Cost Estimate Summary

Capital expenditures (Capex) consist of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are also included. Amounts for the mine closure and rehabilitation of the site are specifically excluded. Capex is reported in United States Dollars ($).

Table 1.6 presents a summary of the initial Capex. Sustaining Capex is distributed over the LOM, separately indicated from the initial Capex. Owner's costs, its contingencies and risk amounts have been included in this Capex.

Table 1.6 – Initial Capex Summary by Major Area

Description	Cost ($ Millions)
Pre-stripping	8.6
Mining (mine fleet, haul, and access roads)	36.7
Processing (heap leach, processing plant)	52.7
Infrastructure	17.2
Waste rock facilities and Other	16.7
Total Direct Costs	**131.9**
Construction Indirect & owner's costs	42.1
EPCM	15.8
Total indirect costs	**57.8**
Contingency	*21.0*
Total Initial Capital	**211.0**
Life of mine	
Sustaining capital expenditures	24.4
Closure and rehabilitation costs	23.3

1.13.2 Sustaining Capital Summary

Sustaining capital comprises the replacement of mobile equipment, expansion of waste rock facilities and systems as shown in Table 1.7.

Table 1.7 – Sustaining Capital Cost Estimate

Main Area	Sustaining Capital ($ US '000)
3700-Heap Leach	8,717
1300-Open Pit Mine Infrastructure	5,597
1100-Open Pit Mine Development	3,521
1200-Open Pit Mining Equipment	1,781
29000-Indirects For Sustaining Capital	3,681
21000-Closure Costs	24,430
Total	**47,727**

1.13.3 Closure and Rehabilitation Costs

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. SLR advised the quantities associated with the Project closure and the prices were included in this estimate as sustaining capital. Closure costs are based on a developed closure plan.

1.14 Operating Cost Estimate

The operating costs (Opex) were estimated for the Project and cover the costs related to open pit mining, ore processing, waste, and water adduction, and General and Administration (G&A) costs including site services, and catering. The Opex summary is depicted in Table1.8.

Table1.8 - OPEX Summary

Description	Annual Average ($ M)	LOM Totals ($ M)	Average/ton ($/tonne)
Mining	19.0	151.6	7.89
Processing	12.4	99.1	5.16
G&A	12.4	33.8	1.76
Water Treatment	0.5	5.2	0.27
Total	**44.2**	**289.6**	**15.07**

1.15 Economic Analysis

For the purposes of the economic analysis, the Project has been evaluated using a constant metal price of $1,500/oz Au. Gold sales provide Project revenues. No other product is being considered as part of the financial analysis.

The Project generates an after-tax Net Present Value (NPV) of $135 M at a 5% discount rate, an after-tax IRR of 20.6% and the after-tax payback period is 3 years from commencement of production.

An economic analysis based on the production and cost parameters of the Project was carried out and the results are summarised in Tables 1.9 and 1.10. All figures are in USD currency.

Table 1.9 – Economic Summary

Description	Units	Value
Gold Price	$/oz	1,500
Total tonnes of ore mined and processed	Mt	19,214
Total tonnes waste mined	Mt	48,345
Strip ratio	waste:ore	2.52
Head grade	g/t Au	1.07
Peak tonnes per day ore mined	tonne	7,610
Weighted average gold recovery – Oxide Ore	%	84.95
Weighted Average gold recovery – Transitional Ore	%	71.49
Total gold ounces recovered to doré	oz	547,034
Average annual gold production to doré	oz	68,379
Peak annual gold production to doré	oz	87,166
Mine life	years	8
Unit Operating Costs		
LOM average cash cost	$/oz Au	606

Note:

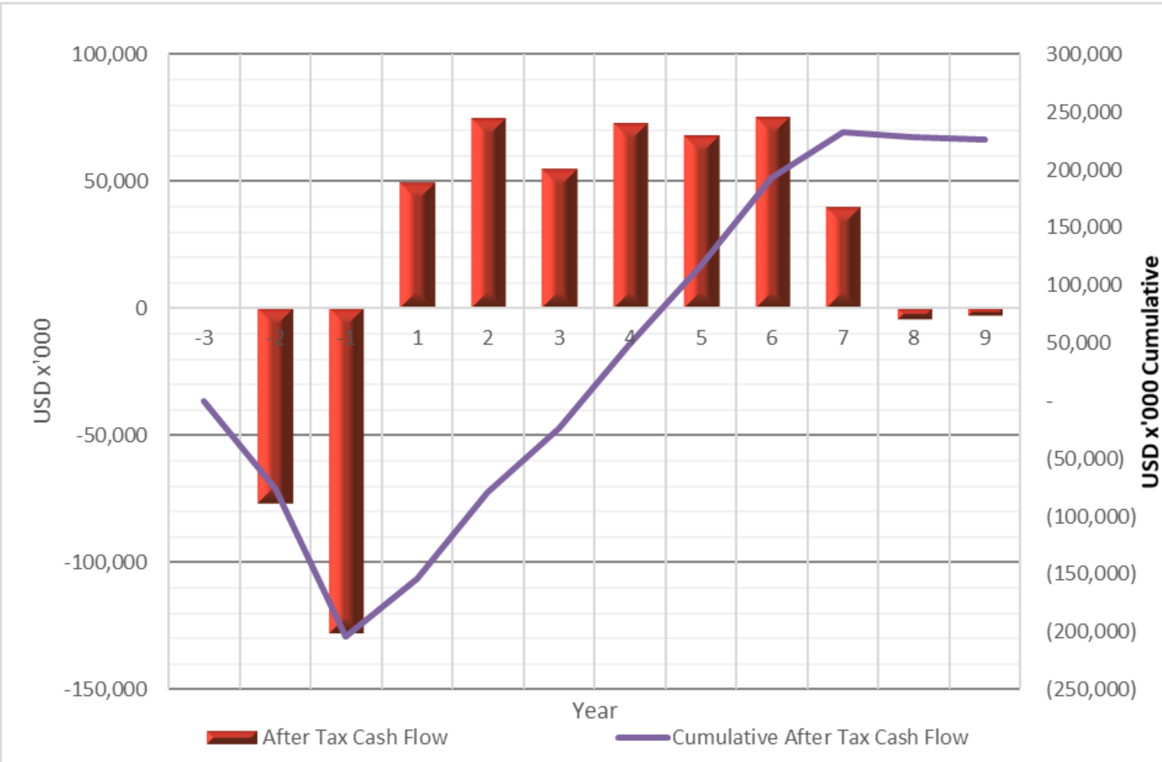
This Report contains certain non-GAAP (Generally Accepted Accounting Principles) measures such as cash cost and ASIC. All-in sustaining cost per ounce for the Project represents mining, processing, site general and administrative costs (“G&A”), water treatment costs, royalties, treatment and refining charges and sustaining capital, divided by payable gold ounces, and excludes corporate G&A. Such measures have non-standardized meaning under GAAP and may not be comparable to similar measures used by other issuers. See DPM’s latest Management’s Discussion and Analysis available on DPM’s website (www.dundeeprecious.com) and on SEDAR (www.sedar.com) for additional general information about non-GAAP measures reported by DPM.

Table 1.10 – Financial Analysis Summary

Description	Unit	Pre-Tax	After-Tax
NPV	$US million	139	135
IRR	%	20.9	20.6
Discount Rate	%	5	5
Payback Period (from start of production)	Years	3	3

The annual Project cash flows are presented in Figure 1.3.

Figure 1.3 – Project Discounted Cash Flow (After-Tax)



2 INTRODUCTION

2.1 The Issuer

This Report has been prepared for Dundee Precious Metals Inc. (DPM or the Company), a gold mining company listed on the TSX, with headquarters at 1 Adelaide Street East, Suite 500, Toronto, Ontario, Canada, M5C 2V9.

2.2 Business Case

DPM is in the process of evaluating the Timok Gold Project (the Project) near Bor, Serbia after acquiring 100% ownership of the Project in April 2016. This Project is being evaluated as part of DPM's potential project pipeline toward long term business sustainability.

In April 2016, DPM completed the acquisition of 49.9% of the common shares of Avala Resources d.o.o (Avala) not already owned by the Company, thus taking full ownership of, amongst others, the Project. Since that time, DPM has undertaken a series of exploration and evaluation programs to better understand the nature of the deposits.

Following positive economic results from previous Preliminary Economic Assessment (PEA) studies, DPM commissioned DRA Americas Inc. (DRA) to perform a pre-feasibility study (PFS) for the three (3) associated Timok prospects: Bigar Hill, Korkan and Korkan West. This Report evaluates and recommends suitable technologies, mining and processing methods, waste, and infrastructure requirements to economically develop the Project.

2.3 Previous Studies and Recommendations

2.3.1 Preliminary Economic Assessment (PEA) 2014

Prior to becoming a wholly-owned subsidiary of DPM, Avala initiated a PEA for the Project in 2016 to evaluate the economic exploitation of the Bigar Hill, Korkan and Kraku Pester Mineral Resources utilising a flotation-based concentrator plant, based on a series of related test work programs conducted during 2012 and 2013.

2.3.2 Mineral Resource Estimate 2018

In 2016, DPM commissioned CSA Global to perform a MRE update (CSA Global, 2018) for the Project, based on flotation related test work previously completed by Avala in 2014. Further introduction of conceptual heap leach specific test work related to the oxide and transitional ore deposits described the potential technology benefit.

An MRE update was performed taking all in-fill drilling information up to May 2018 into account. This MRE is referred to as "Block Model, 2018" and contains the Mineral Resource bodies Bigar Hill, Korkan, Korkan West and Kraku Pester.

Economic analysis was not performed due to the preliminary nature of the heap leach related test work on limited representative samples. Recommendations included further heap leach related test work on a more comprehensive sample selection.

2.3.3 Preliminary Economic Assessment 2019

DPM commissioned CSA Global to update the initial report completed during 2018 (CSA Global, 2018) following completion of further representative sample heap leach related test work. The associated mining simulations identified significant sulphide extraction during the focussed oxide and transitional mining, prompting the evaluation of a sulphide concentrator plant to beneficiate the mined sulphide material. Following an initial process flow sheet development (2.5 Mtpa heap leach and 0.5 Mtpa sulphide concentrator), site layout and cost estimation (AACE Class 5 ± 40%) completion, economic evaluations were performed.

PEA recommendations included further heap leach related test work on a wider sample selection, establishment of a geometallurgical model to better understand the ore deposits, and MRE inclusion of further in-fill drilling results undertaken during 2019. Refinement test work on sulphide concentrator grind size and variability test work was recommended to better define processing requirements. Finally, it was recommended to continue with infrastructure related and pit slope geotechnical evaluations.

2.3.4 Internal Optimisation Study 2019

Following the completion of the updated PEA in 2019 (CSA Global, 2019), DPM conducted an internal optimisation study to evaluate maximising sulphide resource extraction in addition to oxide and transitional ores. Further optimisation of the mine pit design and scheduling was also completed. A 2.5 Mtpa heap leach and 1.5 Mtpa sulphide concentrator were recommended, with associated mine plans and resources. This information was provided to DRA as input considerations to the PFS.

2.4 Pre-Feasibility Study Purpose

The purpose of the PFS is to review and define the optimum configuration for the Timok Mineral Resource, based on the latest available test work and MRE. The PFS will further provide engineering definition through mine design, project infrastructure definition and optimised operations descriptions. The definitions will be followed by estimation and confirmation of project economics.

2.5 Scope of Work

The PFS evaluation is sub-divided into a Project Definition Study and PFS phase:

- Project Definition Study:

- Re-evaluate the entire Timok Mineral Resource to develop the optimal mining method, processing technologies, throughput rates and process configuration through the completion of trade-off studies.
- Compilation of a Project Definition Study.

- PFS Phase:
  - Develop the project definition recommendations into engineered and estimated deliverables:
    - Process and facility design and estimate development.
    - Design and estimating of infrastructure to support site access, mining, and processing.
  - Updating of the MRE (CSA, 2018) with additional in-fill drilling data – referred to as "Block Model 2020".
  - Pit optimisation and mine design based on updated MRE Block Model 2020.
  - Metallurgical test work interpretation and further test work recommendations.
  - Interpretation of geotechnical reports and data to better input parameters related to infrastructure and pit designs.
  - Project execution scheduling and planning.
  - Project capital estimation (Capex).
  - Operations cost estimates (Opex) and operational readiness planning.
  - Financial modelling.
  - Risk and security recommendations.
  - Recommend future workplans for subsequent study phases.

2.6 Information Sources

2.6.1 Reference Documents

A comprehensive list of referenced documents is provided in Section 27.

2.7 Qualified Persons

At the request of DPM, DRA has been mandated to prepare a PFS for the Project with the participation of specialised consultants. Table 2.1 provides a responsibility matrix with roles and responsibilities of each participant during the PFS.

Table 2.1 – Qualified Persons

Section	Title of Section	Qualified Person
1	Summary	ALL QPs
2	Introduction	Philip de Weerdt (DRA)
3	Reliance on Other Experts	Philip de Weerdt (DRA)
4	Property Description and Location	Philip de Weerdt (DRA)
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Philip de Weerdt (DRA)
6	History	Ross Overall (DPM)
7	Geological Setting and Mineralisation	Ross Overall (DPM)
8	Deposit Types	Ross Overall (DPM)
9	Exploration	Ross Overall (DPM)
10	Drilling	Ross Overall (DPM)
11	Sample Preparation, Analysis and Security	Ross Overall (DPM)
12	Data Verification	Ross Overall (DPM)
13	Mineral Processing and Metallurgical Testing	Volodymyr Liskovych (DRA)
14	Mineral Resources Estimates	Schadrac Ibrango (DRA)
14.18	Mineral Resources Estimates	Galen White (CSA)
15	Mineral Reserve Estimates	Daniel Gagnon (DRA)
16	Mining Methods	Daniel Gagnon (DRA)
16.3	Geotechnical	Claude Bisaillon (DRA)
16.4	Pit Dewatering	Luis Vasquez (SLR)
17	Recovery Methods	Volodymyr Liskovych (DRA)
18	Project Infrastructure	Philip de Weerdt (DRA)
18.14	Waste Facilities	David Ritchie (SLR)
18.15	Water Management (excluding Section 18.15.2)	Luis Vasquez (SLR)
19	Market Studies and Contracts	Philip de Weerdt (DRA)
20	Environmental Studies, Permitting and Social or Community Impact	Kevin Leahy (ERM)
21	Capital and Operating Costs	Philip de Weerdt (DRA)
21.2.2	Mining Opex	Daniel Gagnon (DRA)
21.2.3	Process Plant Opex	Volodymyr Liskovych (DRA)
22	Economic Analysis	Alex Duggan (DRA)
23	Adjacent Properties	Philip de Weerdt (DRA)

Section	Title of Section	Qualified Person
24	Other Relevant Data and Information	Philip de Weerdt (DRA)
25	Interpretation and Conclusions	ALL QPs
26	Recommendations	ALL QPs
27	References	ALL QPs

2.8 Effective Date and Declaration

This Technical Report has the following effective dates:

- Technical Report: March 30, 2021
- Date of Mineral Resource Estimate: (Bigar Hill, Korkan, and Korkan West) May 29, 2020
- Date Mineral Resource Estimate: (Kraku Pester) May 18, 2018

This Report is considered effective as of March 30, 2021 and is in support of the DPM's press release, dated February 23, 2021, 2021, entitled "*Dundee Precious Metals Announces Positive Pre-Feasibility Study and Encouraging New Exploration Results for the Timok Gold Project in Serbia*."

The effective date of the Mineral Resource Estimate (MRE) is the 29th May 2020 and includes the Bigar Hill, Korkan and Korkan West deposits only.

The effective date of the Kraku Pester Mineral Resource Estimate (MRE) is the May 15, 2018 and remains valid for the consolidated MRE dated 29th May 2020.

2.9 Site Visit

This Section provides details of the personal inspection on the Property by some of the Qualified Persons.

Table 2.2 – Site Visit by Qualified Persons

Qualified Person	Site Visit	Date of Site Visit
Philip de Weerdt (DRA)	No	-
David Ritchie (SLR)	No	-
Luis Vasquez (SLR)	No	-
Volodymyr Liskovych (DRA)	No	-
Kevin Leahy (ERM)	No	-
Ross Overall (DPM)	Yes	4-7th December, 2019
Galen White	No	
Schadrac Ibrango (DRA)	No	-
Daniel Gagnon (DRA)	No	-
Claude Bisaillon (DRA)	No	

2.10 Units and Currency

In this Report, all currency amounts are in US Dollars ("**USD**" or "**$**") unless otherwise stated. Quantities are generally stated in *Système international d'unités* ("**SI**") metrics units, the standard Canadian and international practices, including metric tonne ("**tonne**", "**t**") for weight, and kilometre ("**km**") or metre ("**m**") for distances.

2.11 Abbreviations

Abbreviations may be used in this Report are listed in Table 2.3.

Table 2.3 – Acronyms and Abbreviations

Abbreviation	Meaning
AAS	Atomic Absorption Spectrometry
Ag	Silver
AHP	Analytical Hierarchy Process
As	Arsenic
ASIC	All-In-Sustaining Costs
Au	Gold
BFS	Bankable Feasibility Study
BLEG	Bulk Leach Extractable Gold
BoQ	Bill of Quantities
BSTP	Biological Sewerage Treatment Plant
CAPEX	Capital expenditure
CIL	Carbon in Leach
COV	Coefficient of Variation
Cu	Copper
DC	Diamond Core
DDH	Diamond Drill Hole
DF	Design Feasibility
DFS	Definitive Feasibility Study
DRA	DRA Americas Inc.
EIS	Environmental Impact Statement
EPA	Environmental Protection Agency
ESIA	Environmental and Social Impact Assessment
FEED	Front End Engineering Design
FEL	Front End Loader
FOS	Factor of Safety
FS	Feasibility Study
g	Grams
G&A	General & Administration
g/t	Grams per metric tonne
Ha	Hectare
Hg	Mercury
ICMC	International Cyanide Management Code
IK	Indicator Kriging
IMR	In-situ Mineral Resource
IP	Induced Potential
KE	Kriging efficiency
Kg	Kilogram
Km	Kilometres
Koz	Kilo ounce (troy)
l	Litres
Li's	Legislative Instruments
LoM	Life of Mine
LUC	Localised Uniform Conditioning
LVD	Land Valuation Division
m	Meters
Ma	Million years
mAMSL	Meters above mean sea level
MCC's	Motor Control Centres
MoM&E	Serbian Ministry of Mining and Energy
MOP	Mining Optimisation Process
MRE	Mineral Resource Estimate
MRev	Mineral Reserve Estimate
MSA	Mining Services Area
Mtpa	Million tonnes per annum
NPV	Net Present Value
NS	Nitro Shear
OLC	Overland Conveyor

Abbreviation	Meaning
OPD	Out Patient Department
OPEX	Operating cost
OK	Ordinary Kriging
Oz	Troy ounce (31.1034768 grams)
Pb	Lead
PFC	Power Factor Correction
PFS	Pre-feasibility Study
PSA	Pressure Swing Absorption
QA/QC	Quality Assurance / Quality Control
QP	Qualified Persons
RAP	Resettlement Action Plan
RC	Reverse Circulation
RF	Revenue Factor
RFQ	Request for Quote
RoM	Run of Mine
RQD	Rock Quality Designation
SABC	SAG and Ball Milling Circuit
SAG	Semi Autogenous Grinding
SIB	Stay-in-business Capital

Abbreviation	Meaning
SMU	Selected Mining Unit
STP	Sewage Treatment Plants
DEGN	Degrees relative to true north
t	Tonnes
Tpa	Tonnes per annum
TSF	Tailings Storage Facility
TSS	Total Suspended Solids
TSX	Toronto Stock Exchange
UC	Uniform Conditioning
UCS	Unconfined Compressive Strength
US$	United States Dollars
VLF	Very Low Frequency
VRA	Volta River Authority
VTEM	Versatile Time-Domain Electromagnetic Surveying
WAD	Weak Acid Dissociable
WBSZ	Western Bounding Shear Zone
WRD	Waste Rock Dump
WRDF	Waste Rock Dump Facility
Zn	Zinc

3 RELIANCE ON OTHER EXPERTS

The QPs prepared this Report using reports and documents as noted in Section 27. The authors wish to make clear that they are QPs only in respect to the areas in this Report identified in their “Certificates of Qualified Person”, submitted with this Report to the Canadian Securities Administrators.

The QPs of this Report are not qualified to provide extensive commentary on legal issues associated with DPM’s Serbian operations, or the legal rights to the mineral properties. DPM has provided certain information, reports and data to DRA in preparing this Report which, to the best of DPM’s knowledge and understanding, is complete, accurate and true.

The QPs who prepared this Report relied on information provided by experts who are not QPs. The QPs who authored the sections in this Report believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the Technical Report.

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This Report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

DRA relied on the Company to describe matters relating property, ownership, exploration, Serbian tax, royalties, and government levies. DRA has relied upon market related information provided by the Company. DRA relied on the Company’s representatives to describe the following sections:

- 4.2 Property Description;
- 4.3 Mineral Tenure;
- 4.4 Exploration;
- 4.5 Royalties Obligations;
- 4.6 Permits, Environmental Liabilities and Risks;
- 19.3 Commodity Price Projections;
- 19.4 Contracts.

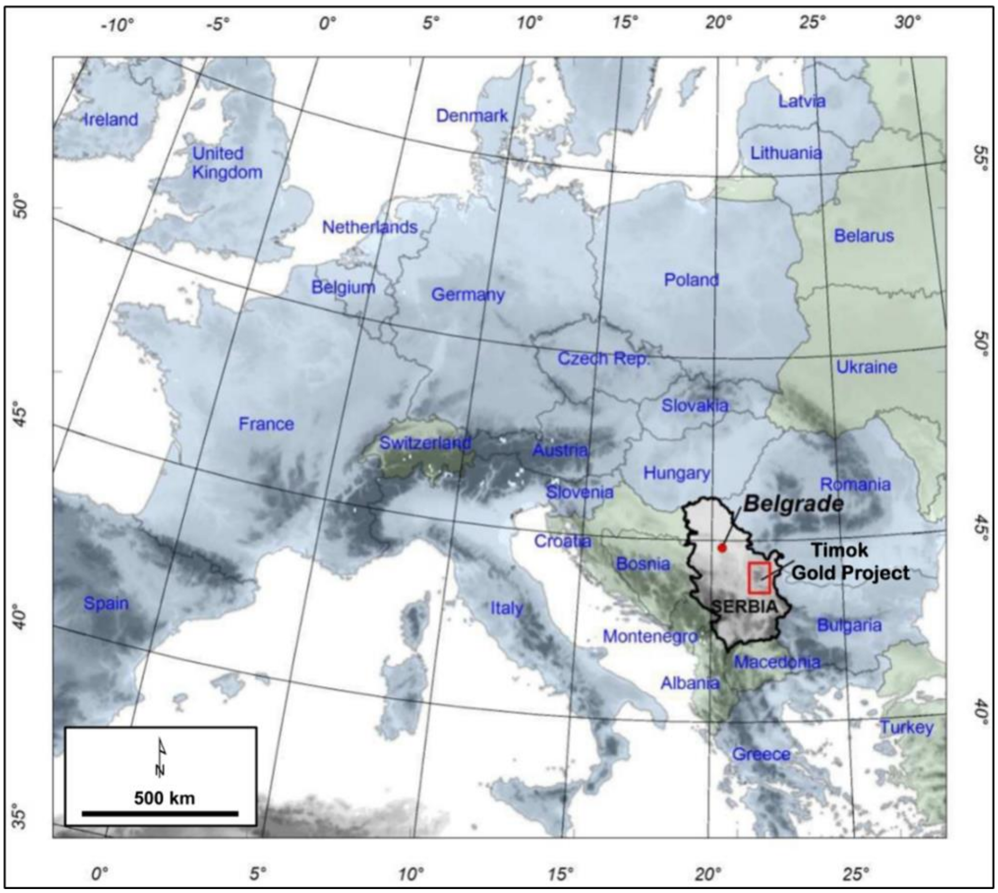
The mentioned sections have not been idenpendantly reviewed by DRA, DRA did not seek independent legal review of these items.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Project is located in the eastern part of the Republic of Serbia, approximately 270 km southeast of its capital, Belgrade, as shown in Figure 4.1. Its northern boundary is positioned about 25 km south from the Danube River and the Project area extends 24 km southwards to a point approximately 14 km west and southwest of Bor at its southern boundary. The main deposits on the Project are located approximately 25 km northwest of the town of Bor, Serbia. Bor is a historical centre for copper mining and smelting in Serbia.

Figure 4.1 – Location Map – Timok Gold Project

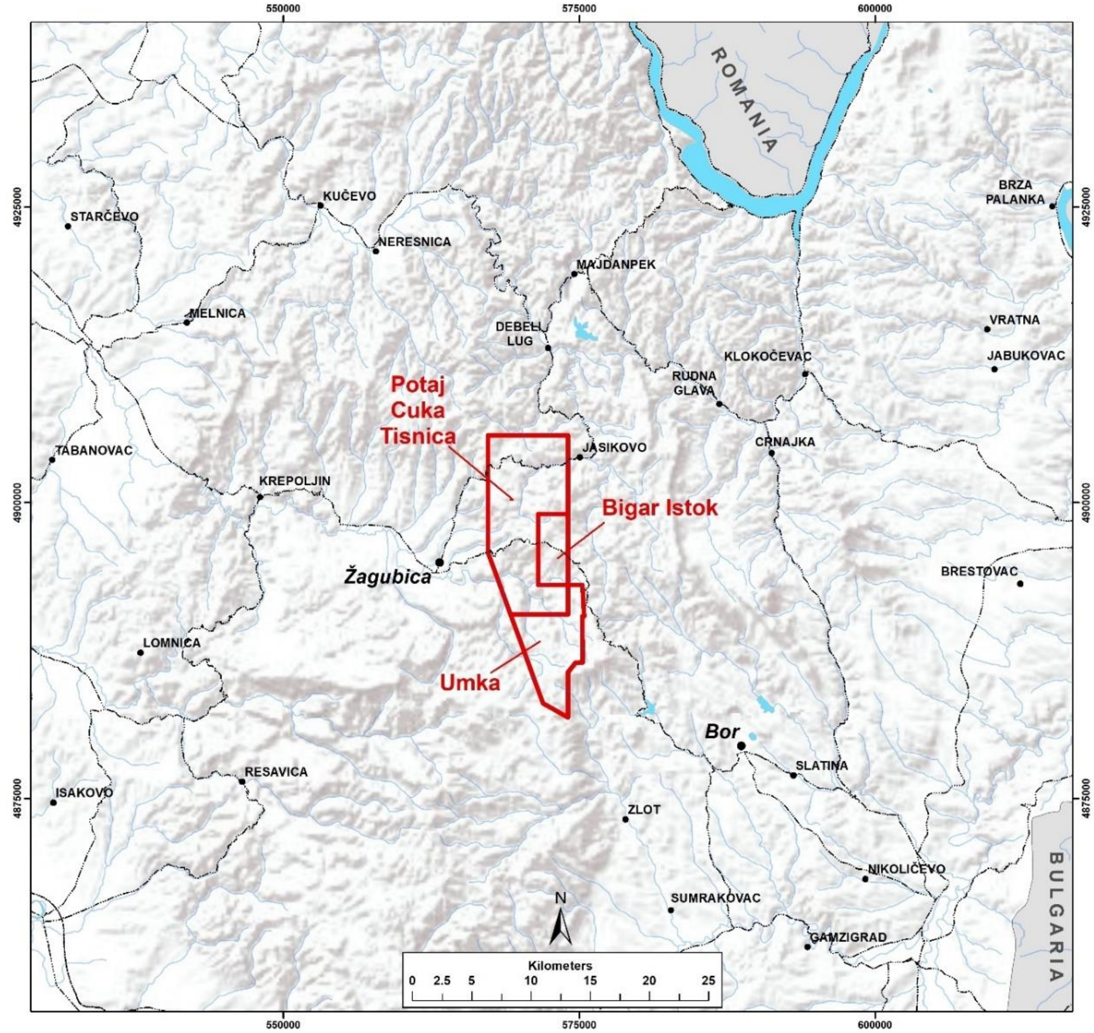


(Source: Avala, 2018)

4.2 Property Description

The Project comprises three (3) exploration licences (Potaj Čuka Tisnica, Umka and Bigar Istok) covering an aggregate area of 131.21 km². Locations of the exploration licences are shown in Figure 4.2. The Bigar Hill, Korkan, Korkan West, and Kraku Pester deposits, which are the subject of this Report, are located within the boundary of the Potaj Čuka Tisnica exploration licence.

Figure 4.2 – Timok Gold Project Exploration Licences



(Source: DPM, 2020)

Exploration licences are currently granted by decisions of the Serbian Ministry of Mining and Energy (MoM&E). They are generally issued on an initial three-year basis and are twice renewable for a further period of three years (first renewal), followed by a period of two years (second renewal), for a total potential term of eight years. An integral part of the exploration licence application and renewal process is submission of a detailed exploration work program. Supporting documentation is also required from the Institute for the Preservation of Cultural Heritage and the Institute for Nature Conservation of Serbia to the effect that the proposed exploration activity is in accordance with Republic of Serbia's environmental and cultural legislation. The obligations of the licence holder are to complete the submitted and approved work program, provide annual exploration activity reports to the MoM&E, and advance the geological knowledge of the property.

Exploration licences can be renewed if the exploration licence holder fulfils its obligations, including the completion of at least 75% of the planned work program. The legislation provides for a clear development process, from discovery through to mine development and operation.

To retain the licences beyond the final two-year extension period, a similar application can be made to request a reservation of the exploration licences for a further three-year period, during which permitting activities may take place.

4.2.1 Ownership

The exploration licences for the Project are held by Avala Resources d.o.o., a Serbian registered, wholly owned subsidiary of DPM, following the amalgamation of a wholly owned subsidiary of DPM with Avala Resources Ltd. in April 2016.

The Potaj Čuka Tisnica and Bigar Istok exploration licences were renewed (second renewal) in July 2019 and are valid until July 2021. The Umka exploration licence was renewed in August 2019 for a further three years (first renewal).

Details of each of the exploration licences are outlined in Table 4.1. The expenditure commitments for keeping the exploration licences in good standing and eligible for renewal at the end of each respective licence period are summarised in Table 4.1. DPM fulfilled all its commitments on the licences renewed in 2019 and fully expects to fulfil all obligated commitments on the licence extensions to maintain the Timok exploration licences in good standing. Upon the expiration of the Timok exploration licenses, DPM is entitled to secure rights to the ground to allow for permitting activities. Permitting is discussed further in Section 20.

Table 4.1 – Tenement Details for Timok Gold Project Exploration Licences

Licence	Licence Number	Holder	Grant Date	Expiry Date	Area (km^2)	Expenditure Commitment* (EUR)
Potaj Čuka Tisnica	310-02-0121/2006-06	Avala Resources d.o.o.	20-Jun-2006	22 Jul 2021	80.38	5,617,333
Bigar Istok	310-02-0262/2013-03	Avala Resources d.o.o.	05-Mar-2014	22 Jul 2021	15	460,740
Umka	310-02-01413/2015-02	Avala Resources d.o.o.	25-Mar-2016	14 Aug 2022	35.83	833,855

* Expenditure commitment relates to the full work program (covering the period from the grant date to the expiry date) as submitted to the Serbian Ministry. The Company is required to meet 75% of this commitment for the licence to be eligible for renewal after the expiry date.

(Source: Avala, 2019)

4.3 Mineral Tenure

There are no other known agreements or encumbrances on the properties. DPM operates with the permission of the MoM&E, in conjunction with the Ministry of Environmental Protection, and the Ministry of Culture and the Media of the Republic of Serbia.

DPM does not currently own the surface rights to any of the land parcels located on the exploration licences. To gain access to the land to conduct exploration activities, land access agreements are negotiated with the local landowners, in the case of privately held land, or with the state, in the case of state land. These land access agreements follow Serbian legislative requirements in terms of proscribed compensation for access and land disturbance, etc. The land access agreements are recorded in a master register to document and maintain transparency in negotiating and maintaining land access compensation.

4.4 Exploration Permit, Rights and Obligations

The QP knows of no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

4.5 Royalties Obligations

The Serbian government levies a royalty of 5% Net Smelter Return (NSR) for production of metallic raw materials.

4.6 Permits, Environmental Liabilities and Risks

The QP knows of no environmental liabilities to which the property is subject. In terms of the current exploration property status, no additional permits are required if the work program associated with the licence application does not fall below or exceed the plan by 25%. An addendum must be filed detailing the work program if the 25% tolerance is exceeded. In terms of future permitting process for implementation and the associated permitting plans and schedule, the reader is asked to review Section 20 of this Report.

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

5.1 Accessibility

The Project is accessible by regional asphalt roads between Bor, Žagubica, Krepoljin, and Zlot, and well-developed unpaved forestry roads. The area is also linked via Bor to Zaječar and Paraćin and via Žagubica to Požarevac (and further to Belgrade). There is a railroad from Bor to Belgrade through Požarevac.

Terrain in the Timok area is hilly to mountainous, ranging from about 500 metres above sea level (masl) to 944 masl at Čoka Rakita, the highest peak in the area. The most important drainage is the Jagnjilo River, which drains into the River Veliki Pek, and further on to the Danube, and incorporates the Bigar Hill and Korkan areas. The lower slopes and valleys are largely given over to seasonal farming, while forests dominate the higher slopes and peaks.

The town of Bor is connected by rail to Belgrade (via Požarevac). This same rail network is part of European Transportation Corridor 10, which extends southwards through the Republic of North Macedonia to Greece and the Mediterranean, and also eastwards through Bulgaria to ports on the Black Sea (and further on to Turkey). Bor is accessible via the national highway grid (Paraćin turnpike), leading to paved roads through Boljevac to Bor.

The PFS design includes for the upgrading of 5 km local roads to a 6 m wide gravel access road with reasonable slopes not exceeding 12%, allowing for delivery of major construction, mining and processing equipment to the Project site.

5.2 Climate and Vegetation

The Project area is characterised by moderate continental climate, with some influence of high mountainous climate. Winters are long and cold, with abundant snow cover, and summers are usually hot. First seasonal frosts occur in October and the last frosts are in April. Site elevations vary between 600 and 830 m. The estimates used to characterize the climate at the Project site have been obtained from long-term observations from the Crni Vrh weather station, located 13 km to the south at an elevation of 1,037 m. Data from this station is considered representative of the upper end of the site elevation range. Site estimates are that the coldest month is January, with an average temperature of -3.4°C, and the hottest month is July, with an average temperature of +17.2°C.

Annual precipitation is in the range of 500 mm to 1,130 mm, with the mean annual precipitation estimated to be 770 mm. The mean monthly precipitation at the Project area is estimated to vary from about 48 mm in both January and February to about 85 mm in both May and June. The mean annual potential evapotranspiration in the Project area is estimated to be 554 mm, varying from about 8 mm in March to about 114 mm in July.

The Project is in a hilly area, mostly forested with steep-sided narrow valleys and broad interfluves. The dominant habitat is beech woodland, interspersed with agricultural land comprising pasture and orchards with scattered homesteads (most seasonally occupied but now often abandoned). The majority of agricultural land was grazing pasture and is now unused, mainly reverting to meadow, and supports good species diversity. Much of the woodland present shows signs of harvesting for timber production; some areas are composed of mature woodland and likely support high species diversity. Several small rivers cross the site, most being tributaries to the main river the Jagnjilo, part of the Danube watershed and support a range of aquatic ecologies. Many ephemeral riverbeds occur in valley floors around the site, likely seasonal watercourses fed by spring snow melt and by limestone springs.

The Project is planned to operate throughout the year with no seasonal shutdowns or weather-related production reductions planned.

5.3 Local Resources and Infrastructure

While there is limited infrastructure within the mineral deposit area, there are existing power lines and networks of well-developed, gravel forestry roads. Water sourcing is planned to be by locally placed boreholes and captured surface run-off.

A site layout in Section 18 depicts the various project surface elements.

DPM does not currently own the surface rights to the entire area. A detailed surface rights procurement plan is in development to complete in the next study phase.

Habitation within the Project area is sparse and generally restricted to summer-months seasonal occupancy of rural farmsteads, although this practice is in decline. DPM has an operational base in the town of Bor (population approximately 40,000).

Bor is a historic mining centre within eastern Serbia, which has been in near-continuous operation since 1902. Currently, the majority of the population is employed by the mining company Serbia Zijin Mining d.o.o, which in December 2018 became majority owner of the previously state-owned mining group, RTB Bor, which operates the Veliki Krivelj and Cerovo open pit copper mines and the underground Borska-Jama copper-gold operation, together with the Bor smelter, all located proximal to the town.

A large proportion of the population has experience in work activities associated with mining operations, and the local availability of technical staff for any future mining operations within the region should be considered high.

Bor is accessible via the national highway grid (Paraćin turnpike), leading to paved roads through Boljevac, Petrovac to Bor, with State Roads 161 and 164 passing north of the Project area. The town of Bor is also connected by rail to Belgrade (via Požarevac); this same rail network is part of

European Transportation Corridor 10, which extends southwards through the Republic of North Macedonia to Greece and the Mediterranean, and also eastwards through Bulgaria to ports on the Black Sea (and further to Turkey).

The processing plant area, Bigar Hill, Korkan and Korkan West deposits are located approximately 3 to 4 km from the 110 kV Serbian national power grid, which extends from Bor to Petrovac and passes near the Project area.

Aggregate for concrete can be supplied by an operating plant located some 30 km west of Bigar Hill, which is in good condition and currently supplies customers across the region. Options remain available for production on site from suitable mine waste rock.

5.4 Physiography

Terrain in the Project area comprises steep, narrow valleys and rounded hilltops, ranging from about 500 masl in valleys in the northern part of the Project, to 944 masl at Coka Rakita, the highest peak in the region directly, south of the Project. Other high peaks are Coka Berbjesce (817 masl), Strez (731 masl), and Coka-Unuk (741 masl).

The processing plant and associated facilities are in the Valja Saka valley at around 680 to 700 masl, and the VHL pad is upstream of that, between 700 and 750 masl, and will reach 773 masl when complete. The pits and waste rock dumps are mainly located on the surrounding hilltops and valley sides, ranging up to 850 masl, with the final design crest of the Bigar Hill waste rock dump at 870 masl and most of the pits extend below 500 masl. The Korkan main pit and pit pond are furthest north and extend onto ground as low as 605 masl.

The Project is within the upper catchment of the Jagnjilo River, which drains northwards into the Veliki Pek, and further on to the Danube, and incorporates the entire Project infrastructure within the catchment of its upper tributaries, except for the southern extent of the haul roads. Some of the Project area is underlain by limestone, especially in small outcrops to the north, west and south, and in these areas; there are caves, swallow holes (where rivers enter subterranean cave systems), steeper, ravine-like valley sides and several springs feeding the Jagnilo and tributaries, such as the Bigar stream.

The lower slopes and valleys were largely given over to seasonal farming, although many summer grazing pastures are now abandoned to wilder meadows and encroaching woodland, while forests dominate the higher slopes and peaks.

Figure 5.1 shows a view of the Project area from Korkan, looking southwards towards Bigar. Typical physiographic landscapes and climate contrasts at the Project are shown in Figure 5.2.

Figure 5.1 – Typical Landscape – Looking South Towards Bigar Hill Deposit

Source: Avala, 2014

5.5 Heritage and Sustainability

A detailed description of the Heritage and Sustainability is found in Section 20.3 of the Report.

Figure 5.2 – Typical Seasonal Variation at the Southern Access Point of the Timok Gold Project

Source: Avala, 2014

6 HISTORY

6.1 Prior and Current Ownership

DPM has been active in minerals exploration in Serbia since 2004 and acquired several exploration licences and concessions between 2004 and 2010.

In July 2010, Avala Resources Ltd. acquired Avala Resources d.o.o. (formerly named Dundee Plemeniti Metali d.o.o.) from DPM through a reverse takeover transaction, pursuant to which DPM acquired a 51% share in Avala Resources Ltd.

In April 2016, DPM completed the acquisition of the remaining 49.9% of Avala Resources Ltd. that it did not currently own, effectively re-acquiring full ownership of the Project.

The exploration licences for the Project are held by Avala Resources d.o.o.

6.2 Exploration History

The Timok region has a long history of exploration and mining, dating back to Roman times. Key periods include:

- Mining during Roman times, as demonstrated by the discovery of slag and mining tools.
- Geological mapping commenced in 1933 by Geozavod, Belgrade, and Geology Institute Bor.
- Geophysical exploration undertaken by French prospectors in the 1930s and during various periods until 1985 by the Institute for Geological and Geophysical Exploration, Belgrade.
- Several geochemical surveys, commencing in 1958, undertaken by Geozavod, Belgrade, and Geology Institute Bor.
- Small-scale adits developed prior to World War II.
- Limited exploration, including drilling, which commenced post-World War II, by RTB Bor.
- Pits and adits of unknown chronology are scattered through the eastern and southern portions of the exploration licences.
- No production of any significance from the property has been undertaken.

Previous exploration at the Project, undertaken from 2007 to 2009, has been summarised by Coffey Mining (2010). DPM is not aware of any exploration for gold taking place within the Project area prior to 2007. Prior to DPM's involvement from July 2010, there are no historical Mineral Resource and Mineral Reserve estimates for the Project.

Extensive soil sampling and surface trenching programs were carried out during the 2007 to 2009 period. Four (4) (581.7 m) diamond core drill-holes and 152 trenches (28,014.6 m for 14,138 samples) were completed on the Project, though much of this was outside the four (4) deposits that are the subject of this Report (Bigar Hill, Korkan and Korkan West, and Kraku Pester).

Avala then focused exploration drilling campaigns from 2010 to 2013 on the Potaj Čuka Tisnica licence to outline mineralisation on the Bigar Hill, Korkan, Kraku Pester and Umka areas. The drilling that relates to Bigar Hill, Korkan and Kraku Pester is covered in more detail in Section 10. Along with drilling, from 2010 onwards, outcrop, soil and trench sampling were conducted. After 2014, several exploration trenches, channels and drill-holes were completed on wide-spaced grids on areas peripheral to the mineralised prospects.

After Avala was fully re-acquired by DPM in 2016, a near-resource target generation exercise was undertaken, which led to the discovery of the Korkan West Deposit during winter 2016/2017.

DPM is conducting further in-fill drilling, exploration and geotechnical investigation work during the ongoing 2020/2021 drilling campaign. Initial drilling results have identified three (3) new mineralised zones in the vicinity of the Bigar Hill pit that will require further exploration. These mineralised zones were identified during the 2020 PFS phase and will require further investigation during the next phase of the Project.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Project is located within the north-western part of the TMC in eastern Serbia. The TMC is part of the greater (ABCD) metallogenic-geodynamic province (Figure 7.1), which, in turn, is part of the Tethyan (or Alpine-Himalayan) orogenic system that extends from Western Europe to South-East Asia. The orogen resulted from the convergence and collision of the Indian, Arabian, and African plates with Eurasia, initially in the Cretaceous and continuing today.

The complex arcuate geometry of the collision interface, and the presence of several micro-plates within the orogenic collage, resulted in a variety of collision products (Gallhofer et al., 2015). Some segments are characterised by extensive regional metamorphism, whereas others by calc-alkaline igneous activity. The structural complexity and present-day geometry of the region reflects large-scale oroclinal bending during post-collision tectonics throughout the Tertiary, including major transcurrent fault systems with overall dextral displacements exceeding 100 km (Knaak et al., 2016).

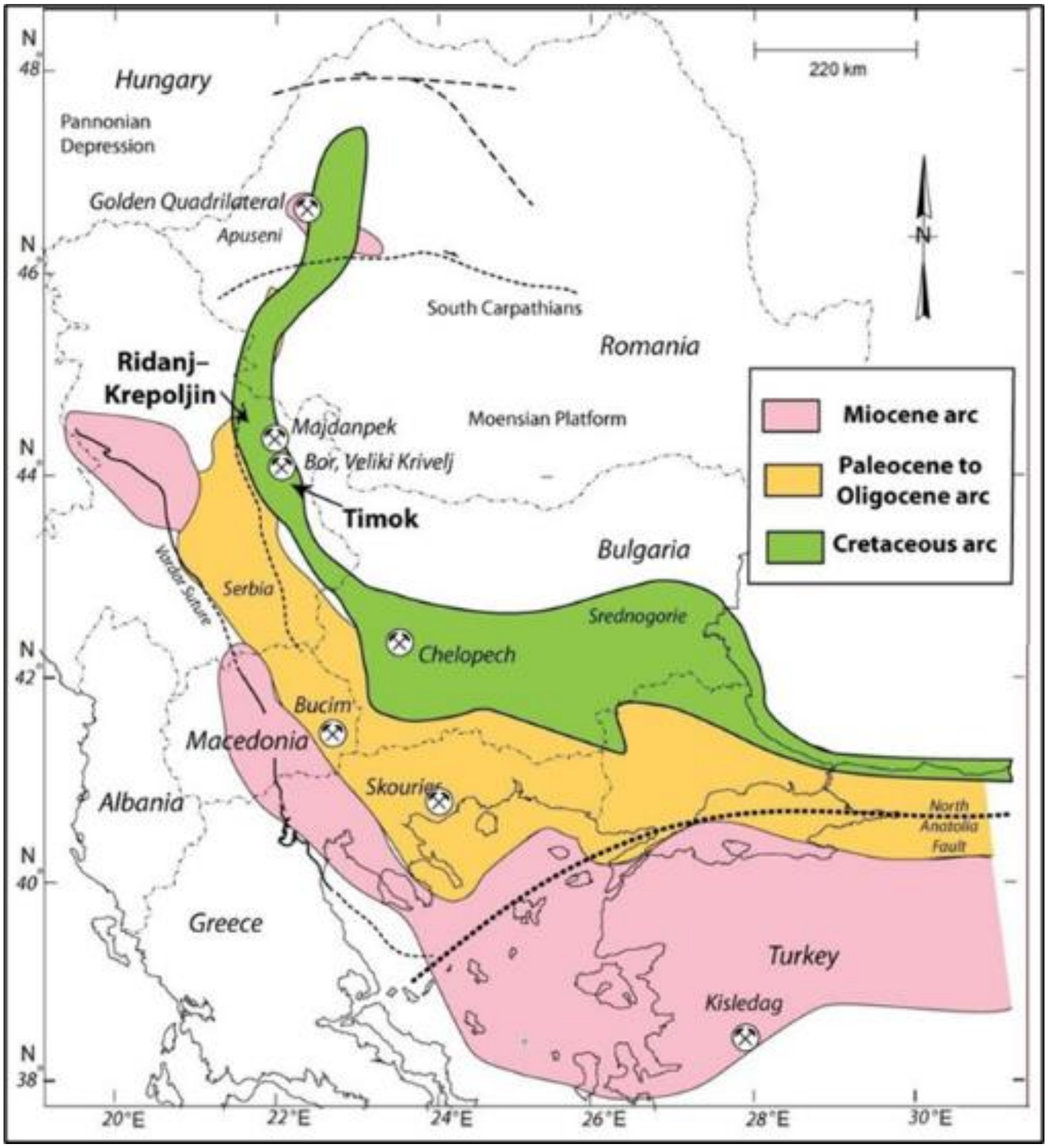
Orogenic segmentation resulted in a discontinuous distribution of mineral deposits within the ABCD province and limited the lateral extents of the various metallogenic belts along the trace of the orogen. These Late Cretaceous to Miocene belts and adjacent segments host significant porphyry copper-gold deposits with related high sulphidation copper-gold mineralisation. The major deposits within these belts are Skouries, Chelopech, Bor, Veliki Krivelj, and Majdanpek, as well as many deposits in the Golden Quadrilateral of Romania.

Within the ABCD province, the most economically significant segment comprises the Upper Cretaceous subduction-related magmatic rocks and mineral deposits, referred as the Apuseni-Banat-Timok-Srednogorie Belt (ABTSB). This L-shaped belt extends from Romania, through Serbia, and into Bulgaria. Plate reconstructions show that the ABTSB originally had an east-west orientation in Late Cretaceous times (Gallhofer et al., 2015 and references therein).

The structural complexity, the present-day L-shape geometry of the region and clockwise rotation (~30°) of the TMC segment reflects large-scale oroclinal bending during post-collision escape tectonics throughout the Tertiary, including major transcurrent fault systems with an overall dextral displacement in excess of 100 km and associated alternating transpressive and transtensional episodes.

Intrusive and extrusive rocks of the ABTSB were emplaced during a 30 million-year (Ma) period from ~90 Ma to 60 Ma and may have been associated with several different subduction zones of varying polarity (Gallhofer et al., 2015). The easternmost magmatic complex in Serbia, the TMC, bounds the Project area on the east.

Figure 7.1 – Tectonics and Chronology of the ABCD Province



(Source: AMEC, 2014)

7.2 Regional Structural Geology

Several fault populations of various inferred ages-of-formation have been identified in the TMC, characterised by relatively more intense development of strike length and density on the western margin of the TMC. All these fault populations are interpreted as products of Cenozoic (Alpine) transpression. From oldest to youngest, the populations constitute:

- Palaeozoic/Mesozoic faulting of metamorphic basement rocks. These faults were undoubtedly reactivated during syn-sedimentary TMC basin formation and subsequent emplacement of igneous intrusions.

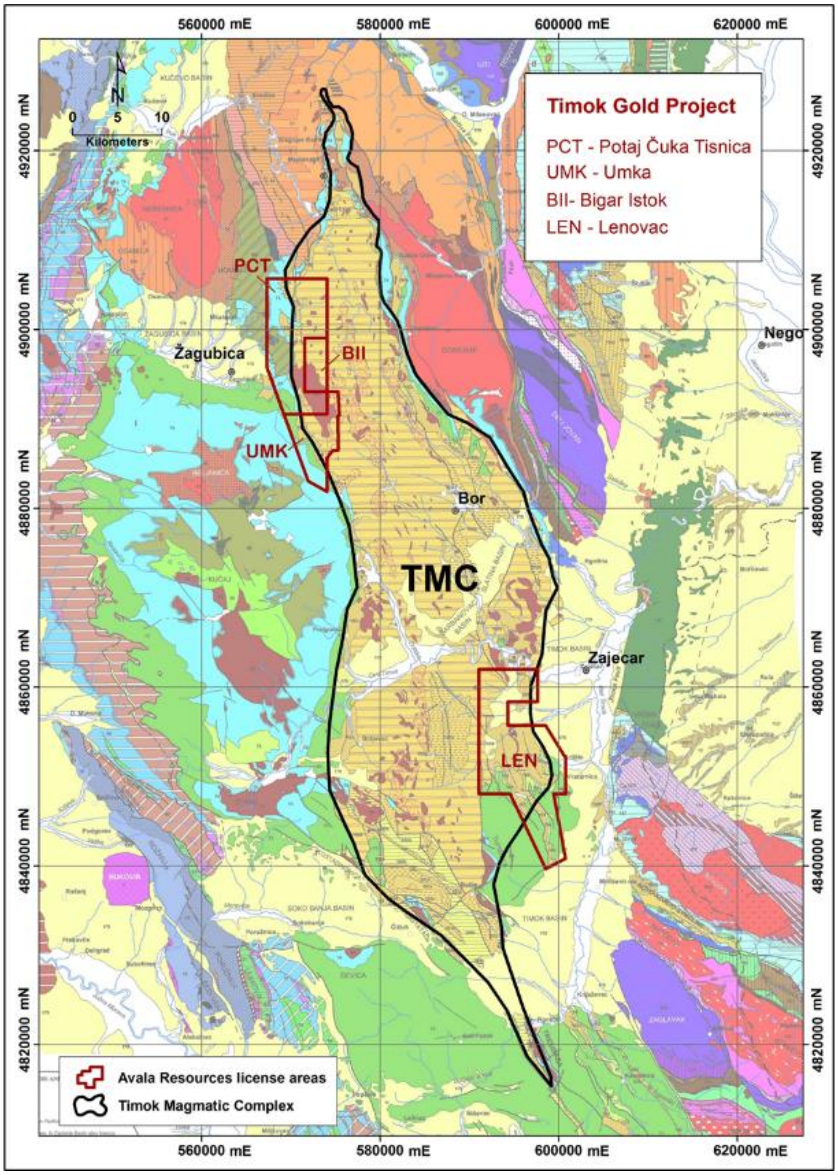
- Early Cretaceous, currently northwest-striking, dislocations that appear to have controlled basin opening. These structures are interpreted as major accommodation-structures during Eocene-Oligocene deformation.
- Late Cretaceous strike-extensive reverse faults, trending north-south to northeast-southwest. These faults were reactivated by Alpine transpression that resulted in accommodation of dextral strike-slip motion. A discontinuous easterly-dipping subpopulation of these faults is developed through the sediment-hosted gold prospects and is interpreted as having been a single structure prior to disruption by subsequent deformation. This feature is defined as a domain-bounding structure and is discussed below. Geology maps at 1:25,000 scale show north-trending, east-dipping reverse faults as part of a larger north-trending reverse fault system at the north-western margin of the TMC.
- Evidence for reverse movement is expressed as repetition/imbrication of stratigraphy and is also associated with local folding and variation in the dip of stratigraphic layering. Northeast-striking faults locally post-date sedimentary rock-hosted mineralisation, as evidenced by their intersection and offset of the margins of the Potaj Čuka monzonite, although the degree to which this can be attributed to fault reactivation is unknown.
- Eocene to Oligocene northwest-striking, strike-slip faults that hosted sinistral movement as a result of oroclinal bending. These structures constrain numerous regionally pervasive, short strike-length northeast-trending faults that are typically expressed as topographic lows.
- Late normal faults are responsible for the geometry of features such as the Miocene Žagubica Basin, which contains approximately 2,000 m of sedimentary infill. These structures extend eastward into Bigar Hill and offset the mineralised system. Similar faults are present at Korkan, but their trace is complicated due to the presence of numerous northwest-striking faults that are also post-mineral. Regionally developed east-west striking faults of variable strike length are expressed as discrete brittle structures at all scales and crosscut all other structural features.

Despite the age relationships indicated above, the assignment of individual faults to populations of particular ages is difficult. Surface expressions of faults are uncommon, and crosscutting relationships are rarely conclusive. Furthermore, a diversity of fault orientations is present, due to different ages of faulting, shifting far-field stress geometries over time, reactivation of older faults, and the role of pre-existing architecture during the formation of each successive stage of faulting. A critical element in the identification of faults has been the resolution of a consistent stratigraphic framework – the components of which can be identified regionally.

7.3 Local Geology

In eastern Serbia, magmatic activity of the Late Cretaceous ABTSB is developed along two (2) subparallel north-trending branches; the narrow Ridanj-Krepoljin Belt to the west, and the wider TMC to the east. The latter branch contains the Bor and Cukaru Peki high-sulphidation type epithermal copper-gold deposits, and hosts major porphyry copper deposits (Majdanpek, Veliki Krivelj) and several Late Cretaceous epithermal occurrences (e.g. Lipa). The TMC is approximately 85 km long and extends from Majdanpek in the north to Bučje in the south. The disposition of the Project's exploration licences and the local geology are shown in Figure 7.2.

Figure 7.2 – Exploration Licences with the TMC



(Source Avala, 2018)

The Late Cretaceous TMC developed on a continental crust composed of different fault-bounded terranes composed of Proterozoic metamorphic to Lower Cretaceous rocks. The area is now incorporated in the Getic Nappe or the Kučaj Terrane, as part of the complex Carpathian-Balkan Terrane in eastern Serbia. Upper Jurassic and Lower Cretaceous shallow marine sedimentary rocks, dominated by homogeneous, massive to bedded limestone and marl, unconformably overlie a metamorphic basement. Carbonate sedimentation terminated in the Early Cretaceous due to the impact of the Austrian deformational phase, which caused weak deformation, uplift, erosion, and subsequent paleokarst formation.

Clastic sedimentation commenced with an Albian transgression, unconformably burying the partially eroded and faulted carbonate platform rocks. These calcareous clastic rocks mark the start of the evolution of the TMC, beginning with Austrian deformation and followed by deformation in the Late Cretaceous (Albian). They outcrop along the eastern and western boundary of the TMC but rarely in the central part. Sedimentation continued through the Cenomanian, with an increasingly volcanic detrital component becoming important with decreasing age. During the Turonian, volcanism commenced, and progressed from east-to-west across the TMC. At this time, the TMC became a topographically positive volcanic area.

Contemporaneous sedimentation, magmatism, and hydrothermal activity were relatively continuous within the TMC throughout the entire Late Cretaceous, as illustrated in Figure 7.3. The sedimentation persisted from the Albian to the Maastrichtian. Late Cretaceous magmatic activity has been documented during a 10-millon-year period from ~89 Ma to 78 Ma and has been interpreted to generally progress from east to west, younging across strike towards the subduction zone. This process can be related to an arc under extension and gradual steepening and rollback of a northward subducting lithosphere slab, derived from the Vardar ocean.

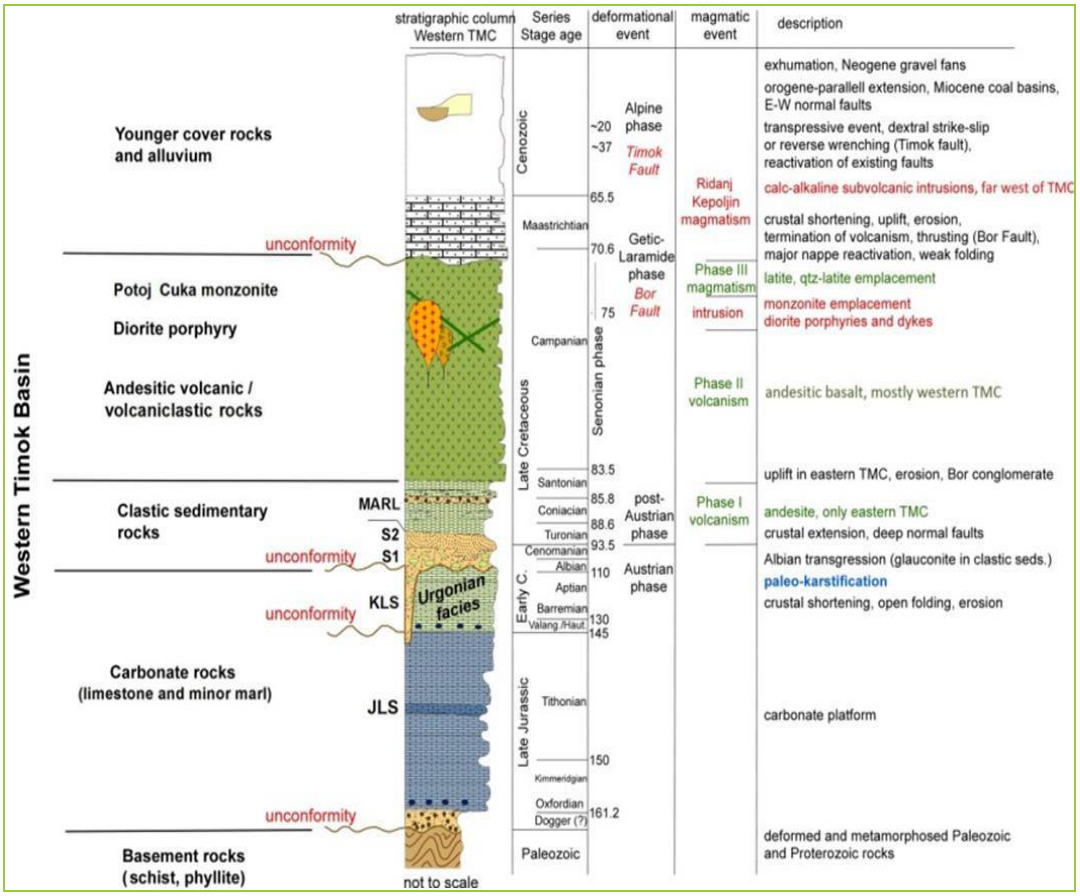
The TMC is dominated by alkaline to high-potassium calc-alkaline magmatic rocks, classically divided into three successive volcanic sequences (commonly referred to as V1 to V3 or Phase 1 to Phase 3) that are intercalated with Late Cretaceous volcaniclastic sedimentary rocks. Diorite dykes and sills are common, but locally difficult to distinguish from the volcanic supracrustal rocks. The first phase of volcanism commenced during the lower Turonian with mainly hornblende andesitic magmatic rocks in the easternmost (present coordinates) parts of the TMC. Cessation of volcanism in the early Campanian and uplift and erosion of the eastern part of the TMC were followed by local turbiditic deposition of the Bor pelites. Magmatism shifted westward into the central and western parts of the TMC during the Santonian.

A second phase of magmatism is represented by two (2) compositionally and geographically distinct assemblages:

- Pyroxene-bearing, subaqueous to subaerial andesitic rocks.
- A sequence of latites, trachytes and trachy-basalt dykes, restricted to the south-western part of the TMC. During Late Cretaceous (Campanian), diorite, quartz-diorite, and monzonite plutonic rocks were emplaced. Magmatism and sedimentation terminated in the upper Campanian and Maastrichtian. The coarse-grained Bor conglomerate records exhumation of the basement within the eastern TMC. Calcareous rocks were deposited in the central part of the TMC at this

time. The Upper Cretaceous rocks of the TMC are overlain by Paleogene to Neogene sedimentary rocks and deposits of quaternary sediments.

Figure 7.3 – Schematic Stratigraphy of the Western TMC



(Source: AMEC, 2014)

The structural complexity and present-day asymmetric lozenge-shaped geometry of the TMC area resulted from oroclinal bending during post-collision tectonics throughout the Tertiary. This has led to tectonic modifications of lithological contacts, including those that represent syn-depositional features, beds, or faults. The extent of deformation is commonly difficult to assess due to variable responses of different rock types to the same deformation event. Much of the deformation has been absorbed by argillaceous horizons due to their ability to accommodate shearing and shortening, whereas sandstone beds have resisted much of the deformation. Similarly, competent massive limestone units forming the base of the sequence exhibit minor deformation and much of this is expressed as fracturing near the contact with the overlying clastic sedimentary rocks.

7.4 Project Stratigraphy

The Project area at the western margin of the TMC can be subdivided into two (2) northerly-trending domains:

- Western Domain – dominated by Proterozoic metamorphic basement, Upper Jurassic and Lower Cretaceous limestones.
- Eastern Domain – dominated by the Cretaceous volcanic, epiclastic rocks, and associated diorite intrusive rocks of the TMC, including the known porphyry copper-gold centres at Valja Strz and Dumitru Potok.

The boundary between the two (2) domains is dominated by calcareous clastic sedimentary rocks, including sandstone, conglomerate, and marl, and is partly defined by a domain-bounding structure. The four identified sediment-hosted gold zones within the Potaj Čuka Tisnica licence are Bigar Hill, Korkan, Korkan West and Kraku Pester. These prospects are hosted by calcareous clastic sedimentary rocks that outcrop within the boundary zone between the two (2) domains.

The overall east-west cross-sectional geometry of the TMC is that of a complexly faulted synclinorium. The stratigraphy generally dips moderately to the east at approximately 20° to 30°, along the western margin of the TMC. This general tilting of stratigraphy at the western margin indicates that the oldest rocks outcrop in the west, and that farther east the stratigraphy becomes younger, with stratigraphically higher units dominating the outcrops. Mapping by Avala, building upon public domain geologic maps and knowledge, has defined litho-stratigraphic interpretive units which are recognised as being important to the Project and are outlined below. In Figure 7.3, these units are summarised in a stratigraphic column and correlated with geological time series, deformation, and magmatic events.

7.4.1 Palaeozoic and Proterozoic Basement

Within the Potaj Čuka Tisnica licence, the oldest outcropping rocks are Palaeozoic phyllites, a meta-sedimentary sequence composed of sandstone, shale, and conglomerate protolith, and a variety of heterogeneous Proterozoic greenschist-facies schistose quartzo-feldspathic schists and gneisses. These units, which have not been further differentiated, commonly outcrop in the cores of anticlines, but have also been encountered in the bottom of some exploration drill-holes within the Project area.

7.4.2 Carbonate Sequence, JLS and KLS

Two (2) units constitute the Upper Jurassic (JLS) and Lower Cretaceous (KLS) carbonate rocks within the Project area. The older Jurassic age unit is characterised by massive limestone, most which is dominated by bedded and massive bioclastic and micritic, white, light-grey, and light brownish reef limestone of Tithonian age. The lower parts are commonly composed of micritic limestone with concretional chert nodules, and the contact with the underlying basement is commonly faulted. Unconformably overlying the Jurassic limestone is Upper Cretaceous dark grey limestone with black concretional chert nodules, deposited during the Valanginian-Hauterivian (Vasić, 2012). Most the Lower Cretaceous rocks are well-bedded bioclastic, nodular, and stromatolitic, and locally sandy limestones deposited during the Barremian and Aptian; these are referred to as the Urgonian limestone.

The limestone units are karsted, with the massive Jurassic limestone being more susceptible to karstification than the well-bedded Urgonian limestone. Some paleokarst formed prior to deposition of the younger and unconformably overlying clastic sedimentary rocks. A typical assemblage of the units is shown in Figure 7.4. These karst areas are partly filled by syn-karst fine-grained sedimentary rocks, as well as along the upper contact with finely laminated upper Lower Cretaceous (Albian) calcareous clastic sedimentary rocks. Locally, paleokarst collapse breccia is developed, and the karsted zones are a host to gold. Recent karst is also evident.

Figure 7.4 – Typical Contact between Upper Jurassic (T) and Lower Cretaceous (V) Limestones with Black Chert Nodules

(Source: AMEC 2014)

7.4.3 Calcareous Clastic Sedimentary Rocks, S1 and S2

Three distinct units of calcareous clastic rocks unconformably overlie the carbonate sequence. Various carbonate units lie beneath the unconformity, indicating exhumation and accompanying faulting during the depositional hiatus in the Early Cretaceous. Formation of the unconformity reflects the effect of the Cretaceous Austrian orogenic event. The clastic units, stratigraphically from lowest-to-highest, include calcareous sandstone with lesser siltstone-dominated sequence, overlain by reddish and iron-rich sandstone containing abundant andesitic volcanic detritus, capped by thinly bedded ferruginous marl. Total stratigraphic thickness of this sequence ranges from 365 m to 840 m:

- S1 unit is a basal clastic sequence that was deposited during the Albian to Cenomanian time (Vasić, 2012), and consists of well-bedded, coarse- and medium-grained calcareous, weakly glauconitic sandstones and conglomerates. Clasts are dominantly angular to sub-rounded limestone fragments sourced locally, but also include a variety of well-rounded metamorphic and igneous clasts from distal sources. Conspicuous rounded white quartz pebbles form a major detrital component. Intercalated with, and locally forming a significant thickness, are black, laminated, fine-grained clastic siltstone and sandstone.

  A chaotic breccia, termed the "basal breccia", is common along the basal unconformable contact. The breccia is composed dominantly of coarse blocks and smaller cobbles and pebbles of limestone in a black, commonly sulphide-rich, fine- to medium-grained calcareous sandy matrix. Locally, the basal breccia appears bedded. Similar angular clastic rock types are present within the underlying limestone sequence at various depths below, but close to, the uppermost limestone contact.

  These clastic rocks have very irregular thicknesses, are not laterally continuous, and are inferred to represent infill of karst features. The thickness of this S1 unit can vary from between 50 m and 250 m above the unconformity. A typical core specimen from the S1 unit is presented in Figure 7.5.

- S2 unit overlies the basal clastic sandstone (S1) and is comprised of reddish, coarse- and medium-grained sandstones and conglomerates, tuffaceous clastic rocks, and air-fall tuff (S2) containing varying abundances of detrital magnetite, mafic silicate minerals, and common volcanic fragments. Pyrite, presumably diagenetic in origin, is also present. This clastic sequence, deposited at the western margin of the TMC during Cenomanian time, records the approach of the volcanic arc to the east. The thickness of the S2 sandstone unit is between 15 m and 90 m. A typical core specimen from the S2 sandstone and conglomerate is shown in Figure 7.6.

- The S1 and S2 units form the principal host to gold in the Bigar Hill, Korkan and Korkan West Deposits.

Figure 7.5 – Typical S1 Unit: Fine-Grained Calcirudite with Stylolites

(Source: AMEC, 2014)

Figure 7.6 – Typical S2 Unit: Conglomeratic Sandstone with Characteristic Red Fragmental Clasts

(Source: AMEC, 2014)

7.4.4 MARL

This unit, overlying the S2 and deposited during Santonian time, is a grey marlstone that is typically finely laminated. The marl unit is interbedded with locally present sandstone, andesite, and andesite volcaniclastic rocks (Vasić, 2012). These rocks, ranging in thickness from 50 m to 500 m, are rarely mineralised, and an example is shown in Figure 7.7. In the Bigar Hill Deposit area, pyroxene-hornblende diorite sills are emplaced into the marl.

Figure 7.7 – Example of Marl Unit: Grey Marlstone with Deformed Laminations, Drill-Hole BHDD044, 59.2 m

(Source: AMEC, 2014)

7.4.5 Andesitic Epiclastics and Diorite Intrusions

This unit, of Late Cretaceous age and overlying clastic units S1, S2, and the marl, is comprised of andesitic shallow intrusive and derivative epiclastic rocks. Rapid facies changes along strike characterise the sequence. The lower part of the epiclastic unit is characterised by polymictic basaltic andesite conglomerate and breccia, whereas the upper part is dominated by monomictic basaltic andesite breccia and conglomerate, which are interpreted being products of epiclastic debris flow deposits.

A finer grained sedimentary rock unit, consisting of well-bedded tuff, marl, sandstone, and volcaniclastic breccia that locally forms a thin, but mappable horizon between the debris flow deposits. The dioritic stocks, dikes, and sill-like intrusions are generally aligned along a north-westerly trend, which most likely represents a structural fabric in the subsurface that controlled their emplacement. An example of this unit is presented in Figure 7.8.

7.4.6 Potaj Čuka Monzonite

This unit comprises coarse-grained equigranular monzonite with visible alkali feldspar phenocrysts, biotite, and minor magnetite and pyroxene. This monzonite is part of the Late Cretaceous Potaj Čuka pluton which, in the latest Cretaceous (79.8±0.6 Ma; uranium-lead in zircon), intrudes the clastic sedimentary units in the region. The Potaj Čuka pluton is located immediately east of the western margin of the TMC and is elongated in a north-westerly orientation.

Figure 7.8 – Example of Andesite Intrusive Unit Sill with Phenocrysts of Hornblende and Plagioclase, Drill-Hole BHD010, 101 m

(Source: AMEC, 2014)

7.5 Structural Geology

This subsection contains descriptions of the regional geological structure and tectonic-stratigraphic relationships of the region.

7.5.1 Structure

The formation of the basin which hosts the TMC is associated with the Alpine Orogeny, which occurred during oblique convergence of the Indian, Arabian, and African plates with Eurasia. Convergence began in the early Cretaceous resulting in an orogenic collage that is characterised by discrete segments that have undergone a distinct geologic evolution. Major phases of mountain building associated with the Alpine Orogeny were ongoing in the Late Cretaceous to Miocene.

The TMC is generally considered to represent a basin which has an overall disjointed, elongate lozenge shape, with apparent sinistral, northwest-striking dislocations. These dislocations appear to have controlled basin opening as well as modified the geometry of the TMC. The regional-scale northwest dislocations were second-order structures to an overall dextral, orogen-scale motion resulting from Eocene-Miocene oroclinal bending.

The interpreted overall east-west cross-sectional geometry of the TMC is that of a synclinorium. Tertiary Alpine deformation was accommodated by several suites of fault zones that are developed across the entire TMC. Regional cross-sections confirm that the bulk of Alpine deformation was concentrated on the TMC margins, whereas the central part of the magmatic complex was only affected by gentle folding and fault reactivation.

Accommodation of deformation by ductile deformation is largely restricted to the eastern and western margins of the TMC and was long-lived, as indicated by open folds and rotation of bedding to sub-vertical dips. Marl units north of the Korkan prospect display vertical dips in road exposures. An east-west cross section of the basin displays strain accumulation toward the eastern and western TMC margins, with inferred synclines and anticlines separated by faults that have accommodated complex movement histories.

Post-mineral structures are interpreted as being active during Cenozoic transpressional deformation. These structures include pre-sedimentary rock-hosted gold-bearing structures that were reactivated in addition to newly formed, post-mineral faults. Late orogen-parallel extension produced early Miocene normal faults that controlled the architecture of Miocene coal basins, such as the Žagubica Basin, and numerous regional east-west trending normal faults extending into the TMC. Examples of all these structures occur at Bigar Hill and Korkan, where northwest-trending strike-slip faults, reactivated north-south to north-northeast striking strike-slip faults, and east-west trending normal faults are developed.

7.5.2 TECTONIC-STRATIGRAPHIC RELATIONSHIPS

The spatial relationships between mineralisation styles in the Timok region suggest that the region might be composed of successively westward-migrating metallogenic belts. The belts are temporally distinct events, lying from east-to-west, beginning with the Bor-Veliki Krivelj Belt, succeeded by younger Kuruga high-sulphidation belt, and the still younger Timok diorite porphyry belt. These younger porphyry copper-gold prospects, including the Valja Štrz, the Dumitru Potok, Crna Reka porphyry copper-gold and Čoka Rakita porphyry gold prospects, are present in the eastern part of the Project area.

Evidence for sedimentary-hosted gold along the western boundary of the TMC extends over a strike length of more than 30 km and is up to 8 km wide. The mineralised belt was initially identified by soil geochemistry programs conducted by DPM. The geology, geochemistry, and available drill intersections of known prospects suggest a strong similarity to the sedimentary rock-hosted or Carlin-style deposits.

Most of Avala's exploration property is located on the margins of the TMC, namely the Potaj Čuka Tisnica, Bigar Istok and Umka licences (Figure 7.2) on the western margin, and the Lenovac licence on the eastern margin of the TMC. Upper Jurassic to Upper Cretaceous calcareous rocks, including limestone, marl, and calcareous clastic and volcaniclastic rocks, capped by TMC volcanic and derivative clastic rocks, underlie most of the licences. Several overprinting generations of fault systems disrupt the stratigraphy and caused structural complexity, including local reverse faulting and thrusting of stratigraphic units of the TMC area.

The sediment-hosted gold prospects: Bigar Hill, Korkan, Kraku Pester and Korkan West, are all part of the Potaj Čuka Tisnica licence and are located on north-south to north-northwest trending segments of the western margin of the TMC, centred around the Late Cretaceous Potaj Čuka monzonite batholith. Exploration within the other licences has been limited to soil geochemistry, trench sampling and scout diamond drilling.

7.6 Metamorphism

Thermally metamorphosed rocks are present in the contact aureole of the Late Cretaceous Potaj Čuka monzonitic pluton. Calc-silicate skarns are locally present. The most evident thermal effect is present at the Kraku Pester prospect and south of Bigar Hill. At Kraku Pester, the fine-grained clastic sequences adjacent to the pluton are converted to biotite-magnetite and calc-silicate hornfels, depending upon the protolith composition. South of Bigar Hill and the southern-bounding, post-mineral normal fault, the carbonate rocks are converted to marble near the monzonite, with distal bleaching of the normally grey limestone to white colours in outcrop.

7.7 Alteration

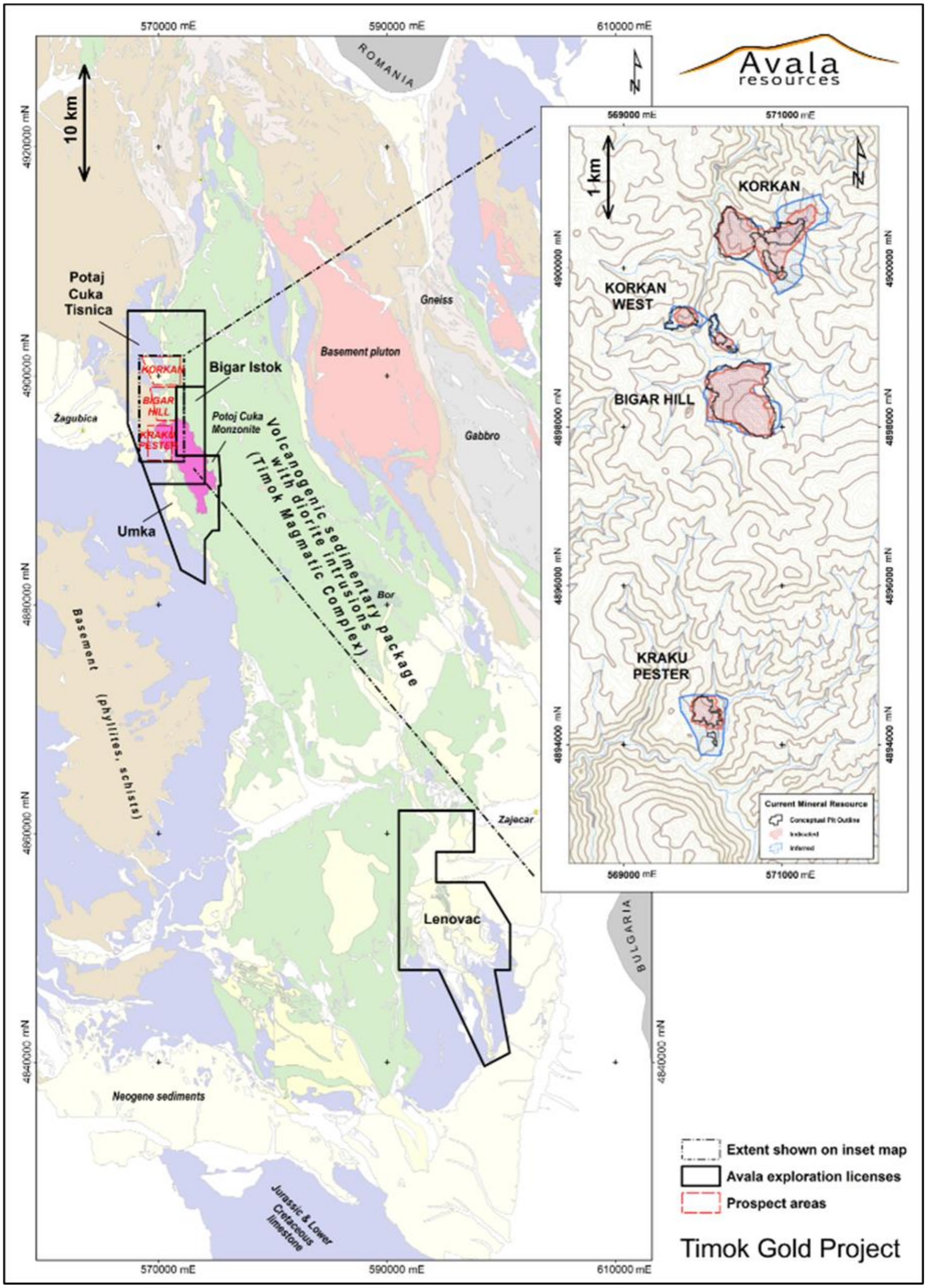
Except for the quartz-bearing zones in the andesite sill at Bigar Hill, no macroscopically visible silicate alteration minerals are evident. However, the fine grain size of the horizons, coupled with the common evidence for additional post-mineral brecciation precludes easy identification of silicate alteration minerals. Analysis of the geochemical characteristics of the mineralised horizons at Kraku Pester using 1 m drill-hole data suggests that clay minerals, presumably combinations of kaolinite, illite, and probably smectitic clays, from part of the hydrothermal alteration associated with pyrite deposition.

Elevated gold is also associated with rocks containing sufficient iron, thus suggesting that the depositional mechanism for gold was likely the sulphidation of iron present in the rocks. The recognition of auriferous concentrations in karst infill sedimentary rocks is consistent with this interpretation as iron is a common residual element during carbonate dissolution. Decarbonisation of diagenetic and detrital carbonate is associated with gold zones.

7.8 Mineralisation

Four (4) important mineralised zones have been identified within the Potaj Čuka Tisnica exploration licence. These areas comprise the Bigar Hill, the Korkan, the Korkan West, and the Kraku Pester Deposits, and are summarised in Figure 7.9. All four (4) zones share a similarity with the style of mineralisation defined at the Bigar Hill Deposit and are associated with a large hydrothermal system that has been identified within the Project.

Figure 7.9 – Exploration Areas and Geology of TMC



(Source: Avala, 2018) (inset: Mineral Resource Plots of the Deposits)

7.8.1 Bigar Hill Deposit

The Bigar Hill Deposit is the most advanced Mineral Resource within the Project. The deposit comprises Bigar Hill and the adjacent Bigar Au-polymetallic replacement showing (immediately south and east from Bigar Hill) and is located immediately north and outside of the thermal aureole of the Potaj Čuka monzonite. Rock types beneath Bigar Hill comprise of Proterozoic metamorphic basement, Late Jurassic and Early Cretaceous limestone, which are unconformably overlain by a Late Cretaceous clastic sequence (S1, S2, and marl), capped by Late Cretaceous andesitic volcanic and derivative epiclastic rocks. Diorite porphyry has also intruded the stratigraphic package. Bigar Hill is bounded to the north and south by east-west-striking faults that have brought the Late Cretaceous clastic units in tectonic contact with Late Jurassic limestone.

Gold mineralisation at Bigar Hill is located principally along two (2) stratigraphic horizons, with lesser amounts present along peripheral steeply dipping fractures zones within the clastic rocks and andesite sill. A lower zone is localised along the unconformable and brecciated lower contact of S1 and isolated karst-infill zones at the upper boundary of the KLS unit. The most continuous horizons lie at shallow stratigraphic levels along the contact between the S1 and S2 units forming a middle zone.

Above this zone, gold mineralisation occurs on the contacts between Marls and andesite intrusive unit and where the Marl is not present, between the andesite using and the S2 unit. Mineralisation is found within sub-horizonal vein zones and silicified breccia infills located on the contacts.

Within the andesite intrusive unit gold is found within vein zones containing auriferous pyrite within quartz infill, hosted by a larger phyllic alteration envelope. Individual veins are thin, averaging 10mm but are found with a frequency of 3-5 veins per metre. Such vein zones are steeply dipping with a North-South strike have been delineated by detailed drilling and trenching, over a strike extent of 200m and down-dip by 120m.

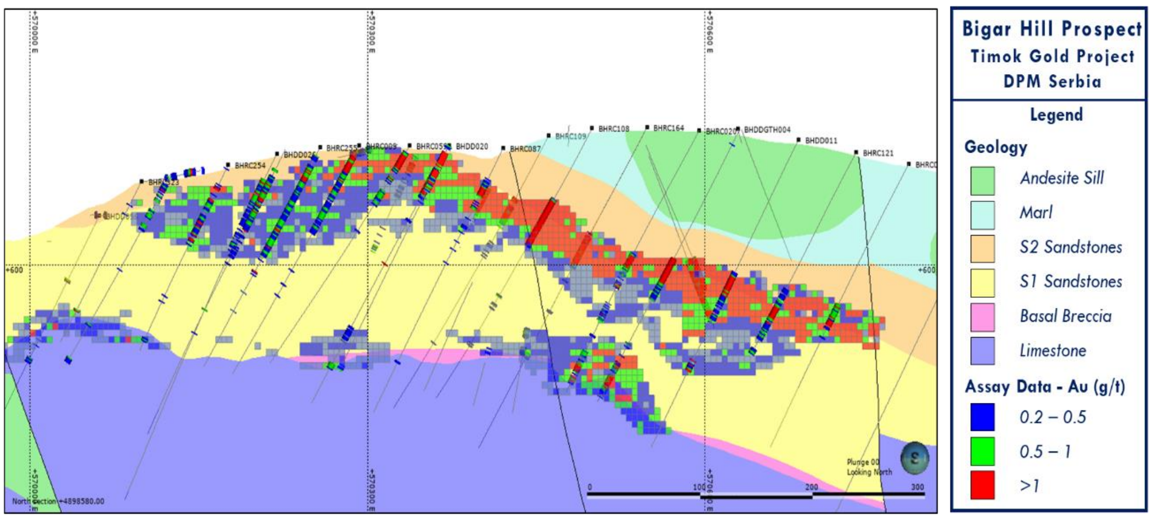
At the Bigar Hill Deposit, the highest concentrations of mineralisation are along each of the KLS/S1 and S1/S2 contacts as illustrated in Figure 7.10. Metal distribution and thickness variations of the host-rocks suggest the presence of west-northwest and northeast striking sub-vertical feeder structures.

Mineralisation is continuous and follows the dips of the stratigraphy. It has a north-south extent of approximately 900 m and an east-west extent of approximately 900 m. Mineralisation is largely from surface, and in the south its depth extent is greatest (approximately 500 m). Depth extent reduces to 200–300 m below surface moving further north. There is a small zone in the centre, where mineralisation starts from approximately 80 m vertical depth from surface.

Within the basement of the Bigar Hill area (also known as the Rapture Fault Zone), located south of Bigar Hill, is a Palaeozoic phyllite comprising a meta-sedimentary sequence composed of sandstone, shale, and conglomerate protolith. Jurassic and Cretaceous limestone are juxtaposed against the phyllite along steeply dipping, normal separation faults; on a regional basis, these rocks unconformably overlie the phyllite unit. Brecciated horizons at Bigar Hill contain clasts of intense calcite network veining, clasts of ferroan carbonate, and local veins with base metal sulphides.

A northwest-southeast trending portion of the contact zone discontinuity has localised emplacement of a dioritic porphyry intrusion. Smaller dioritic intrusions define northwest-southeast trends in the phyllite and both northwest-southeast and north-south trends in the overlying sequence.

Figure 7.10 – Cross Section of Bigar Hill Deposit



Cross Section 4898940N - Looking North

(Source: DPM, 2020)

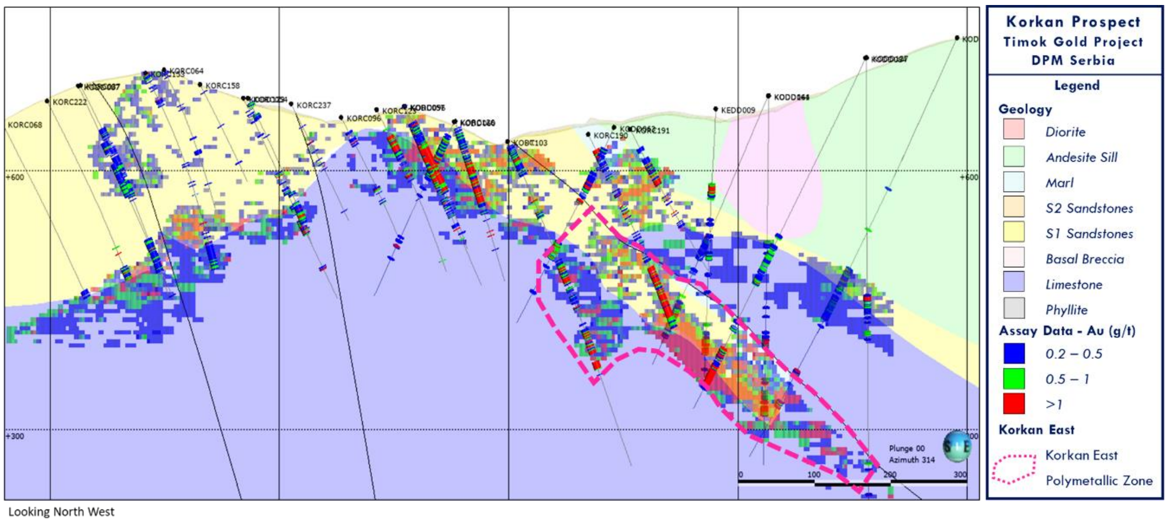
7.8.2 Korkan Deposit

The Korkan Deposit is the second most advanced exploration target within the Project, after the Bigar Hill Deposit. The deposit constitutes a generally easterly-trending zone of mineralised rocks and incorporates both the Korkan and adjacent Korkan East zones. Korkan East is located to the east of Korkan, across a braided post-mineralisation, strike-slip fault zone.

The Korkan deposit shares similar characteristics with the Bigar Hill Deposit, located 2 km to the south. Rock types in the Korkan deposit comprise Late Jurassic and Early Cretaceous limestones, which are unconformably overlain by a Late Cretaceous lower clastic sequence (S1 and lower parts of S2) and farther east also by a Late Cretaceous upper clastic sequence (upper parts of S2 and marl), capped by Late Cretaceous andesitic volcanic and derivative clastic rock.

Unlike Bigar Hill, stratiform gold mineralisation at Korkan occurs primarily along the unconformable and breccia-like lower contact zone of the clastic S1 sequence, against the underlying KLS limestone unit, and in karst-infill zones at the upper boundary of the KLS limestone unit. It is presumed that erosion has removed some mineralisation related to the S1/S2 contact that would have sat higher in the stratigraphic sequence. Korkan mineralisation along each of the KLS/S1 and S1/S2 contacts is illustrated in Figure 7.11.

Figure 7.11 – Mineralisation Cross-Sections of Korkan Deposit



(Source: DPM, 2020) (Korkan East Extension is shown in the Cross-Hatched Area on the Lower Cross Section)

Mineralisation is less continuous at Korkan compared to Bigar Hill, due to higher structural complexity. As at Bigar Hill, it tends to follow the dips of the stratigraphy. The mineralised footprint has a northeast-southwest extent of approximately 1,100 m and a northwest-southeast extent of approximately 1,100 m. Mineralisation commences from surface and can be traced to a maximum depth of 400 m below surface.

Structurally, Korkan is dominated by northwest-striking faults which, though apparently associated with mineralisation, have also dismembered mineralisation and earlier structures during late reactivation. Late east-striking faults such as those found at Bigar Hill have been recorded at Korkan but are less important in forming boundaries to the deposit and in juxtaposing stratigraphy.

Unlike Korkan proper, significant base metal sulphide minerals accompany gold mineralisation at Korkan East. Overall, the mineralised zones in this environment have the appearance of carbonate replacement deposits. Local repetition or imbrication of stratigraphy and mineralisation are related to north-northeast striking faults.

7.8.3 KORKAN WEST DEPOSIT

The Korkan West Deposit is the newest discovery within the Project. It lies between the Bigar Hill and Korkan Deposits, along a northwest trending structural corridor. The Korkan West Deposit shares many characteristics with the Bigar Hill Deposit, located approximately 1 km to the southeast, and the Korkan deposit located approximately 1 km to the northeast. Almost all mineralised intervals are manifested as oxide and transitional weathering states.

Host rocks for gold mineralisation are: (1) oxidised fine to very coarse-grained (0.1 mm to 2 mm) sandstone belonging to the S1 or S2 units; (2) conglomerate layers containing quartzite clasts and/or limestone clasts (S1 or S2 units). Mineralisation at the S2/S1 contact can commonly be observed.

The presence of several oxidised, discontinuous intervals occurring throughout the S1 or S2 units, and associated higher individual gold assays, suggests gold mineralisation may also locally be associated with structurally controlled zones.

Non-oxidised, interbedded medium to dark grey coloured calcareous mudstone and fine-grained sandstone beds, known as the IB unit (Interbedded unit), underlies the S1 unit. This unit typically does not host gold mineralisation.

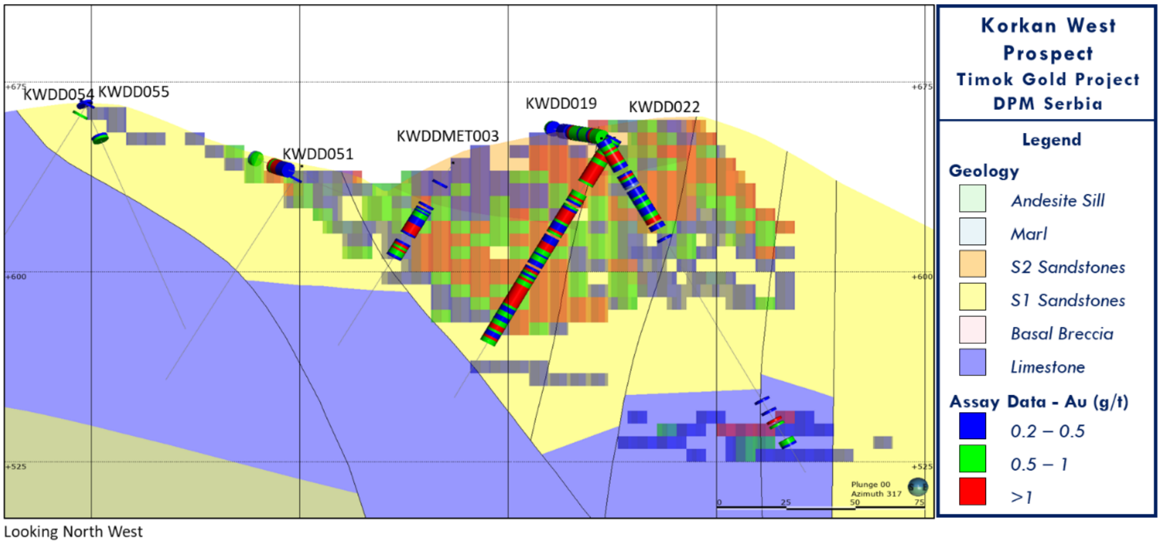
A thin sequence of conglomerate and breccia, with angular clasts of limestone within a clay matrix, occurs at the boundary sandstone-limestone and is known as BBX (Basal breccia unit). This unit usually carries no gold mineralisation, which is contrary to the BBX at Bigar Hill and Korkan. Limestone hosted gold mineralisation can be observed in fractured zones proximal to feeder structures, as well as at the Cretaceous-Jurassic limestone contact, in Jurassic limestone and karstified zones.

The orientation of structures in the Korkan West area are currently interpreted to be striking predominantly along a west-northwest to east-southeast orientation. These structures are located within a 300 m wide and 600 m long corridor and were most likely the feeder zones for hydrothermal fluids.

Structural modelling has demonstrated the presence of additional fault sets striking either northeast-southwest or east-west. It is assumed that in fault intersection zones, particularly in areas where west-northwest trending deep-seated structures intersect northeast-southwest trending structures, strata-bound mineralisation in the S1/S2 units can be observed. East-west trending structures are interpreted as the youngest and are not mineralised. Post-mineralisation faulting can locally displace mineralisation by up to 10 m in places.

Figure 7.12 is an example showing the distribution of Korkan West mineralisation relative to the main lithological units.

Figure 7.12 – Cross-Section of the Korkan West Deposit



(Source: DPM, 2020)

7.8.4 KRAKU PESTER DEPOSIT

The Kraku Pester Deposit is the third most advanced exploration target within the Project after the Bigar Hill and the Korkan Deposits. Kraku Pester shares similar characteristics with the Bigar Hill Deposit, 3.7 km to the north, and is located in an embayment at the north-western tip of the Potaj Čuka monzonite.

Gold at Kraku Pester is hosted in a variably disrupted stratigraphic sequence comprising, from base to top, shale metamorphosed to biotite ± magnetite phyllite, calcareous rocks; including marl and limestone metamorphosed to calc-silicate hornfels and marble, and tuffaceous rocks that locally may be calcite-rich and interbedded with coherent hornblende andesite. Metamorphism is due to emplacement of the Potaj Čuka monzonite unit that produced a thermal aureole up to 800 m in width. Direct correlation of the stratigraphic sequence at Kraku Pester with those recognised regionally at Bigar Hill and Korkan is uncertain.

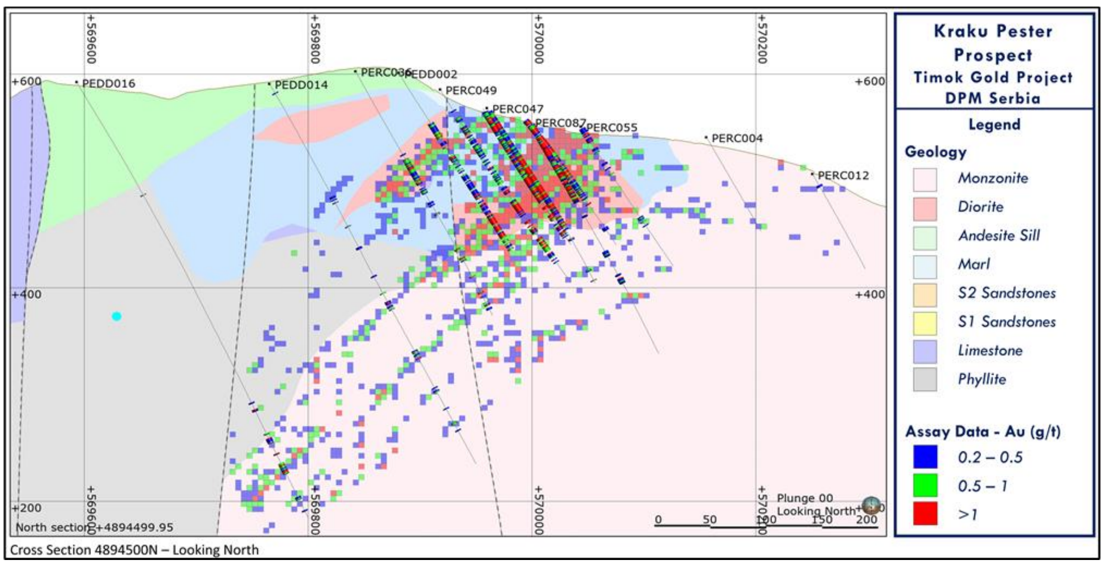
Mineralisation is less continuous at Korkan compared to Bigar Hill, due to greater structural complexity. As at Bigar Hill, it tends to follow the dips of the stratigraphy. It has a northeast-southwest extent of approximately 700 m and a north-south extent of approximately 600 m. Mineralisation can generally be traced from approximately 30 m below surface, and has a depth extent of 300 m.

Disruption of stratigraphic continuity at Kraku Pester indicates structural complication of the host sequence. Low-dipping structures of appreciable thickness are exposed, and fabric asymmetries associated with these faults indicate accommodation of down-dip extension. The presence of massive Jurassic limestone structurally above the heterogeneous Cretaceous sedimentary sequence suggests that the moderately-dipping structures originated as reverse faults that were reactivated. Steeply dipping fault damage zones have also been recognised, and cataclastic zones noted in the monzonite are locally host to auriferous pyrite.

Gold deposition is interpreted as being relatively late in the geological-structural evolution of Kraku Pester, post-dating the emplacement of the monzonite.

Unlike Bigar Hill, gold mineralisation at Kraku Pester is hosted in brittle fault rocks composed of pyritised fault breccia to cataclasite, with relatively higher gold concentrations being associated with finer-grained cataclasite. Fluid flow associated with gold mineralisation was controlled by a permeability fabric produced by brittle reactivation of a complicated geometric architecture in a north-westerly trending cross fault and the footwall intrusive contact with the monzonite. Figure 7.13 is an example showing the distribution of Kraku Pester mineralisation relative to the main lithological units.

Figure 7.13 – Cross-Section of the Kraku Pester Deposit



(Source: Avala, 2018)

7.9 Metallogeny and Paragenesis

Except for Korkan East and Bigar prospects, there is a common character to the sedimentary rock-hosted horizons, regardless of the prospect. The quantity of fine-grained pyrite increases from the margins toward the central and higher gold-content zone in all mineralised horizons. Except for the quartz-bearing zones in the andesite sill at Bigar Hill, no macroscopically visible silicate alteration minerals are evident. However, the fine-grained mineralisation, coupled with the common evidence of additional post-mineral brecciation, precludes easy identification of silicate alteration minerals.

Analysis of the geochemical characteristics of the mineralised horizons at Kraku Pester, using 1 m composites, suggests that clay minerals, presumably combinations of kaolinite, illite, and probably smectitic clays, form part of the hydrothermal alteration associated with pyrite deposition. Elevated gold is also associated with relatively iron-rich rocks, thus suggesting that the depositional mechanism was likely the sulphidation of iron present in the host.

The recognition of auriferous concentrations in karst-infill sedimentary rocks is consistent with this interpretation, as iron is a common residual element during carbonate dissolution. Decarbonisation of diagenetic and detrital carbonate is associated with gold zones.

Previous petrographic studies and metallurgical testwork on the Bigar Hill, Korkan and Kraku Pester sedimentary rock-hosted gold deposits suggest that gold is present in sulphide mineralisation as 0.5–40 µm grain size native gold, electrum or telluride crystals intergrowths with pyrite and other sulphides/sulfosalts, or as solid solution or submicroscopic scaled colloidal gold locked within As-rich pyrite bands (SGS, 2012a, 2012b, 2013; Pacevski, 2012a, 2012b, 2013; Magyar, 2018).

Most recently, Magyar (2018) showed on SEM images and EMPA maps the arsenic-rich pyrites (potentially associated to Au-mineralisation) have complex growth-zoning in various pathfinder elements, thus indicating several hydrothermal and supergene Au-mineralisation stages and implying variable Au-liberation metallurgical properties.

Two (2) textural types characterise the gold-bearing horizons; breccia, and replacement. The breccia-type consists of the basal breccia and karst horizons localised principally along the lower contact between the subjacent carbonate rocks and the overlying calcareous clastic rocks. Many of the breccia horizons are also the locus for post-mineral faulting, thus complicating the interpretation of the original mineralised rock texture.

The sedimentary rock-hosted deposit at Bigar Hill and Korkan are characterised by both textural types. The upper horizon along S1/S2 contact is a mixture of stratabound replacement type textures, and brecciated horizons that may, or may not, have formed post-mineralisation. Brecciated mineralised rocks are concentrated along the lower contact of the clastic rocks with the underlying carbonate.

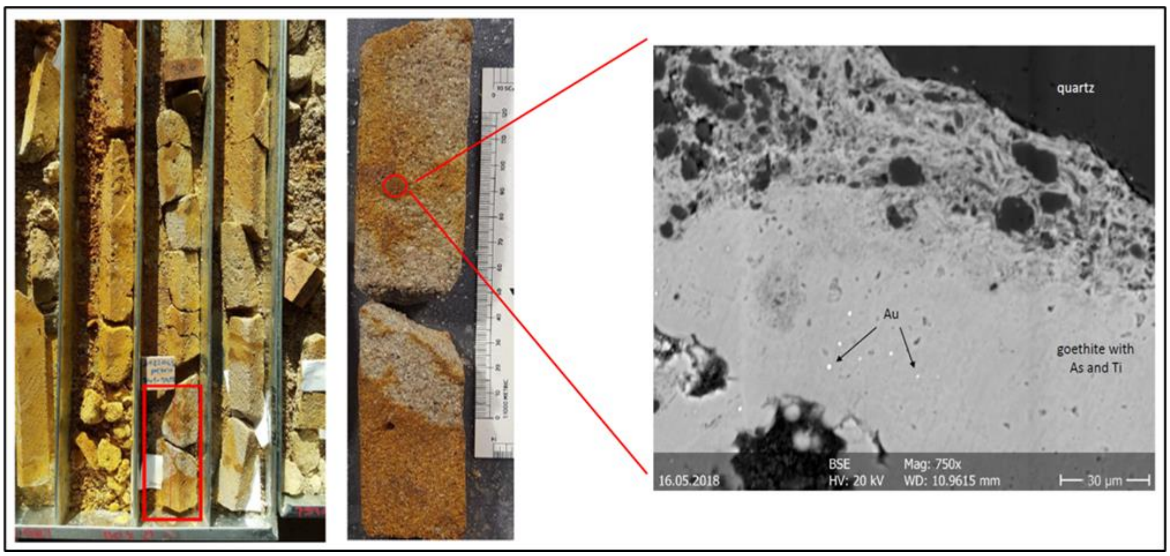
7.10 Weathering Profiles

Bigar Hill, Korkan and Korkan West prospects show extensive weathering and oxidation of iron bearing minerals. The Kraku-Pester prospect is weathered to a far lower extent compared to the other prospects. Weathering characteristics vary within each of the stratigraphic settings. A petrographic study by Magyar (2018) into the variability of weathering at the Project is summarised below.

Within higher stratigraphic elevations, late Cretaceous marls, andesites and magmatic derived clastics typically exhibit a shallow weathering profile, detectable up to 15 m below surface, which can be extended further downward when in proximity to faulting. These levels within the Project generally contain limited oxide and transitional mineralisation.

The S1 and S2 sandstones and conglomerates show pervasive weathering that can extend hundreds of metres below surface. Structural corridors, such as faults and lithologic contacts, allowed meteoric water to permeate downward. When these waters came into contact within mineralised zones, the oxidation of gold-bearing sulphides such as pyrite resulted in the formation of secondary iron oxides such as goethite. Corollary to this decomposition of sulphides, nanoscale gold particles were either liberated and left in situ or taken in solution and re-precipitated as native gold or electrum in geothites (Figure 7.14).

Figure 7.14 – SEM imaging of Mineralised Goethite taken from S1 Sandstones in Bigar Hill



(Source: Magyar, 2018) (Sample taken at 30.2 m depth from drill-hole BHDDMET001)

In certain locations, the meteoric water has pooled beneath impermeable caps, permitting extensive oxidation to occur. This is most notable within the S1 and S2 horizons that are capped by the relatively impermeable marls. In these locations, tabular bodies of sulphide mineralisation can be underlain by zones of oxidation which can extend for many hundreds of metres down-dip.

Within the lower levels of the Project stratigraphy, Early Cretaceous and Late Jurassic limestone may also display oxidation controlled along structural pathways. The breccia horizons that mark the unconformable contact between the S1 unit and Lower limestone almost always exhibit a zone of oxidation.

The Limestones of the early Cretaceous are typically karsted beneath the unconformity. The sub-tropical climate of the Cretaceous era, coupled with tectonic uplift, resulted in the formation of karstic cavities. These cavities appear to have been infilled with the overlying S1/S2, which often results in the re-deposition of mineralised pyrite and/or secondary iron minerals such as goethite. Fluctuations in groundwater levels resulted in gold grains being remobilised to microfractures on the edges of weathered pyrite grains.

8 DEPOSIT TYPES

The dominant mineral prospects in the clastic sedimentary rocks along the western margin of the TMC are relatively low-temperature auriferous deposits that share many characteristics with Carlin-type gold deposits, as outlined by Cline et al. (2005). The interpretation of the sediment-hosted gold prospects within the Project area as Carlin-type is based upon the following criteria (Knaak et al., 2016):

- Character of the sedimentary host.
- The metal association (gold, arsenic, mercury, thallium, sulphur and antimony).
- The fine-grained nature of the gold, high gold-to-silver ratio and alteration types including argillisation, decarbonisation, and locally, addition of quartz.

Sulphidation reactions appear to have controlled gold deposition, although the potential influence of a simple redox boundary along stratigraphic horizons and a decrease of gold solubility of mineralising fluids due to temperature decrease cannot be discounted.

The anomalous prospects associated with sedimentary rocks within the NW Timok area are Korkan East and Bigar, where significant gold is associated with carbonate replacement deposits composed of a variable assemblage of sphalerite-galena-arsenopyrite ± chalcopyrite concentrated along the brecciated contact between limestone and the overlying clastic sequence. Additionally, at NW Timok porphyry copper-gold and gold-only deposits are associated with hornblende-biotite-plagioclase-phyric diorite porphyry intrusions emplaced into the andesitic volcanic and volcaniclastic rocks.

The most significant and previously known deposits are the Valja Štrz and the Dumitru Potok porphyry copper-gold deposits. Exploration since 2000 has discovered the Kraku Ridji and Crna Reka porphyry copper-gold and Čoka Rakita porphyry gold prospects, largely as a result of soil and stream sediment geochemical survey by DPM.

Although spatially related, the timing and genesis of the sediment-hosted Au systems are uncertain as these deposits are always separated from porphyry gold-copper and polymetallic replacement deposits by faults.

The current understanding is that the various Late Cretaceous Au mineralisation types from NW Timok form a continuum and are part of larger magmatic-hydrothermal system(s) and represents various lithological traps (intrusive host, contacts, limestone replacement, clastic sediments replacement) or temperature segments (from porphyry toward epithermal) of the same system (Knaak et al., 2016).

The sediment-hosted gold belt lies west of a well-endowed metallogenic belt containing a range of magmatic-related deposits, including high sulphidation copper-gold and porphyry copper-gold. These deposits have formed the basis of significant mining activity at Bor for over 100 years.

Exploration by Avala and DPM has defined the previously unrecognised sediment-hosted gold prospects along the western margin of the TMC.

9 EXPLORATION

9.1 Introduction

Intensive exploration at the Project commenced in July 2010 following the acquisition of the Project by Avala Resources Ltd. A systematic exploration approach has been undertaken with the assembly of the following data sets over the whole Project area: topography, geological mapping, rock chip sampling, trenching, channeling and stream sediment geochemistry. Stream sediment sampling was previously completed over the entire Project area, at a nominal density of one sample per square kilometre.

Anomalous areas were followed up by rock chip sampling, mapping, and soil sampling, on a first-pass 400 m x 50 m grid in some of the anomalous areas and, very locally, subsequently with 100 m x 50 m grid sampling.

Trenching was used as a follow-up strategy to explore areas with anomalous soil geochemistry and to assist in defining key geological relationships due to the limited outcrop in the Project areas. There was a high success rate in intersecting sediment-hosted gold mineralisation by drilling near extensive and well mineralised trench intercepts.

Geophysical survey works within Timok belt started in 2006, by means of a heliborne VTEM survey, covering the original exploration license areas of the Project. Additionally, Induced Polarization surveys have been used since from the commencement of exploration works at Timok. Profiling arrays (Dipole-Dipole) with variable dipole spacing, depending on the target in question.

9.2 Geological Mapping

Outcrop exposure over the exploration licences is generally poor. However, in areas with outcrop, ground geological mapping together with rock sampling was undertaken. All existing surface outcrops have been mapped, including those created by earthworks activities associated with drill pad construction and cuttings for access roads. Geological maps were created using available observed lithology, alteration and structure data, followed by interpretation.

This has improved the definition of the geology in plan, with cross-checking during three-dimensional (3D) modelling of drill results for the Bigar Hill, Korkan, Korkan West and Kraku Pester Deposit areas.

9.3 Outcrop Sampling

Rock chip sampling has been conducted by Avala across the Project area. Rock samples, representing a wide range of rock types, were taken and analysed for gold by 50 g fire assay with atomic absorption finish. Pathfinder elements were analysed using multi-element inductively coupled plasma-mass spectrometry (ICP-MS) analysis covering 53 elements. All sample locations were surveyed by handheld GPS.

Data for each sample, including lithology, sample description, coordinates and assay results, are stored in an acQuire database. A total of 3,520 outcrop rock chip samples were collected by Avala over the Potaj Čuka Tisnica, Bigar Istok and Umka exploration licences.

9.4 Soil Geochemistry

Soil sampling has proven to be a very effective exploration method for localising potential sediment-hosted mineralisation. Gold, as well as low-temperature pathfinder elements such as arsenic, mercury, and thallium, have been found to be important elements in soil geochemistry surveys. An overview map of the Au in soils results is shown below.

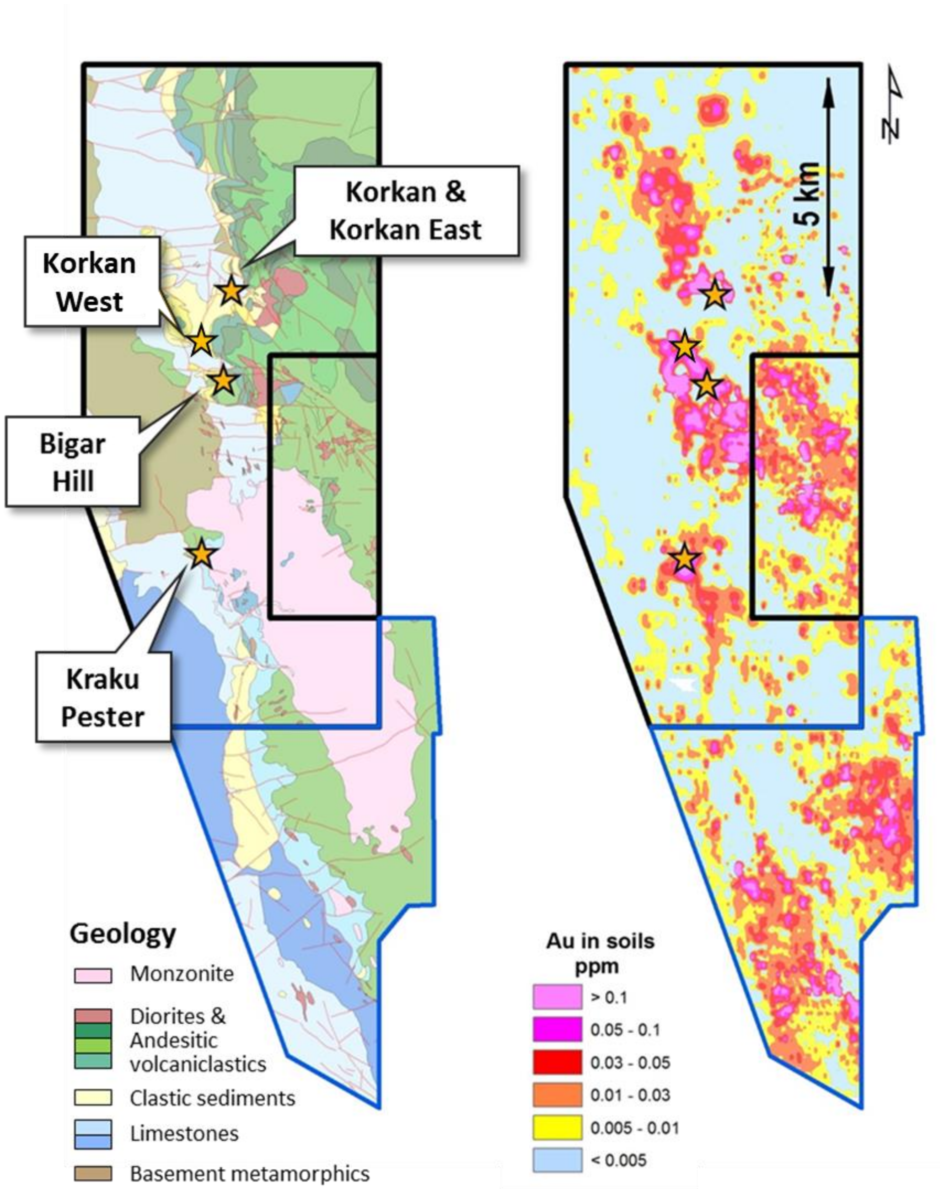
Avala collects soil samples from small pits, which are hand-dug by a sampling team. All samples are collected from the lower B-horizon. In the Potaj Čuka Tisnica licence area, most samples were collected at depths of 0.5 m to 1 m. Sampling was conducted in a grid pattern, beginning with a grid line spacing of 400 m and sample collection at 50 m intervals along each line.

Follow-up or detailed sample grids were configured at a line spacing of 100 m, with 50 m samples collected along each line. The sampling approach was based on orientation surveys completed by Avala in a similar environment from the Eastern Rhodope Mountains of Bulgaria. Soil field duplicates are collected at frequency of 1:20. Soil samples are collected by Avala field staff and transported to the Avala core storage facility in Bor on the same day they are sampled

As of May 2020, Avala has collected 20,010 soil samples over the Potaj Čuka Tisnica, Bigar Istok and Umka licences. Recent (from 2016 onwards) soil sampling activities involved the infilling previous sample lines. The results of soil sampling in general has been to confirm the size and shape of anomalies at a higher resolution.

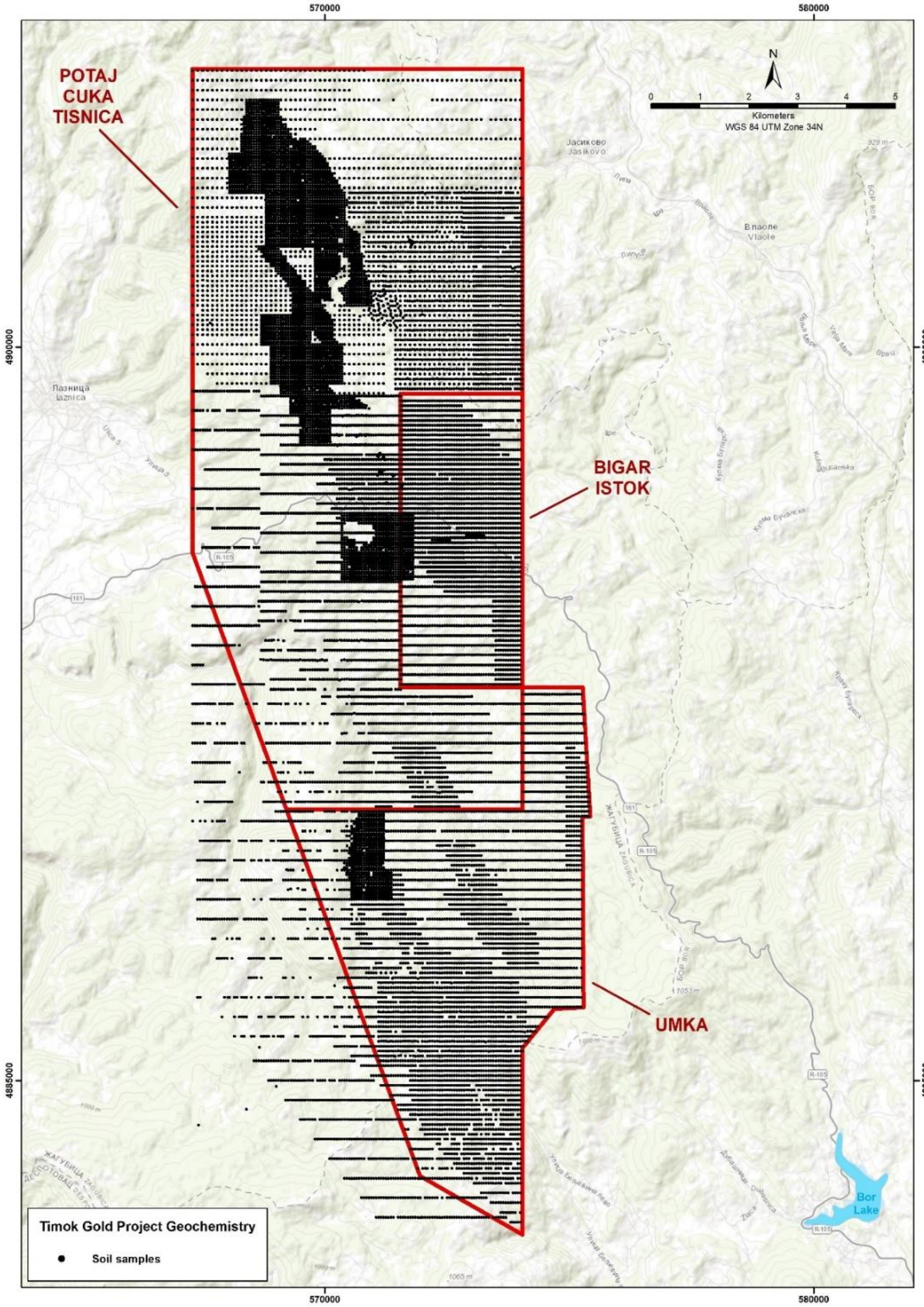
The results of all soil sampling to date has highlighted a near-continuous 20-kilometer-long, combined, gold-arsenic-antimony-mercury-thallium anomaly. The results of soil sampling are shown in Figure 9-1 whilst a plan map of all soil sample locations is shown in Figure 9.2.

Figure 9.1 – Geology Map with Key Prospects, Adjacent to Gridded Soil Sample Results for Au in Soils



(DPM, 2018)

Figure 9.2 – Location of Soil Sampling Lines



(Source: DPM, 2020)

9.5 Channeling and Trenching

Trenching was used as a follow-up strategy to explore areas with anomalous soil geochemistry and to assist in defining key geological relationships due to the limited outcrop in the Project areas. There was a high success rate in intersecting sediment-hosted gold mineralisation by drilling near extensive and well mineralised trench intercepts.

Channel samples are routinely taken on road cuttings or where outcrop exists. Channels are typically cut using a hammer and chisel, which allows sufficient penetration to excavate a channel approximately 100 mm high and 30 mm deep. Samples are caught into a chip tray which is cleaned at the end of every interval.

Trenches were completed under the supervision of exploration geologists. The dimensions of the trench are set out according to safety regulations, with a maximum depth of 1.5 m and a minimum width of 0.8 m. During excavation, the upper humus layer is separated from the underlying soil material so that it can be replaced and revegetated during rehabilitation.

Trenches were sampled as channels, with channel samples collected just above the trench floor at either 1 m or 2 m intervals. Except where extensive soil cover is encountered, trenches are sampled in their entirety. The samples were routinely weighed prior to final bagging to maintain an even sample size and to avoid sampling bias in harder rock types. An average channel sample weight of 3 kg/m was maintained. Field duplicate samples and certified standards were taken at a frequency of 1:20. All data collected in the field is routinely entered into geology and structural geology spreadsheets using Field Marshal software and later exported to an acQuire database.

Channel samples are collected by Avala field staff and transported to the Avala core storage facility in Bor on the same day they are sampled. The total amount of channels and trenches taken on the Timok licenses is shown in the table below.

Table 9.1 – Total Trench and Channel Sample Numbers

License	Trenches	Trench Distance (km)	Taken Trench Samples	Channels	Channel Distance (km)	Taken Channel Samples
PCT	367	46.7	27,388	154	8.7	6,671
BI	113	12.9	7,611	8	0.8	348
UM	87	18.5	9,638	17	0.7	371
TOTAL	**567**	**78.1**	**44,637**	**179**	**10.2**	**7,390**

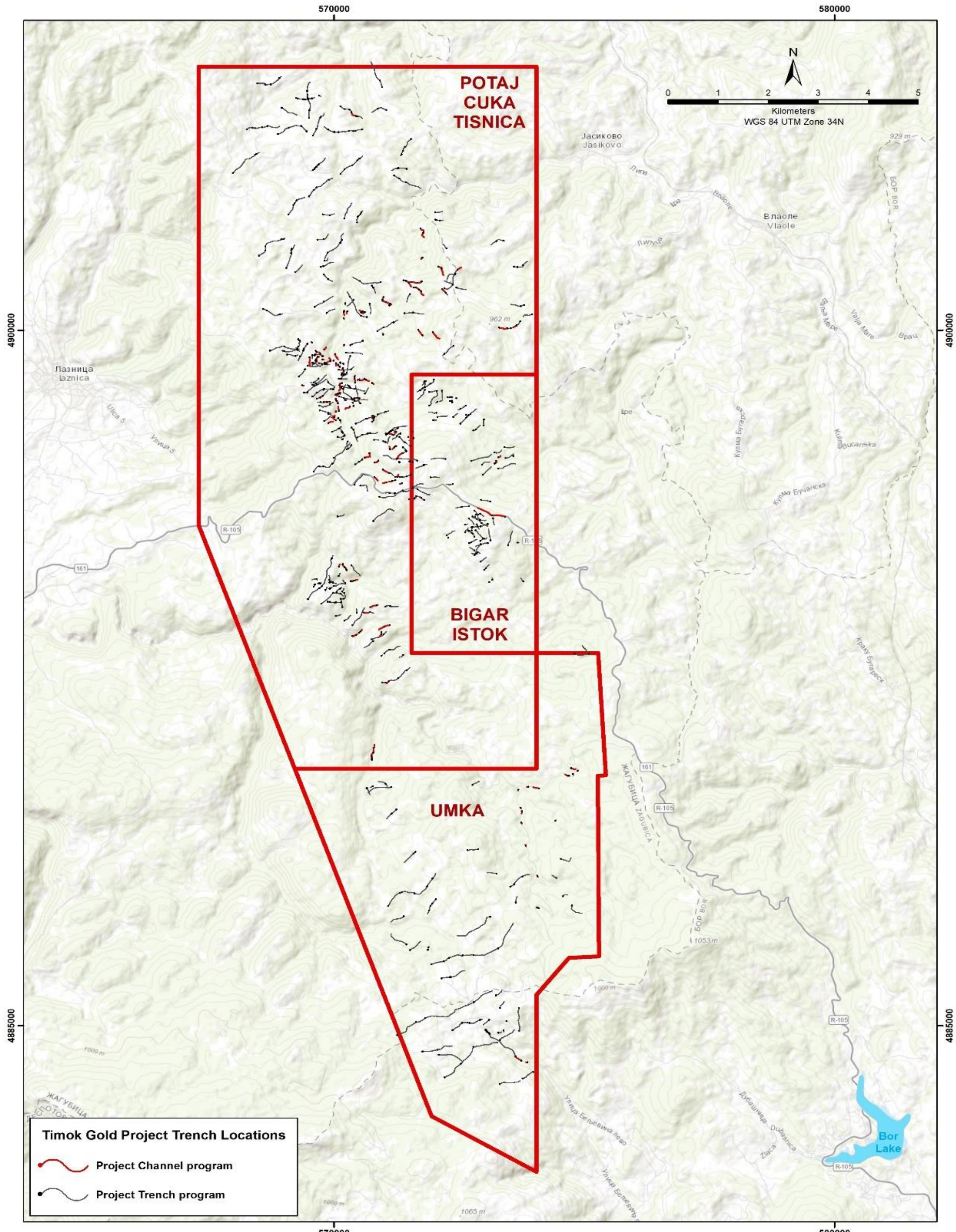
9.6 Topographic Surveys

All survey activities are conducted using by a licensed third-party surveyor. A base geodesic operational network within the Project has been established that covers the entire exploration tenement areas. This primary survey control network was implemented using AUSPOS, an online GPS processing service provided by Geoscience Australia.

High resolution topographic surveys are completed using GPS using a real time kinematic method which provides a centimetre level of precision. The system (Trimble R8 GNSS) uses two (2) receivers; one (1) is centred on control point with known coordinates whilst a second receiver is mobile and is used to determine survey points across the terrain. All coordinates are recorded using UTM coordinate system, specifically Zone 34 North in WGS 84 datum.

During 2020, a new topographic survey was completed using a DJI Phantom P4pro unmanned aerial vehicle (“UAV”). After processing in a photogrammetry software, an orthophoto mosaic was created at a resolution of 3.5 cm. Subsequently, a Digital Surface Model (“DSM”) at a resolution of 15 cm and a down-sampled digital terrain model (“DTM”) at a resolution of 1m was created. The survey encompassed an area that covers all the expected infrastructure requirements for the Project.

Figure 9.3 – Trenching and Channeling



(Source: DPM, 2020)

9.7 Geophysical Surveys

Geophysical survey works within Timok belt started in 2006, by means of a heliborne VTEM survey, covering the original exploration license areas of the Project. This survey method measures the electro-magnetic response, which is measured by a receiver loop towed by a helicopter along evenly spaced profiles at an approximate equal distance above ground. It also measures the total Magnetic Intensity of the Earth's magnetic field.

The survey was flown along traverses oriented at an 80° azimuth and a nominal line spacing of 100m with significant portion of infill at a 50 m line spacing. The objective of VTEM survey was to identify conductive targets in first couple of hundred meters below surface which could be caused by High Sulfidation and possible Porphyry styles of mineralisation. More recently, the areas not covered with by the heliborne VTEM have been flown with a Drone Magnetic survey.

This data was subject to further processing to calculate an "Airborne Inductively Induced Polarization" (AIIP) response over areas of sediment hosted gold mineralisation. Subsequently, Maxwell plate modelling was applied to nine selected conductive anomalies. The magnetic maps were used to help determine the lithological and structural architecture of the Mineral Resource areas.

The results of this study identified that un-altered intrusives (monzonite batholith), dykes and volcanic epiclastic units appear as the most magnetic units in the Project area. The Basal breccias, which are an erosional product that lies between the Jurassic Limestones and S1 sandstone can be magnetic, particularly if volcanic clasts are present.

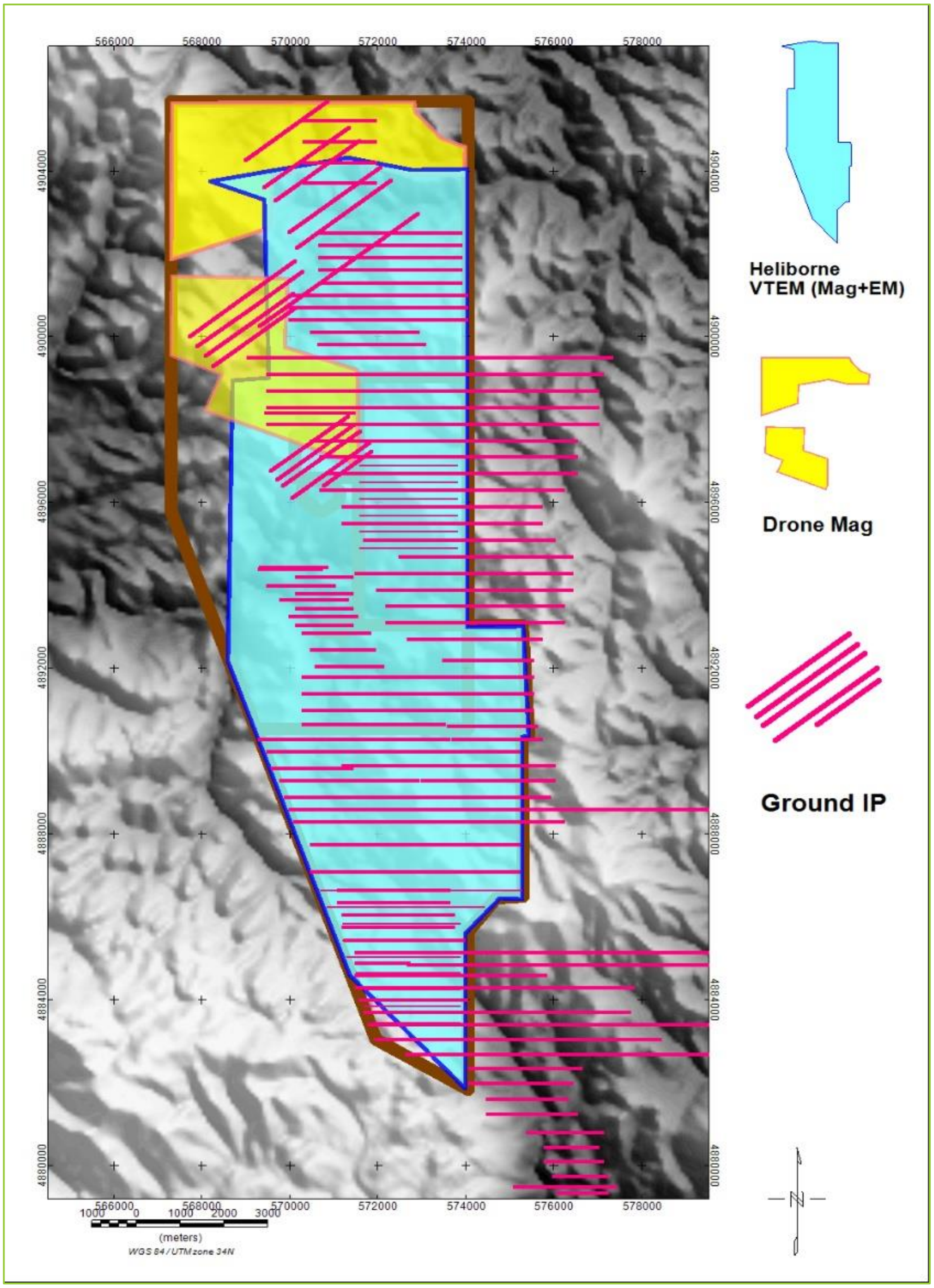
The porphyry systems to the east and south east of Bigar Hill appear with variable geometries with a typical pattern characterised by a moderate magnetic anomaly, surrounded by de-magnetised halo.

Induced polarisation surveys have been used since from the commencement of exploration works at Timok. Profiling arrays (Dipole-Dipole) with variable dipole spacing, depending on the target in question. A significant portion of the area was covered with a large dipole radius (200 m), in order to target blind porphyry systems. At a later stage, smaller (50 or 25 m) dipoles were employed with the aim to achieve better resolution at shallow levels, targeted around areas related to sediment hosted gold mineralization.

The results of such surveys showed that sulphide mineralisation appears as distinct anomalies of chargeability (Particularly Kraku Pester and Korkan East), however weathering of sulphides significantly lowers the chargeability response. Anomalies of elevated chargeability are interpreted as positive indicator for potential sediment hosted gold mineralisation, particularly if supported by nearby surface or downhole geochemistry.

A map of the coverage of all geophysical works completed on the Potoj-Čuka license area is shown Figure 9.4.

Figure 9.4 – Plan View of Geophysical Works Completed on the Potoj-Čuka License Area



(Source: DPM, 2020)

10 DRILLING

10.1 Introduction

Avala has employed a combination of diamond drilling and RC drilling across the Bigar Hill, Korkan, Korkan West and Kraku Pester exploration areas, and diamond drilling at Umka.

Drilling was carried out by Serbian contractors using Atlas Copco CS-14 and Atlas Copco Mustang 9/13/18, Alton HD, Coretech, YDX 1300G and Gemex MP 1200 rigs for diamond drilling, and GEMSA 500RC rigs for RC drilling. Examples of drilling activities are shown in Figure 10.1, dated January 2012. Drilling operations are summarised by area in Table 10.1.

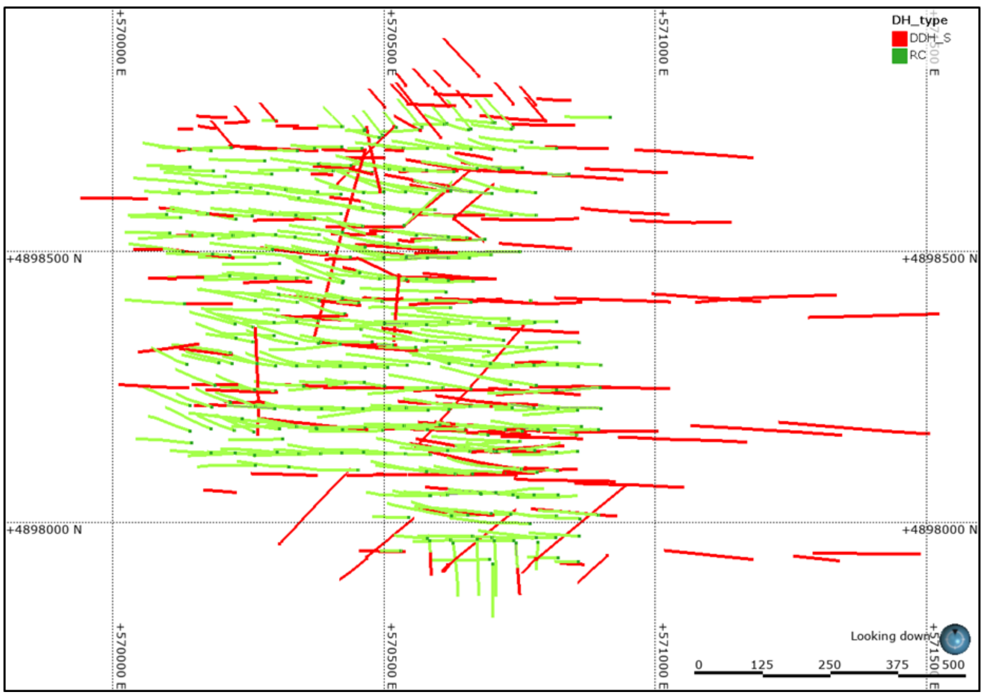
Figure 10.1 – Diamond (left) and RC Drilling (right) at Bigar Hill

(Source: AMEC, 2014)

Table 10.1 – Summary of Exploration Drilling, Channel Sampling and GC Drilling (as at May 29, 2020)

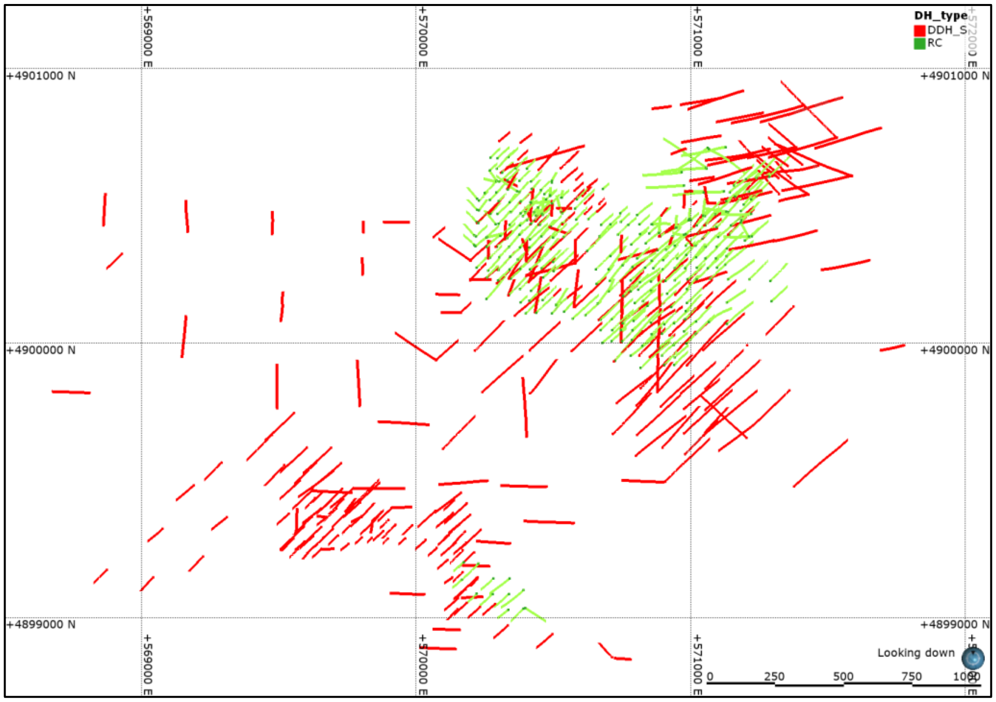
Deposit	Diamond Drilling		RC		RC Pre-Collar/ Diamond Tail		Geotechnical/ Hydrogeological		Metallurgical		Total	
	Drill Holes	m	Drill Holes	m	Drill Holes	m	Drill Holes	m	Drill Holes	m	Drill Holes	m
Bigar Hill	129	32,157	333	71,287	30	3,423	5	1,134	14	1,879	511	109,880
Korkan (including Korkan West)	281	65,614	295	49,804	9	1,237	7	850	22	2,099	614	119,604
Kraku Pester (inc. Kraku Pester South)	51	12,478	94	14,962	7	960	-	-	-	-	152	28,400
Total	**461**	**110,249**	**722**	**136,053**	**46**	**5,620**	**12**	**1,984**	**36**	**3,978**	**1,277**	**257,884**

Figure 10.2 – Drilling Completed at Bigar Hill



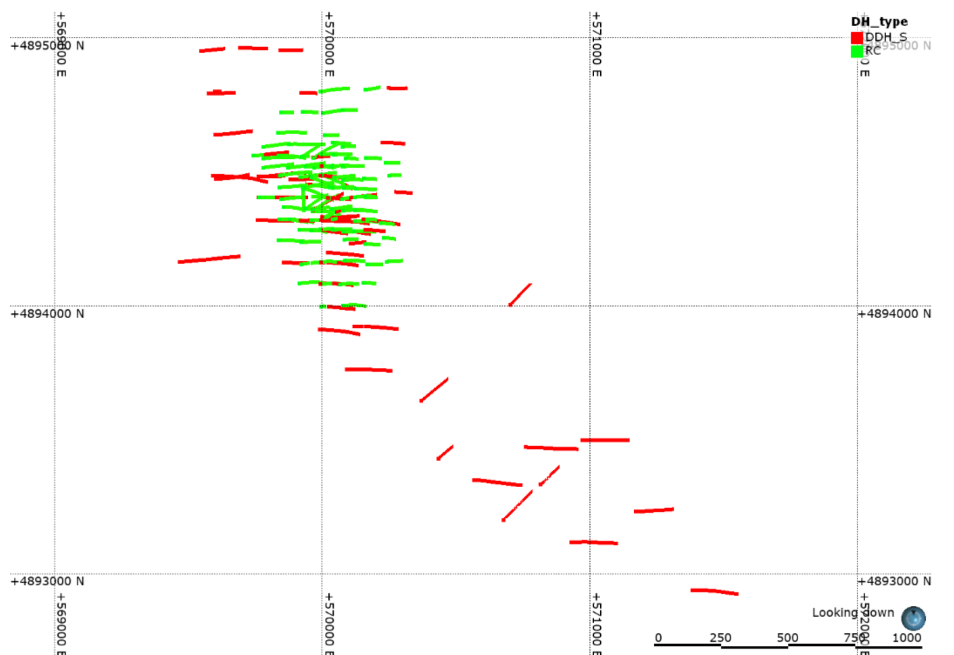
(Source: DPM, 2020) Diamond drillholes are shown in red whilst RC drillholes are shown in green

Figure 10.3 – Drilling completed at Korkan (including Korkan West) (right)



(Source: DPM, 2020). Diamond drillholes are shown in red whilst RC drillholes are shown in green

Figure 10.4 – Drilling Completed at Kraku Pester



(Source: DPM, 2020) Diamond drillholes are shown in red whilst RC drillholes are shown in green.

10.2 Methodology and Planning, Site Preparation, Setup, and Rehabilitation

After testing exploration targets with a detailed trenching program, prospects were evaluated with two (2) to three (3) drilling campaign stages, as follows:

- First stage: Wide-spaced diamond drill holes on a nominal grid spacing of 160 m x 160 m, with the objective of outlining the boundaries of the deposit.
- Second stage: Infill drilling, using RC drill holes and diamond holes, at nominal 80 m x 80 m spacing. Diamond drilling is used to support RC holes by twinning about 5% of RC holes.
- Third stage: Delineation drilling, using RC drill holes, at a nominal grid spacing of 40 m x 40 m.

Since 2016, DPM has opted to use solely diamond for all three stages of drilling.

The majority of drill holes at Bigar Hill are oriented at an azimuth of 270° and inclined approximately 60° to the west. In the case of Kraku Pester, drill holes are mostly inclined at 60° to the east to intersect gently west-dipping mineralisation. At Korkan and Korkan West, however, the orientation of the mineralisation is much more variable than at either Bigar Hill or Kraku Pester, and this is reflected in the greater range of drill hole orientations.

The central portion of the Korkan deposit is dominantly explored using holes inclined between 60° and 70° to the northeast. Toward the north, most of the drilling is inclined between 50° and 60° toward just south of west. A significant number of holes are aligned and inclined in a variety of angles outside of these two (2) patterns.

10.3 Collar and Downhole Surveying

Diamond drill hole downhole surveys are carried out by drilling contractors at 30 m intervals. Typically, a Devi Tool digital multi-shot camera is used for diamond holes. RC holes are surveyed at intervals of about 48 m using a Globaltech Pathfinder survey tool after the drill hole has been completed and drill rods have been extracted. On a few occasions, an Eastman single-shot camera was used on both diamond and RC holes. Survey results indicate downhole deviations from the drill hole collar azimuth and dip measurements are typically small.

Drillhole collar surveying is undertaken after every drill hole after been completed using a total station. The collar data is then imported into the Acquire database.

10.4 Drill Hole Logging, Data Acquisition and Sampling

This subsection describes the methods and protocols used for RC and diamond core drilling.

10.4.1 Reverse Circulation Drilling

Avala staff and drilling contractors followed a comprehensive set of drilling quality control and safety procedures for all RC drilling programs. All RC drilling was conducted under constant on-site supervision by the Rig Geologist.

RC drilling was undertaken using downhole hammers with face sampling drill bits. All drilling and sampling were confined to dry downhole conditions. Predominantly 141 mm and, to lesser extent, 147 mm and 139 mm drill bits were used with a shroud annulus of 2-3 mm to enhance sample recovery. All collars were lined with a 6 m casing of PVC pipe.

To ensure sampling was under dry conditions, and to enhance sample recovery, two (2) 1250 cfm compressors and an 870-psi booster was used at each drill site. Pressurised air blowbacks were routinely used after every metre of advance so that all the material within the drill stem was displaced into the sample bag prior to advancing to the next metre. At every rod change, compressed air blowdowns were used for cleaning the air system and for conditioning the hole before drilling resumed.

If drilling could not be continued under dry conditions, the RC drill hole was abandoned and re-entered using a diamond core drill to advance the hole. A dedicated compressed airline from the rig compressor was available at all times for cleaning of the cyclone and the sample splitter. All RC sample splits were collected daily by Avala staff from the drill rigs and transported to a secure core-shed facility in Bor where they were maintained under 24-hour security by Avala staff.

RC drilling samples have been routinely collected at 1 m intervals. Drill cuttings for each drilled metre are collected in a new plastic bag and marked with the drill hole number and interval sampled. Each bag of cuttings is weighed at the drill site using electronic scales. Cutting weights are recorded using handheld data loggers for input into the acQuire database and are monitored in real time during drilling for consistency using expected weights based on drill rods, bit sizes and shroud sizes being used and rock types. Changes in the weight of cuttings are also monitored by evaluating the statistical variations of cutting weights for each drill hole.

Routine sampling procedures require that the cyclone be cleaned at each rod change and after a wet sample. At every rod change, any material in the hole is cleared before the first new sample is collected. The riffle splitter is cleaned with compressed air and bottle brushes after each sample is split.

The average sample recovery is 91.2%, with an average 1 m sample bag weighing approximately 38 kg.

Upon arrival at the Avala core shed in Bor, all RC samples are measured for magnetic susceptibility, using a handheld meter. A small sample split is washed, and the chips kept in a chip tray for reference.

10.4.2 Diamond Drilling

Avala staff and drilling contractors followed a comprehensive set of drilling quality control and safety procedures for all diamond core drilling programs. Diamond drilling was carried out such that drill holes were always started using PQ core and then reduced to HQ triple tube (HQ3) once competent rock had been intersected. The diamond drill core size was maintained at HQ3 for as long as possible. NQ2 core diameter was used to extend RC holes that had not reached target depth because of drilling difficulties.

Core was transferred directly from the core barrel into appropriately labelled aluminium core boxes to ensure that core was correctly placed, and no core was lost. Wooden core blocks were placed between runs, recording the length of the run and any core loss. Forced breaks made by the drillers were marked on the core with a red cross on both sides of the breaks. At the drill site, core was washed clean of surface mud or other drilling fluids. All core boxes were labelled with the drill hole number, starting and ending depths for the core box, and box number.

Drill core orientation procedures were carried out at approximately 3 m intervals, and less in mineralised zones or areas of poor ground conditions. EzyMark, or occasionally spear-orientation equipment was used to mark the orientation of drill core.

Core boxes were collected by Avala staff at least once a day from the drilling rigs and transported to the Avala core storage facility in Bor on the same day. For transportation, core boxes lids were fitted by adhesive-coated fastening tape, and boxes were firmly secured with strapping in the transport vehicle.

Diamond drilling core recovery averages 98%. The majority of drill core was HQ3 size, followed by PQ3 and a small proportion of NQ. Specialised drilling muds and polymers were used throughout the program to maximise core recovery and, in areas of poor core recovery, drill runs were reduced to less than 0.5 m.

At the Avala core facility in Bor, all core is photographed dry and wet using a digital camera before logging commences. Core photos record the drill hole number, box number, starting and ending depths, and date. Photo sets are integrated with the Avala acQuire drill hole database.

Logging procedures are initiated with geotechnical logging, during which rock quality designation (RQD), joint strength and roughness, rock strength classification, and detailed core recovery are recorded. Core with drilling orientation marks is aligned with adjacent core intervals so that an orientation line can be drawn consistently over most of the drill core.

Geological structures are measured based on alpha, beta, and gamma angles relative to the orientation line. True orientations of features are determined using either a jig or by calculation. Geological logging is recorded using a digital logging form that provides an extensive geological description through a system of codes for lithology, alteration, veins, mineralisation, weathering, and vein descriptors.

After core logging has been completed, core is marked up for sampling at regular 1 m intervals corresponding to drilled depths. The 1 m sample intervals may be adjusted at key geological contacts or in sample intervals with significant core loss. These intervals must be less than 1.5 m and greater than 0.5 m long. Core is split along orientation lines using a diamond saw. Half the core is placed in a heavy cotton sample bag, together with a sample tag. Core samples weigh (on average) 3-4 kg. The remaining split core is replaced in the core box and retained at Avala's core shed facilities in Bor.

10.5 Deposit Drilling

Diamond drilling and RC drilling form the basis of modelling and tonnage-grade estimation mineralisation at each of the Bigar Hill, Korkan (including Korkan West), and Kraku Pester deposits.

Bigar Hill drilling consists of 129 diamond core drill holes for 32,157 m, and 333 RC drill holes for 71,287 m. A further 30 holes were drilled with RC pre-collars and diamond drill tails, for 3,423 m. Twenty-four (24) drill holes were twinned to confirm repeatability of drilling methods in identifying mineralisation.

Korkan (including Korkan West) drilling consists of 281 diamond core drill holes for 65,614 m and 295 RC drill holes for 49,804 m, as well as nine (9) drill holes for 1,237 m that were drilled with RC pre-collars and diamond tails. This included 18 drill holes which were twinned.

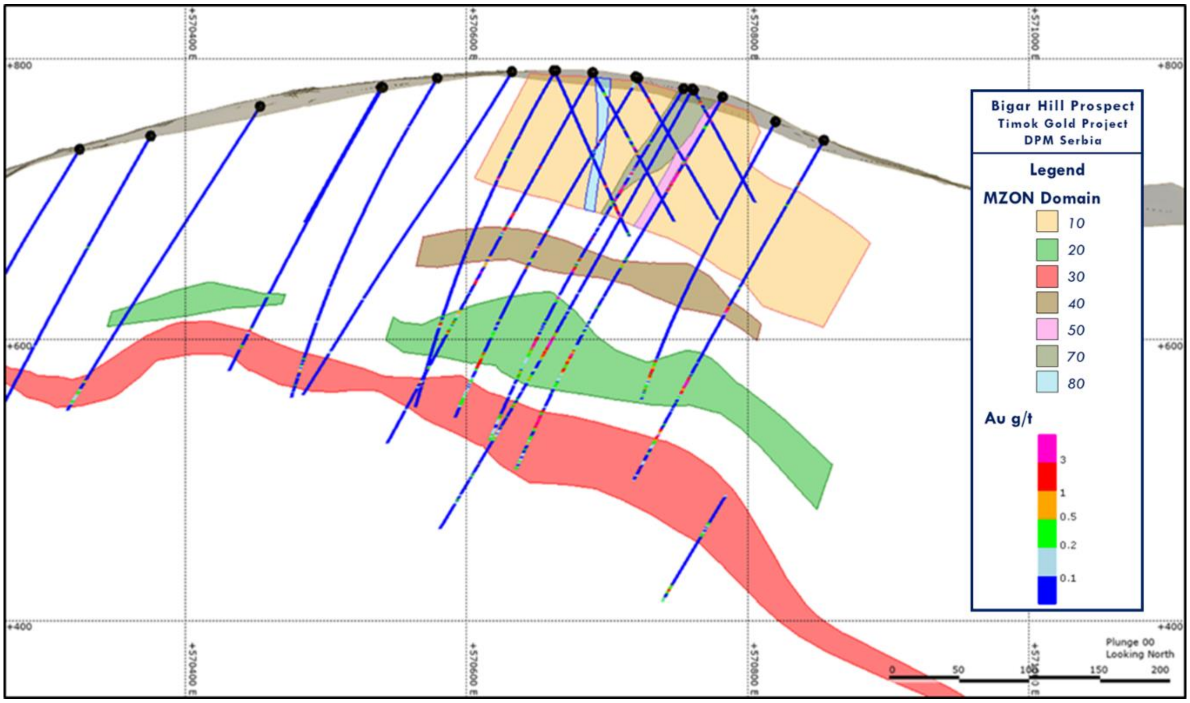
Drill hole collar locations for Bigar Hill, Korkan and Korkan West are shown in Figures 10.2 to 10.4.

At the Kraku Pester Deposit, drilling consists of 51 diamond core drill holes for 12,478 m and 94 RC drill holes for 14,962 m. A further seven (7) drill holes were drilled with RC pre-collars and diamond tails, for 960 m. Seven (7) drill holes were twinned to confirm repeatability of drilling methods in identifying mineralisation.

Drilling was generally done perpendicular to the mineral deposits to attempt to intersect the true thickness. The author/QP has identified no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results. More details regarding this assertion can be found in Section 12.

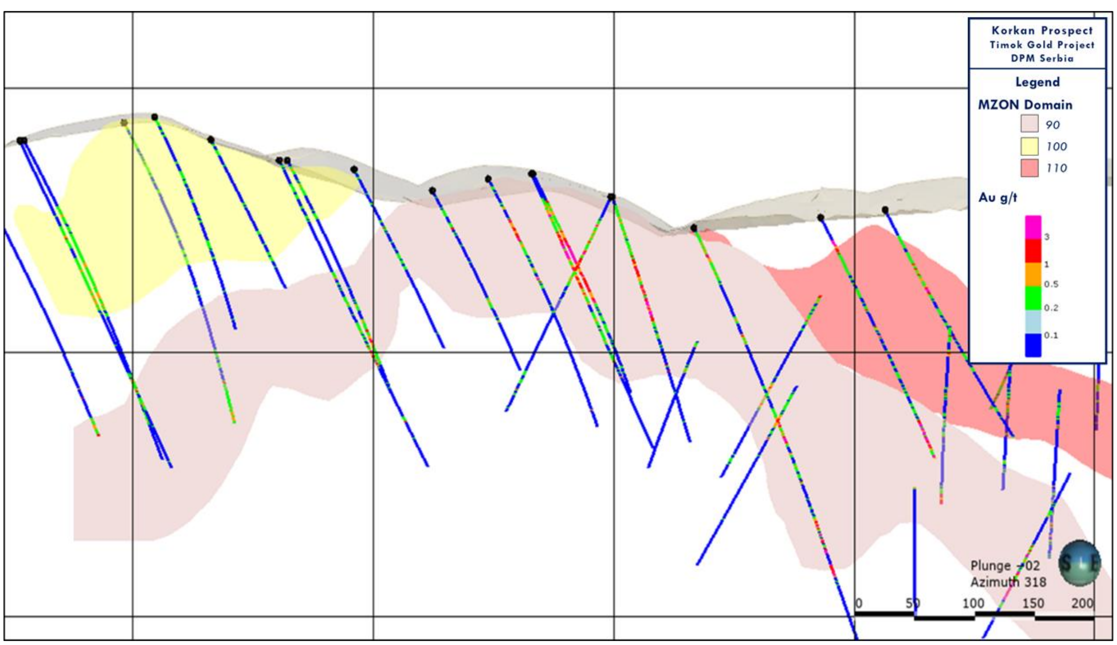
Representative examples of drill sections through the four (4) mineral deposits are presented in Figures 10.5 to 10.8.

Figure 10.5 – Bigar Hill - Cross Section showing Drilling and Interpreted Mineralisation Domains



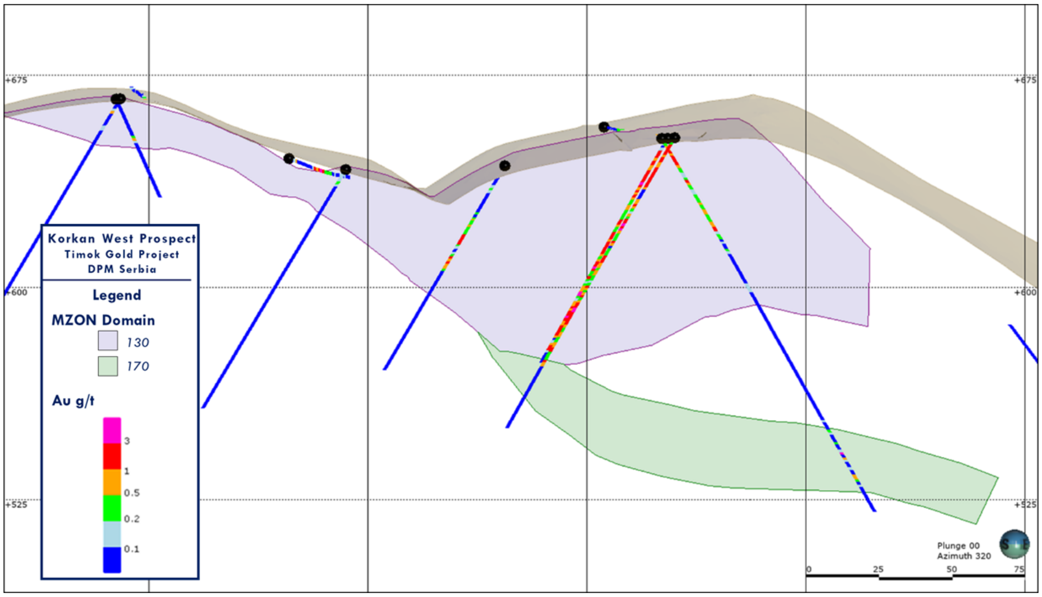
(Source: DPM, 2020)

Figure 10.6 – Korkan - Cross Section Showing Drilling and Interpreted Mineralisation Domains



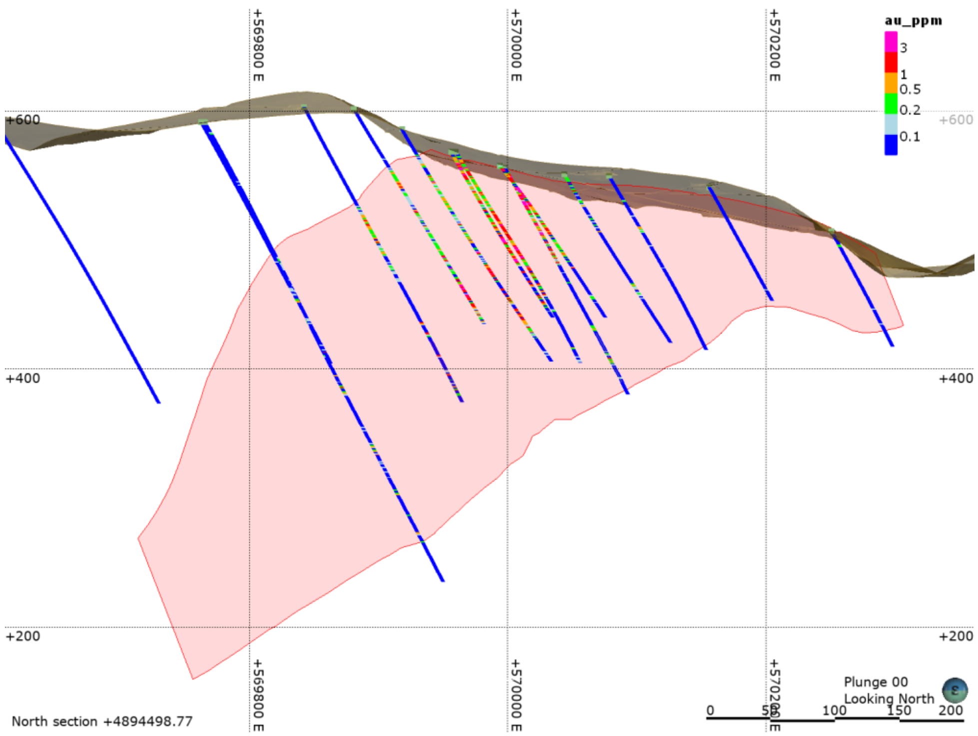
(Source: DPM, 2020)

Figure 10.7 – Korkan West - Cross Section showing Drilling and Interpreted Mineralisation Domains



(Source: DPM, 2020)

Figure 10.8 – Kraku Pester - Cross Section Showing Drilling and Interpreted Mineralisation Domains



(Source: DPM, 2020)

10.6 Metallurgical Drill Holes

Avala completed a series of six (6) metallurgical twin-holes during December 2017 in order to collect sulphide material for initial scoping test work carried out in 2018. An additional sixteen drill holes were drilled in 2018 to prepare additional metallurgical composites for testing in 2019. The metallurgical holes and sampling are detailed in Section 13.2.

10.7 Geotechnical Drilling

As part of the PFS, a geotechnical drilling program was designed by SRK Canada Inc with the primary objectives to characterize the geotechnical conditions and provide recommendations for the DPM pit design. A total of fourteen (14) geotechnical drill holes for a total of 2,204 m were drilled over the period of January 2019 to July 2020.

10.8 Hydrogeological Drilling

As part of the geotechnical program, a field program to characterise the hydrogeology conditions was conducted as part of the investigations by the University of Belgrade (Faculty of Mining and Geology, Department of Hydrogeology). The program included in-situ-hydraulic conductivity testing, and the installation of standpipe and vibrating wire piezometers (VWPs). Where possible, televiewer (optical and acoustic), caliper and full waveform sonic downhole surveys were conducted.

10.9 Exploration Drilling

In 2009, four (4) diamond drill-holes were drilled at the Project (Potaj Čuka Tisnica exploration licence). Two (2) drill holes were drilled on the Kraku Pester area (PEDD001 and PEDD002), and two (2) in the Bigar (Rapture Fault Zone) area (BIDD001 and BIDD002).

Avala then focused exploration drilling campaigns from 2010 to 2013 on the Potaj Čuka Tisnica licence to outline mineralisation on the Bigar Hill, Korkan, Kraku Pester, and Umka areas. The drilling that relates to Bigar Hill, Korkan, and Kraku Pester is covered in more detail in Section 10.

Apart from this resource drilling from 2010 to 2014, a number of exploration drill holes were completed at wide space grid in areas around mineralised prospects that include Bigar, Rapture zone, Korkan West, Pester South, Zumeri and Vizur areas.

After 2014, several exploration drill-holes were completed at wide space on areas around mineralised prospects which led to the discovery of Korkan West Deposit during winter 2016/2017. This led to further development of this area with independent drill programs during year 2017 (1st and 2nd Phase Korkan West Drilling Program).

Since 2017, exploration activities continued to focus on adding additional mineral resources within the Project area. Several trenches and channels intersected intervals with gold mineralization in areas of carbonate lithologies. Subsequently, the drilling of carbonate-rock hosted gold targets commenced during 2017, with 13 holes for 2,045 metres completed.

During 2018, 95 diamond drill holes were drilled, totaling 14,642 meters that were part of various exploration programs. The main focus was on the Umka license, testing for extensions to the sediment hosted gold mineralisation at the Božuluj prospect as well as testing anomalous soil and trench results elsewhere. In total 3420m of exploration drilling was completed. Results from the program located anomalous gold mineralisation in fault zones, karst infill material, on the contact zones with diorite dykes and in brecciated and marbleized zones in limestone. Although anomalous values were detected, none were significant to Mineral Resource inventories at Timok.

In 2019 a regional scale targeting model of the Timok property was prepared in order to aid with regional exploration. The model was built using drillhole and mapping data, 3D local geologic interpretations, geochemistry, IP data, magnetic and gravity data. Targets were generated using machine learning algorithms. From 12 proposed, eight (8) drill holes (total 1920.1 m) were drilled to check for new mineralization in proximity to current Mineral Resources and to test other poorly explored areas of the licenses. Results were disappointing with narrow, low-grade gold mineralisation between 0.1 ppm to 0.27ppm intercepted from 4 holes.

During year 2019, 148 diamond drill holes were drilled with a total of 21,338 m that were part of exploration and Project development drilling campaigns.

10.10 2020 Drilling

During 2020, Avala commenced RC infill drilling on a 20 m × 20 m drilling grid within shallow areas of oxide and transitional mineralisation the Mineral resource areas for Bigar Hill, Korkan and Korkan West. The purpose of the drilling was to improve confidence in Mineral Resource estimates in these areas and to allow re-classification to Measured Mineral Resource category. Furthermore, step out diamond drilling is also planned to close off open trends of mineralisation on the flanks of the prospects. In total 21,510 m of RC drilling and 1,770 m of diamond drilling was undertaken during the year.

Hydrogeological and geotechnical drilling was conducted at Bigar Hill, Korkan, and Korkan West areas in order to collect data to support the development of the Project studies. In total 4723 m of hydrogeological and geotechnical drilling has been undertaken during 2020.

Table 10.2 – 2020 Drilling Program

2020 Drilling Program	Start Date	Proposed		Completed	
		metres	holes	metres	holes
Infill RC drilling	20/08/2020	21,920	204	21,510	197
Infill Diamond Drilling	25/09/2020	1,930	19	1,770	18
Geotechnical and Dual Drilling	06/08/2020	3,170	17	2,260	12
Hydrogeology Drilling - Monitoring wells	16/08/2020	2,220	18	2,093	18
Hydrogeology Drilling - Pumping wells	05/09/2020	690	4	370	2

11 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 Field Sample Preparation Methods

Avala collected different types of samples including density, soil and trench samples and samples from RC and diamond core drilling. Sample preparations conducted by Avala prior to delivery to the laboratory are described below.

11.1.1 Dry Bulk Density Measurements

Bulk density measurements are restricted to diamond core only. Half-core samples of 20 cm to 30 cm are collected at an interval frequency of approximately every 3 m of all drilled core.

11.1.2 Soil, Channel and Trench Samples

Soil field duplicates are collected at frequency of 1:20. Blanks and low-level gold certified reference standards are inserted at the same frequency.

Trench and channel samples were routinely weighed prior to final bagging to maintain an even sample size and to avoid sampling bias in harder rock types. An average channel sample weight of 3 kg/m was maintained. Trench field duplicate rock samples and certified standards were taken at a frequency of 1:20. As of Q1 2017, blanks were similarly inserted at a frequency of 1:20 samples.

11.1.3 Reverse Circulation Hole Samples

RC drilling samples have been routinely collected at 1 m intervals. Drill cuttings for each drilled metre are collected in a new plastic bag and marked with the drill-hole number and interval sampled. Each bag of cuttings is weighed at the drill site using electronic scales. Cutting weights are recorded using handheld data loggers for input into the acQuire database and are monitored during drilling for consistency using expected weights based on drilling equipment and rock types. Changes in the weight of cuttings are also monitored by evaluating the statistical variations of cutting weights for each drill-hole.

Routine sampling procedures require that the cyclone is cleaned at each rod change and after a wet sample. At every rod change, any material in the hole is cleared before the first new sample is collected. The riffle splitter is cleaned with compressed air and bottle brushes after each sample is split.

Drill cuttings are split using a Jones three-tier riffle splitter to provide a sample that will be submitted to a laboratory for analysis. A typical split sample weighs approximately 4 kg to 5 kg. RC field duplicates, pulp duplicates and certified standard reference material are submitted at a frequency of 1 in 20 samples.

Blank samples consisting of un-mineralised quartz sand are submitted at a frequency of one for each drilling location at the start of the drill-hole sample sequence.

RC recovery was calculated by dividing actual sample weight (split + reject) by the theoretical sample weight. For RC drillholes from the Bigar Hill, Korkan and Korkan west Mineral Resource datasets, an average recovery of 91.2% was calculated.

Upon arrival at the Avala core shed in Bor, all RC samples are measured for magnetic susceptibility, using a handheld meter. A sample is washed, and the chips kept in a chip tray for reference.

11.1.4 Diamond Drill Core Hole Samples

After core geological logging has been completed, core is marked up for sampling at regular 1 m intervals corresponding to drilled depths. The 1 m sample intervals may be adjusted at key geological contacts or in sample intervals with significant core loss. These intervals must be less than 1.5 m and greater than 0.5 m long. Core is split along orientation lines using a diamond saw. Half the core is placed in a heavy cotton sample bag, together with a sample tag. Core samples weigh (on average) 3 kg to 4 kg. The remaining split core is replaced in the core box and retained at Avala's core-shed facilities in Bor.

Core "field duplicates" are prepared by producing split samples after the jaw crushing stage of sample preparation, with each split being assigned a unique sample number. Pulp duplicates and certified standard reference material are submitted into the assay sequence at a frequency of 1 in 20 samples. Blank samples of un-mineralised quartz sand were submitted at one in every batch submitted to the analytical laboratory at the beginning of the batch sample sequence. The procedure was updated in 2017, wherein coarse blanks (rocks) are now used instead of sand and blanks are now inserted at a 1 in 20 frequency.

11.2 Security

Samples collected from field operations are transported to the geology core shed based in Bor, where the samples are geologically logged and are prepared for chemical analysis. The sampling procedures are appropriate and adequate security and supervision exists on the site to minimise any risk of contamination or inappropriate mixing of samples. A pulp library is maintained of all samples prepared by SGS Bor, which are stored in a locked warehouse onsite.

Core shed, laboratory and pulp library facilities are located within a gated compound located in Bor, that requires a secure key card to access. The facility has an alarm system and CCTV cameras distributed across the site.

11.3 Laboratory Sample Preparation and Analyses

11.3.1 Laboratory Sample Preparation

Sample preparation for all samples (soil, trench, channel sample, RC and diamond core) is undertaken at the SGS Bor (SGS) sample preparation facility in Bor. This facility is owned by Avala, but independently managed by SGS, such that the chain of custody is transferred from Avala to SGS at the laboratory door. The SGS facility is located adjacent to Avala's core shed facilities in Bor.

All submissions to the sample preparation facility are accompanied by sample submission forms with instructions for preparation methods, insertion-of-standards protocols, and analytical process codes. Once the samples are delivered to the SGS sample preparation facility, chain of custody records are maintained until reject sample pulps are returned to Avala's jurisdiction.

All samples submitted to the facility are initially dried at 105°C for a minimum of 12 hours.

Core, trench, and rock samples are then crushed to 4 mm, using jaw crushers. Crushing is checked by confirming that 85% of the crushed material can pass through a 4 mm sieve. Core "field duplicates" are produced by splitting crushed samples on a 1:20 basis at the jaw crusher output stage. Each "field duplicate" subsample is assigned its own identification number for the remainder of the assay procedure. All crushed sample material is then pulverised using LM5 pulverising mills (of which there is currently a bank of eight).

RC drilling samples are pulverised in their entirety using the LM5 pulverising mills.

A standard part of the SGS laboratory operating procedures is for 1:10 pulps to be wet sieved using a motorised sieve bank in order to confirm that the sample passes a P90 of 75 µm. If a sample fails the test, the previous ten (10) samples are re-pulverised.

Pulverised material, from all types of sample, is split into 250 g and 600 g pulps, where the former is used for assay determination, and the latter is stored as part of the reference pulp library which is securely stored within the Avala sample office facility. An additional 250 g pulp duplicate is split from the pulverised material at a frequency of 1:13.

11.3.2 Laboratory Analyses

Routine analysis of all samples is currently performed at the SGS analytical laboratory in Bor, or previously at the SGS analytical laboratory in Chelopech, Bulgaria. The labs are independent of Avala, DPM and CSA Global. All laboratory methods, procedures, and QA/QC protocols are consistent with standards adopted by SGS worldwide standards.

Gold analysis methodology is conventional 50 g fire assay, with an atomic absorption finish. Silver and base metal analyses (copper, molybdenum, arsenic, bismuth, lead, antimony, and zinc) are performed using a 0.3 g charge, aqua regia digestion, and atomic absorption analysis. Sulphur samples are analysed by combustion with an infrared finish.

The Bor and Chelopech laboratories are not ISO 9002 or ISO 17025 accredited for the above analytical procedures. However, the procedures routinely used at both the SGS laboratories include the following established and standard specifications at all SGS laboratories worldwide:

- Cross-referencing of sample identifiers.
- Use of compressed air gun and vacuum gun, along with routine barren quartz "washes", for cleaning of crushing and pulverising equipment.
- Routine assaying of quartz washes.
- Assaying of SGS-submitted certified standards at a rate of two per batch of 40 original samples.
- A minimum of 10% of submitted samples are subject to repeat analysis.

Second splits generated by the SGS CCLAS system are produced at a rate of 1:13 and represent a second subsample taken from the LM5 pulverised pulp.

All soil samples were assayed by ALS Chemex in Perth and SGS Vancouver, using methods Au-TL43 (gold by aqua regia digestion with ICP-MS and ME-MS41 (combined ICP-MS and ICP-AES dependent on concentration). Elements assayed for are silver, aluminium, arsenic, boron, barium, beryllium, bismuth, calcium, cadmium, cerium, cobalt, chromium, caesium, copper, iron, gallium, germanium, hafnium, mercury, indium, potassium, lanthanum, lithium, magnesium, manganese, molybdenum, sodium, niobium, nickel, phosphorous, lead, rubidium, rhenium, sulphur, antimony, scandium, selenium, tin, strontium, tantalum, terbium, thorium, titanium, thallium, uranium, vanadium, tungsten, yttrium, zinc and zirconium. The ALS Chemex laboratory in Perth is certified to ISO 9002, but not ISO 17025 accredited for this technique.

An ICP-MS machine has recently been installed and brought online at the SGS Bor laboratory where relevant samples are analysed for 49 elements.

Umpire pulp aliquots, sent to two external accredited labs, are assayed for gold by 50 g fire assay, with an atomic absorption finish. Silver is analysed using a 0.3 g charge, aqua regia digestion, and atomic absorption analysis. Sulphur is analysed by combustion furnace.

Pulp aliquots for dispatch to other laboratories (abroad) are packed in boxes which are plastic-wrapped or taped-shut for transport in sealed containers. The sealed sample boxes, accompanied by chain-of-custody documents, are transported door-to-door by an international courier delivery company.

Reject pulps, returned to Avala jurisdiction, are stored in an enclosed "pulp library", with access through secure key card only.

11.3.3 LeachWell Agitated Cyanide Leach Assay Program

An agitated cyanide-gold leach assay program was conducted in 2013 on 3,930 five-metre composite pulp samples taken from diamond and RC drill-holes from the Bigar Hill (1,810), Korkan (1,201) and Kraku Pester (919) prospects. Analysis was conducted by SGS Mineral Services, Perth Airport Laboratory, which holds ISO/IEC 17025 accreditation.

The samples were ground to 85% passing 75 µm and subsequently 200 g subsamples were analysed by conventional fire assay (“Au-FA505-ppm”) and then processed for agitated cyanide leach using tap water at ambient temperature for four hours using the SGS LeachWELL method.

The SGS LeachWELL is an accelerated partial digest technique designed to determine the cyanide extractable gold content of samples. The settled solutions were analysed for gold by AAS (Au-LWL69J_ppm). The post-leach residues were washed, dried, re-ground and analysed by 25 g fire assay with atomic absorption spectroscopy (AAS) finish to determine the undissolved gold contents; each measurement was replicated (“Au-residue-avg-ppm”, Au_FAA303_ppm and Au_FAA303R_ppm). The residue and solution assays were used to calculate the total gold content of the samples and a ratio of gold leached by the cyanide solutions ("CNAu/Au"). Cyanide soluble gold CRM was inserted on a 1:20 frequency.

11.3.4 Dry Bulk Density Measurements

Half core billets are submitted to the SGS sample preparation facility at Bor for determination using a wax-sealed core water immersion method. After measurements have been completed, the core was returned to Avala and replaced in the core boxes.

In 2011, an external check of 188 bulk density measurements was performed by sending samples for retesting to the Evrotest-Control laboratory in Sofia, Bulgaria, which is certified for BS EN ISO 9001:2008. This data was not available for review, although site management confirm this test work was undertaken and no significant bias was observed.

11.4 Avala Assay QA/QC Procedures

Avala performs routine checks on every laboratory submission upon import to the drill hole database, using acQuire QA/QC tools. These checks are initially undertaken on receipt of the assay results, in order to determine if the submission has passed the Avala control test. If the submission fails, it is re-assayed. On a monthly basis, the QA/QC data in general is assessed using custom acQuire tools to identify any QC issues or trends, so they can be acted on in a timely manner. Failures in QC samples can be immediately discussed with the analytical laboratory and, if needed, batches can be rapidly resubmitted.

Avala routinely inserts internationally certified standards, covering a wide grade range, along with blanks, into the sample submission stream. The samples are in standard pulp packets, but the recommended values of the samples are unknown to the SGS laboratories. The standards and blanks are inserted at a rate of 1:20 samples. In addition, Avala has produced, as part of the sample sequence, RC field duplicates, which are also unknown to the SGS laboratory. Coarse crush duplicates have been produced from the diamond core samples by SGS and included for analysis.

Avala considers Certified Reference Material (CRM) that assays 10% outside of the expected value for gold, or 15% outside of the base metal expected values to be a failure and will require the laboratory to re-assay 10 samples prior to, and 10 samples following the failed QC assay. This instruction includes the submission of standard reference material control samples. If more than two standards have failed in a submission, the entire submission will be required to be re-assayed. If a failed standard is amid a sequence of results below the detection limit, it is up to the geologist assessing the data to determine if re-assay is required.

Between 2008 and 31st December 2013, all diamond and RC drill samples were submitted on a 1:20 frequency for Umpire analysis. Since July 2016, DPM has changed the umpire selection procedure in that preference is given to individual samples that displayed greater than 0.1 g/t gold in order to provide an accurate test of laboratory performance and avoid analysing a large number of near-detection level samples. All prospects drilled in a given year were represented in the umpire sample selection.

11.5 Conclusions and Recommendations

The QP has reviewed active RC drilling, diamond drilling and channel sampling during site visits from 2012 onward. The sample preparation procedures employed onsite by Avala are consistent with DPM exploration protocols and are adequate given the type of mineralisation at the Project. The QP concludes that the sample preparation, security, and analytical procedures are robust and follow industry best practice.

The QA/QC procedures are comprehensive and are suitable to monitor assay contamination, accuracy and precision. The author of this report recommends the following:

- Although the failure limits used for the standards are adequate, DPM should move to using standard deviations to obtain acceptable limits. Any standard result that varies from the expected value by more than three (3) standard deviations, or any two (2) consecutive standards differing more than two standard deviations would constitute a failure.
- Although Ag is not a significant contributor to the project, for completeness, DPM should include Ag CRM with expected grades ≥ 10 x LDL as well as analyse blanks and field duplicates for Ag.
- DPM should send a suite of bulk density samples for umpire analysis since the previous testwork was unavailable for review. A minimum of 30 umpires by rock type for each prospect would be adequate considering the low levels of variability within the dataset.

12 DATA VERIFICATION

12.1 Data Verification Completed by DRA (2020 Mineral Resource Update)

12.1.1 SOURCE DATA VERIFICATION

Source drill holes data were received from DPM in Comma-Separated Values (.csv) format files that constitute exports from the AcQuire database used by DPM for exploration data storage and management. Files received were related to Assays, Bulk Density, Collars, Geology, Geotech, MagSusc, Structure, and Survey.

Prior to importing the data into the modelling software package, DRA did verify for inconsistencies such as sampling overlaps, unexplained sampling gaps, unusual values, etc. Some errors were found in the assay file and were reported to DPM and most of them were due to data export from SQL. They were corrected and the final assay file clean of any error was received on May 29, 2020.

In the course of data verification, DRA also randomly selected 99 holes from the drill hole database and requested DPM to supply the original samples certificates for all samples related to these selected holes. Selected holes refer to 24,682 assays, which represent about 8% of the entire Timok drill hole database. Holes have been selected keeping in mind the objectives to have assaying results covering the whole range of gold values variation, starting from low grades and up to high grades intervals. Original lab certificates of the selected samples were transmitted to DRA as PDF files with signature of the laboratory manager on each certificate. DRA has cross compared gold analytical results as plotted on the lab certificates and as digitalised in DPM digital database. There were no major discrepancies were found.

Based on the information gathered, DRA can comment that the source data appears to have been accurately captured in the drill hole database received. Therefore, DRA was not limited in its access to any of the supporting data used for the MRE.

12.1.2 DATABASE VALIDATION

A number of validation tests on the sample data in the database were undertaken upon import into Vulcan mine planning software and then subsqeuently in Leapfrog Geo modelling software. Both software packages have a standard validation process to ensure database integrity. Checks were undertaken to ensure the data meets the following validation rules:

- Collar Table: Unique collar cordinates and appropraite collar distance from topography model.
- Survey Table: Duplicate entries, survey intervals past maximum depth in collar table, overlapping intervals.
- Assay Table: Missing samples, overlapping intervals, invalid entries, excessive sample weights.

- Geotechnical Table: Core recoveries and RQDs greater less than % or greater than % (Recovery), or % (RQD), overlapping intervals.

Minor issues effecting sample intervals and sample weights were observed and reported to the DPM database team. No further errors were detected and the dataset is considered valid for use for MRE.

12.1.3 Core Inspection

Ross Overall, Corporate Mineral Resource Manager, of DPM, inspected core whilst onsite in December 2019 in relation to the metallurgical drilling programs. Drillholes KODDMET010, KODDMET011 and KODDMET012 were reviewed with site personnel team, along with assay results and logging data. Discussions were held with site personnel in regards to the observed mineralisation styles, stratigraphic relationships and overall suitability of the core for metallurgical testwork.

12.2 QA/QC Review

A review of Quality Control (QC) data for the Project (all prospects included in the Mineral Resource Estimate) was completed for Au, Ag and S samples for both phases of project ownership.

- Phase 1 - Avala (2008 to 31 December 2013);
- Phase 2 - DPM (1 July 2016 to date).

Results of cross contamination (blanks), assay accuracy (CRMs) and assay precision (duplicates) were reviewed separately for each of the analytical laboratories used. In addition, the QC results from the cyanide leach Au assaying in 2014 were also reviewed.

Table 12.1 lists the ratio of blanks, CRM and field duplicates to the primary samples for the Avala and DPM phases of exploration. A ratio of 3% to 5% is the usual industry accepted standard practice for each type of QC sample in this type of gold environment. Avala ratios are appropriate except for blank ratios which are lower than is commonly used and there were insufficient S CRM included. DPM had no QC samples for Ag, but ratios are appropriate for Au and S samples.

Table 12.1 – Ratios of Quality Control SAMPLES to Primary Samples

Phase	Element	Assays	Blank	Blank %	CRM	CRM %	Field Dup	Field Dup %	Umpires	Umpire %
AVALA	Au	237,561	1,294	1	12,247	5	7,083	3	12,373	5
AVALA	Ag	80,547	662	1	2,679	3	1,520	2	3,219	4
AVALA	S	228,684	1,144	1	422	0	6,536	3	6,439	3
DPM	Au	38,175	2,446	6	2,554	7	2,472	6	297	1
DPM	Ag	38,181	0	0	0	0	0	0	281	1
DPM	S	38,118	1,769	5	1,609	4	1,788	5	294	1

Note that although the overall percentage of DPM umpire samples is lower than the Avala percentage, DPM have a targeted approach whereby they only selected mineralised samples (≥ 0.1 g/t Au) for analysis. This results in a lower number (and percentage) of umpires overall but ensures that the samples analysed are relevant with mean grades exceeding ten times the lower detection limit (Au LDL 0.01 g/t). Samples with mean grades within ten times the LDL are usually excluded from precision calculations as these results are inherently imprecise. Therefore, the percentage of DPM umpire samples for Au and S is deemed appropriate.

12.2.1 CROSS CONTAMINATION

A coarse or preparation blank undergoes sample preparation with the primary samples and is used to check for cross contamination in the preparation process. Pulp blanks are used to monitor contamination in the analytical process. Avala used two blanks: a non-certified coarse blank (BLANK_BOR) and a certified Au pulp blank (GREY BLANK). DPM continued to use the BLANK_BOR as their preparation blank. Warning and failure limits of three and ten times the lower detection limit (LDL) respectively were used for each analytical method.

12.2.1.1 Avala Drilling

The results for the Avala blank analysis are listed in Tables 12.2 to 12.4 for Au, Ag and S respectively. No failures for BLANK_BOR were noted, although there were some values greater than 3 x LDL. Failures were noted for the GREY BLANK for S, but this blank is not certified for S and therefore, these failures are not material.

Table 12.2 – Avala Au Blank Data

Phase	Laboratory	Blank Type	No. of Samples	Mean Au (g/t)	No of Failures (> 10 x LDL)	No of Warnings (> 3 x LDL)
Avala	SGS_BO	BLANK_BOR	784	0.007	0	5
Avala	SGS_BO	GREY BLANK	396	0.005	0	0
Avala	SGS_CH	BLANK_BOR	114	0.007	0	2

Table 12.3 – Avala Ag Blank Data

Phase	Laboratory	Blank Type	No. of Samples	Mean Ag (g/t)	No of Failures (> 10 x LDL)	No of Warnings (> 3 x LDL)
Avala	SGS_BO	BLANK_BOR	250	0.53	0	0
Avala	SGS_BO	GREY BLANK	330	0.01	0	0
Avala	SGS_CH	BLANK_BOR	42	0.59	0	1

Table 12.4 – Avala S Blank Data

Phase	Laboratory	Blank Type	No. of Samples	Mean S (%)	No of Failures (> 10 x LDL)	No of Warnings (> 3 x LDL)
Avala	SGS_BO	BLANK_BOR	675	0.0259	0	0
Avala	SGS_BO	GREY BLANK**	386	0.1011	6	33
Avala	SGS_CH	BLANK_BOR	83	0.0390	0	1

** not certified for S

No concerns were noted with respect to potential cross contamination of the Avala samples.

12.2.1.2 *DPM Drilling*

The DPM blank results are listed in Table 12.5 and Table 12.6 for Au and S respectively. There are no Ag blank results for the DPM drilling and therefore no control on potential Ag cross contamination.

Table 12.5 – DPM Au Blank Data

Phase	Laboratory	Blank Type	No. of Samples	Mean Au (g/t)	No of Failures (> 10 x LDL)	No of Warnings (> 3 x LDL)
Avala	SGS_BO	BLANK_BOR	2,446	0.005	0	0

Table 12.6 – DPM S Blank Data

Phase	Laboratory	Blank Type	No. of Samples	Mean S (%)	No of Failures (> 10 x LDL)	No of Warnings (> 3 x LDL)
Avala	SGS_BO	BLANK_BOR	1,769	0.0250	0	0

No issues were noted with the blank results for Au and S and therefore, there are no concerns with cross contamination of these samples.

12.2.2 Assay Accuracy

Certified Reference Material (CRM) are pulp samples with a certified expected value and standard deviation and are used to monitor assay accuracy (bias). The standard deviation (SD) is a measure of the amount of variation or dispersion of a set of values, with a low standard deviation indicating that the values tend to be close to the expected value, and a high standard deviation indicating that the values are spread out over a wider range.

CRM used at Timok were a mixture of commercially available CRM (supplied by Geostats) as well as project specific standards (certified by Geostats) which were prefixed with TGP, DPM or Mo. Results were reviewed separately for Au, Ag and S for each phase of exploration.

Shewhart control charts were plotted and three standard deviations from the expected value were used as the failure limits. In a normally distributed population, 99.73% of values should fall within three times the SD. Mean biases for each CRM were calculated [(mean assay value – expected value) / expected value] and the number of failures noted, and failure percentage calculated.

Mean biases exceeding 5% (positive or negative) for CRM with an expected value ≥ 10 x LDL are highlighted in red in the tables below. Where expected values are within 10 x LDL these haven't been highlighted due to the inherent imprecision of these results. Where there were less than five instances of a CRM, these have been excluded from the tables below.

12.2.2.1 Avala Drilling

The CRM results for the Avala exploration are summarised in Table 12.7 to Table 12.9. Failures are noted, but many of these appear to be due to misidentification of the CRM (or blank) type.

Au results are discussed below:

- The Geostats CRM analysed at SGS-BO and SGS-CH performed well with no systematic bias noted.
- There was a systematic negative bias at SGS-AN (-3 to -4% for all CRM).
- The bias noted in the GLG CRM at all laboratories is due to the expected values being below the detection limit.
- Most of the failures and significant bias was noted in the TGP series of CRM (particularly TGP002 and TGO007), but these appear to be due to misidentified CRM. Once corrected, bias is within acceptable limits.

Table 12.7 – Avala Au CRM data (Absolute Bias > 5% and Failures Highlighted in Red)

Lab	Std Code	Assay Method	No. of Samples	Mean Au (g/t)	Exp Value Au (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	G302-7	FA_AAS	67	2.07	2.14	-3%	0	0
SGS-BO	G303-8	FA_AAS	376	0.26	0.26	0%	0	0
SGS-BO	G305-3	FA_AAS	337	0.72	0.72	0%	0	0
SGS-BO	G306-1	FA_AAS	260	0.4	0.41	-2%	0	0
SGS-BO	G307-2	FA_AAS	214	1.06	1.08	-2%	0	0
SGS-BO	G308-3	FA_AAS	130	2.51	2.5	0%	0	0
SGS-BO	G308-8	FA_AAS	1,567	2.46	2.45	0%	2	0
SGS-BO	G311-1	FA_AAS	1,766	0.53	0.52	2%	3	0

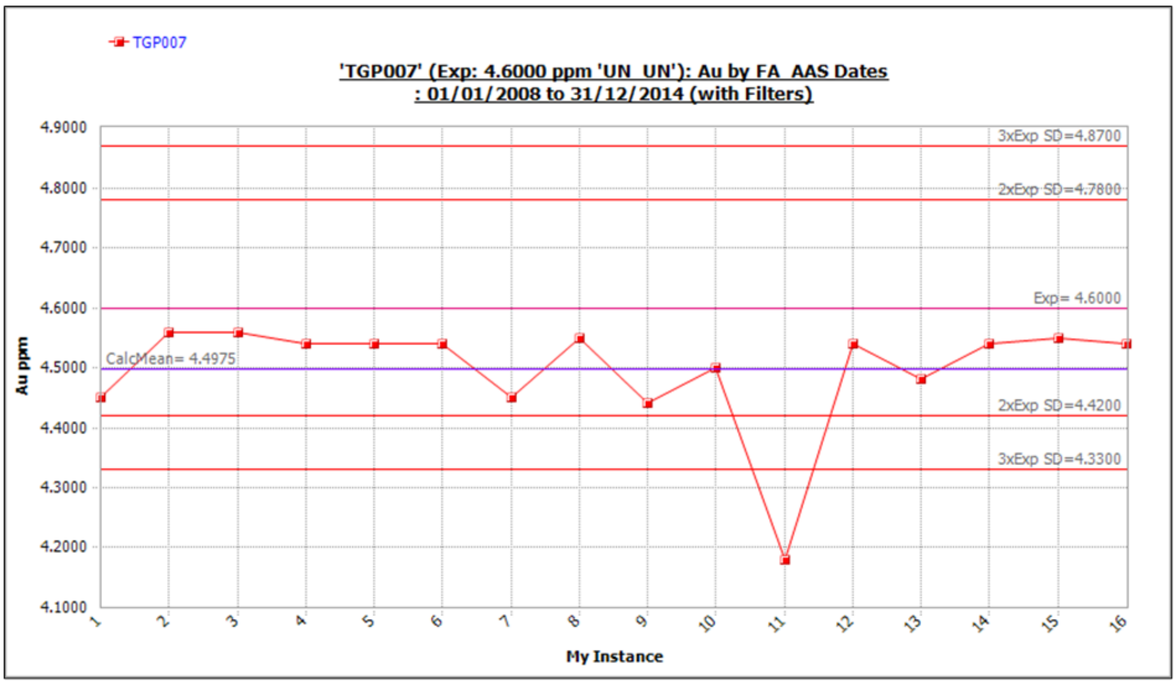
Lab	Std Code	Assay Method	No. of Samples	Mean Au (g/t)	Exp Value Au (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	G901-5	FA_AAS	381	1.64	1.65	-1%	0	0
SGS-BO	G901-7	FA_AAS	213	1.51	1.52	-1%	1	0
SGS-BO	G902-1	FA_AAS	190	0.38	0.4	-5%	0	0
SGS-BO	G903-6	FA_AAS	230	4.09	4.13	-1%	0	0
SGS-BO	G904-7	FA_AAS	1,380	1.62	1.58	3%	0	0
SGS-BO	G905-7	FA_AAS	1,783	3.95	3.92	1%	0	0
SGS-BO	G905-9	FA_AAS	250	1.92	1.86	3%	0	0
SGS-BO	G908-7	FA_AAS	128	4.85	4.82	1%	0	0
SGS-BO	G911-4	FA_AAS	197	2.46	2.43	1%	0	0
SGS-BO	G998-6	FA_AAS	251	0.8	0.8	0%	0	0
SGS-BO	GLG307-1	FA_AAS	523	0.0051	0.0029	76%	8	2
SGS-BO	GLG307-3	FA_AAS	382	0.005	0.0028	79%	0	0
SGS-BO	GLG907-1	FA_AAS	801	0.0051	0.0036	42%	3	0
SGS-BO	GLG911-3	FA_AAS	160	0.005	0.0037	35%	0	0
SGS-BO	TGP001	FA_AAS	21	0.82	0.85	-4%	1	5
SGS-BO	TGP002	FA_AAS	23	0.51	0.47	9%	3	13
SGS-BO	TGP003	FA_AAS	28	0.38	0.38	0%	1	4
SGS-BO	TGP004	FA_AAS	25	1.29	1.31	-2%	0	0
SGS-BO	TGP005	FA_AAS	19	3.01	2.99	1%	2	11
SGS-BO	TGP006	FA_AAS	23	1.74	1.73	1%	0	0
SGS-BO	TGP007	FA_AAS	18	4.01	4.6	-13%	4	22
SGS-BO	TGP007**	FA_AAS	15	4.50	4.6	-2%	1	7
SGS-CH	DPMA	FA_AAS	29	2.158	2.202	-2%	0	0
SGS-CH	DPMB	FA_AAS	36	1.530	1.556	-2%	0	0
SGS-CH	DPMC	FA_AAS	37	5.554	5.873	-5%	2	5
SGS-CH	DPMD	FA_AAS	40	3.19	3.26	-2%	0	0
SGS-CH	DPME	FA_AAS	40	7.41	7.24	2%	0	0
SGS-CH	G306-2	FA_AAS	15	1.06	1.05	1%	0	0
SGS-CH	G901-9	FA_AAS	30	0.69	0.69	0%	1	3
SGS-CH	GBMS304-3	FA_AAS	17	2.68	2.68	0%	0	0
SGS-CH	GBMS304-5	FA_AAS	14	1.70	1.62	5%	1	7
SGS-CH	GLG902-1	FA_AAS	21	0.005	0.0028	79%	0	0
SGS-AN	G308-8	FA_AAS	30	2.3407	2.45	-4%	0	0
SGS-AN	G311-1	FA_AAS	36	0.5028	0.52	-3%	0	0
SGS-AN	G904-7	FA_AAS	42	1.5174	1.58	-4%	0	0
SGS-AN	G905-7	FA_AAS	38	3.805	3.92	-3%	0	0

Lab	Std Code	Assay Method	No. of Samples	Mean Au (g/t)	Exp Value Au (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-AN	GLG907-1	FA_AAS	47	0.0057	0.0036	58%	0	0
SGS-AN	G308-8	FA_AAS	30	2.3407	2.45	-4%	0	0

** Shows results with apparent misidentified CRM removed.
Note that the above table includes the CRM included with the 2014 cyanide leach assays.

Figure 12.1 shows the Shewhart control chart for TGP007 with three misidentified CRM removed. One failure is still apparent and there is a small negative bias (-2%), but this is within acceptable limits.

Figure 12.1 – Avala Au CRM TGP007 – Three Apparent Misidentified CRM Removed



Most of the Ag CRM are too low grade (within 10 x LDL) to allow for meaningful bias calculations. Apart from GBM311-11, where a negative bias of -6% can be noted as well as one failure, no significant or systematic bias for the CRM with expected values ≥ 10 x LDL was observed.

Table 12.8 – Avala Ag CRM Data (Absolute Bias > 5% and Failures Highlighted in Red)

Laboratory	Std Code	Assay Method	No. of Samples	Mean Ag (g/t)	Exp Value Ag (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	GBM303-8	4A_ICPMS	603	6.57	7.00	-6	1	0
SGS-BO	GBM307-3	4A_ICPMS	580	0.55	0.60	-8	6	1
SGS-BO	GBM309-4	4A_ICPMS	619	41.74	42.30	-1	1	0

Laboratory	Std Code	Assay Method	No. of Samples	Mean Ag (g/t)	Exp Value Ag (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	GBM311-11	4A_ICPMS	51	18.54	19.70	-6	1	2
SGS-BO	GBM907-6	4A_ICPMS	620	26.75	26.80	0	13	2
SGS-BO	GBM909-11	4A_ICPMS	46	24.73	25.50	-3	1	2
SGS-BO	GBM909-13	4A_ICPMS	31	129.26	127.30	2	1	3
SGS-BO	TGP005	4A_ICPMS	14	34.05	34.60	-2	0	0
SGS-CH	DPMA	4A_ICPMS	13	5.87	6.43	-9	1	8
SGS-CH	DPMB	4A_ICPMS	15	4.23	4.29	-1	1	7
SGS-CH	DPMC	4A_ICPMS	14	10.37	10.01	4	0	0
SGS-CH	DPMD	4A_ICPMS	15	6.14	6.04	2	0	0
SGS-CH	DPME	4A_ICPMS	21	11.55	11.64	-1	0	0
SGS-CH	GBMS304-3	4A_ICPMS	6	1.00	1.50	-33	0	0
SGS-CH	GBMS304-5	4A_ICPMS	5	0.84	0.80	5	0	0

Table 12.9 summarises the results of the S CRM analysis. No significant or systematic bias was noted for CRM with expected values ≥ 10 x LDL.

Table 12.9 – Avala S CRM Data (Absolute Bias > 5% and Failures Highlighted in Red)

Laboratory	Std Code	Assay Method	No. of Samples	Mean S (%)	Exp Value S (%)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	GBM311-11	LECO	13	3.247	3.300	-2	0	0
SGS-BO	GBM909-11	LECO	20	4.855	4.800	1	0	0
SGS-BO	GBM909-13	LECO	13	19.005	18.100	5	0	0
SGS-BO	Mo1	LECO	20	0.360	0.300	20	17	85
SGS-BO	Mo2	LECO	12	3.309	3.250	2	0	0
SGS-BO	Mo3	LECO	20	2.926	2.940	0	0	0
SGS-BO	Mo4	LECO	15	2.773	2.760	0	0	0
SGS-BO	Mo5	LECO	18	1.819	1.820	0	0	0
SGS-BO	TGP001	LECO	13	0.278	0.319	-13	8	62
SGS-BO	TGP002	LECO	12	0.136	0.017	700	1	8
SGS-BO	TGP003	LECO	13	2.678	2.692	-1	0	0
SGS-BO	TGP004	LECO	15	2.966	2.959	0	0	0
SGS-BO	TGP005	LECO	7	5.689	5.512	3	1	14
SGS-BO	TGP006	LECO	13	1.941	2.018	-4	2	15
SGS-BO	TGP007	LECO	9	1.423	1.447	-2	0	0
SGS-CH	DPMA	LECO	29	12.290	12.400	-1	0	0
SGS-CH	DPMB	LECO	36	12.016	12.000	0	0	0

Laboratory	Std Code	Assay Method	No. of Samples	Mean S (%)	Exp Value S (%)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-CH	DPMC	LECO	37	16.402	16.400	0	0	0
SGS-CH	DPMD	LECO	40	11.546	11.300	2	2	5
SGS-CH	DPME	LECO	40	19.760	19.700	0	0	0
SGS-CH	GBMS304-3	LECO	13	2.302	2.350	-2	0	0
SGS-CH	GBMS304-5	LECO	14	1.015	1.040	-2	0	0

Acceptable accuracy has been demonstrated for the Au results, although there is a small negative bias with the Au samples analysed at SGS-AN. No accuracy issues were observed with the Ag or S CRM results, and whilst the overall percentage of these CRM is appropriate, the number of relevant CRM (> 10 x LDL) is lower than industry standards.

12.2.2.2 DPM Drilling

DPM CRM results are summarised in Table 12.10 and Table 12.11. Results are summarised below:

- Au CRM results are mostly accurate with no significant bias.
  - Multiple failures were observed in TGP002 and TGP003 and investigation is recommended to determine whether there is an issue with the CRM or whether the control limit standard deviation is too small.
- No Ag CRM results are available and therefore there is no control on assay bias for the DPM Ag results.
- The S CRM results are accurate with no significant bias for CRM with expected values ≥ 10 x LDL.

Table 12.10 – DPM Au CRM Data (Absolute Bias > 5% and Failures Highlighted in Red)

Laboratory	Std Code	Assay Method	No. of Samples	Mean Au (g/t)	Exp Value Au (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	G303-8	FA_AAS	103	0.2551	0.26	-2	0	0
SGS-BO	G307-8	FA_AAS	5	1.982	1.99	0	0	0
SGS-BO	G310-4	FA_AAS	83	0.4243	0.43	-1	0	0
SGS-BO	G311-1	FA_AAS	48	0.53	0.52	2	0	0
SGS-BO	G312-7	FA_AAS	135	0.2181	0.22	-1	0	0
SGS-BO	G901-7	FA_AAS	39	1.5244	1.52	0	0	0
SGS-BO	G903-6	FA_AAS	6	4.1667	4.13	1	0	0
SGS-BO	G904-7	FA_AAS	43	1.5595	1.58	-1	0	0
SGS-BO	G905-7	FA_AAS	22	4.08	3.92	4	0	0
SGS-BO	G907-1	FA_AAS	51	0.7908	0.79	0	0	0

Laboratory	Std Code	Assay Method	No. of Samples	Mean Au (g/t)	Exp Value Au (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	G911-4	FA_AAS	28	2.4507	2.43	1	0	0
SGS-BO	GLG911-3	FA_AAS	51	0.0053	0.0037	43	1	2
SGS-BO	OREAS 501c	FA_AAS	6	0.2217	0.221	0	0	0
SGS-BO	TGP001	FA_AAS	346	0.8403	0.85	-1	0	0
SGS-BO	TGP002	FA_AAS	597	0.4527	0.47	-4	141	24
SGS-BO	TGP003	FA_AAS	752	0.3768	0.38	-1	12	2
SGS-BO	TGP004	FA_AAS	152	1.2748	1.31	-3	0	0
SGS-BO	TGP005	FA_AAS	35	3.0263	2.99	1	0	0
SGS-BO	TGP006	FA_AAS	38	1.7255	1.73	0	0	0
SGS-BO	TGP007	FA_AAS	12	4.435	4.6	-4	0	0

Table 12.11 – DPM S CRM Data (Absolute Bias > 5% and Failures Highlighted in Red)

Laboratory	Std Code	Assay Method	No. of Samples	Mean S (%)	Exp Value S (%)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS-BO	GBM311-11	LECO	13	3.247	3.300	-2	0	0
SGS-BO	GBM909-11	LECO	20	4.855	4.800	1	0	0
SGS-BO	GBM909-13	LECO	13	19.005	18.100	5	0	0
SGS-BO	Mo1	LECO	20	0.360	0.300	20	17	85
SGS-BO	Mo2	LECO	12	3.309	3.250	2	0	0
SGS-BO	Mo3	LECO	20	2.926	2.940	0	0	0
SGS-BO	Mo4	LECO	15	2.773	2.760	0	0	0
SGS-BO	Mo5	LECO	18	1.819	1.820	0	0	0
SGS-BO	TGP001	LECO	13	0.278	0.319	-13	8	62
SGS-BO	TGP002	LECO	12	0.136	0.017	700	1	8
SGS-BO	TGP003	LECO	13	2.678	2.692	-1	0	0
SGS-BO	TGP004	LECO	15	2.966	2.959	0	0	0
SGS-BO	TGP005	LECO	7	5.689	5.512	3	1	14
SGS-BO	TGP006	LECO	13	1.941	2.018	-4	2	15
SGS-BO	TGP007	LECO	9	1.423	1.447	-2	0	0
SGS-CH	DPMA	LECO	29	12.290	12.400	-1	0	0
SGS-CH	DPMB	LECO	36	12.016	12.000	0	0	0
SGS-CH	DPMC	LECO	37	16.402	16.400	0	0	0
SGS-CH	DPMD	LECO	40	11.546	11.300	2	2	5
SGS-CH	DPME	LECO	40	19.760	19.700	0	0	0
SGS-CH	GBMS304-3	LECO	13	2.302	2.350	-2	0	0
SGS-CH	GBMS304-5	LECO	14	1.015	1.040	-2	0	0

DPM Au and S CRM results are mostly accurate with no significant bias for CRM with expected values ≥ 10 x LDL. No Ag CRM results are available and therefore there is no control on assay bias for these Ag results.

12.2.3 Assay Precision

Field and laboratory duplicates (duplicates, replicates and splits) as well as external check (umpire) results were compared for the Avala and DPM samples for Au, Ag and S for each laboratory and sample type.

The duplicate data were assessed using average coefficients of variation (CV_{AVR}% = standard deviation/average and presented as a percentage – also known as relative standard deviation) calculated from individual duplicate pairs and averaged using the RMS (root mean squared) approach. This approach is recommended (Abzalov, 2008) as a way of defining a fundamental measure of data precision using duplicate paired data. Mean grade biases were also calculated. Pairs with a mean grade < 10 times the lower detection limit (LDL) were excluded from the review due to the inherent imprecision of results with 10 x LDL.

12.2.3.1 Avala Drilling

Au field duplicates have an acceptable precision for a nuggetty Au deposit (based on Abzalov's acceptable and best values). No bias was observed for the RC field duplicate pairs with some bias in the other sample types (smaller datasets). Most Ag pairs were excluded from the precision review as they were with 10 x LDL. RC field duplicate pairs (S) were repeatable with no significant bias.

Table 12.12 – Avala Field Duplicate Data (Au, Ag and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV (AVR) %	Mean Orig	Mean Dup	Bias %
TR	ALS-RO	Au	g/t	23	12	37	1.77	1.68	-5
RC	SGS-AN		g/t	189	59	39	1.19	1.09	-8
CH	SGS-BO		g/t	79	7	36	0.17	0.23	36
RC	SGS-BO		g/t	6,334	1,478	33	0.87	0.88	0
TR	SGS-BO		g/t	223	57	36	0.80	0.74	-7
CH	SGS-CH		g/t	7	2	32	0.15	0.18	17
TR	SGS-CH		g/t	228	23	40	0.67	0.71	6
RC	SGS-BO		g/t	1,380	8	8	21.38	21.64	1
TR	SGS-BO		g/t	32	2	13	31.85	30.45	-4
RC	SGS-AN	S	%	7	2	15	0.15	0.18	20
RC	SGS-BO		%	6,309	1,797	23	1.85	1.82	-1
CH	SGS-CH		%	7	2	15	0.29	0.33	16

Figure 12.2 and Figure 12.3 show the RC field duplicate scatterplots (with acceptable repeatability) for Au and S respectively. The Ag scatterplot has not been shown as there are only eight relevant pairs.

Figure 12.2 – Avala RC Au Field Duplicates

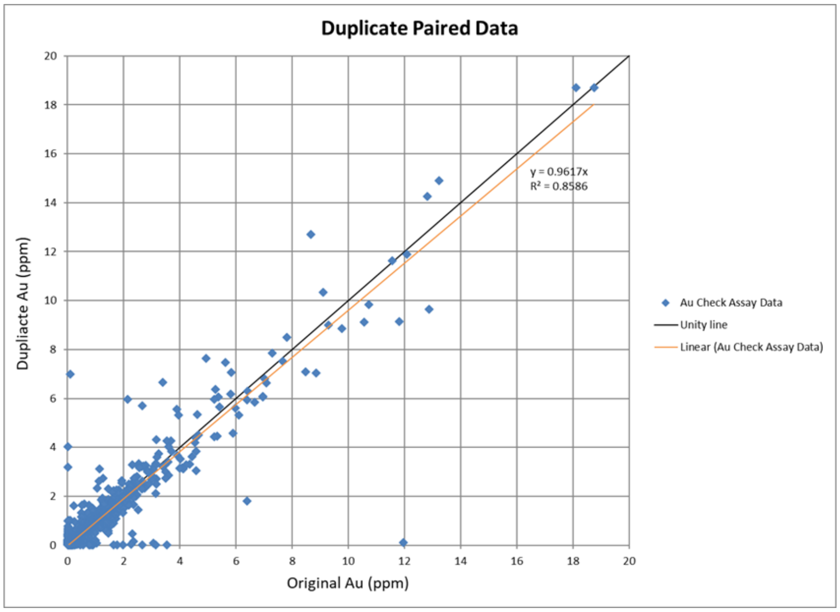


Figure 12.3 – Avala RC S Field Duplicates

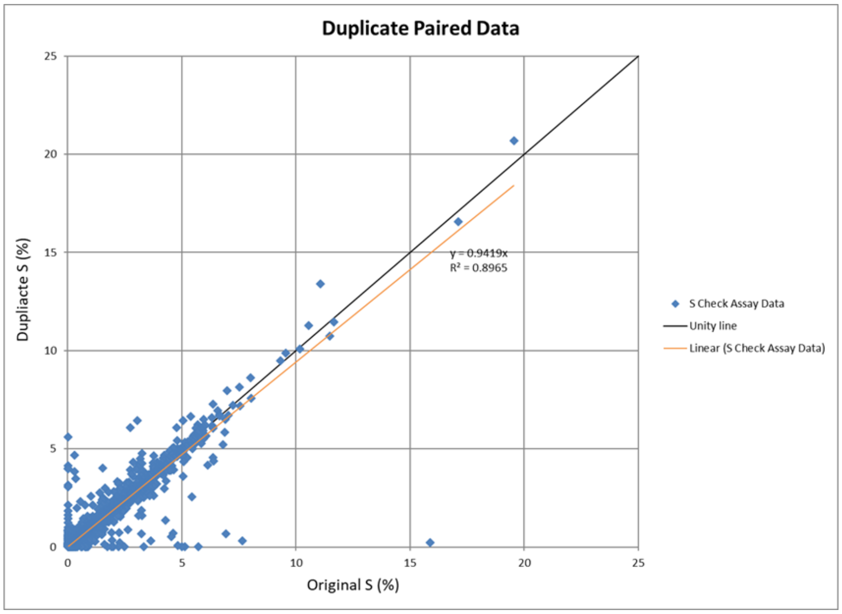


Table 12.13 to Table 12.15 summarise the results for Au, Ag and S laboratory duplicate, laboratory replicate and laboratory split data respectively. Results for pairs without values > 10 x LDL have been excluded. Au and S pairs are precise with no significant bias and most Ag pairs are too low grade to make a meaningful comparison.

Table 12.13 – Avala Laboratory Duplicate Data (Au, Ag and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias %
TR	ALS-RO	Au	g/t	22	14	8	1.33	1.33	0
CH	SGS-BO		g/t	84	8	10	0.30	0.30	-1
HCORE	SGS-BO		g/t	4,527	557	17	0.85	0.83	-3
RC	SGS-BO		g/t	6,105	1,439	18	0.89	0.89	0
TR	SGS-BO		g/t	231	64	16	0.79	0.78	-2
CH	SGS-CH		g/t	13	4	15	0.27	0.29	6
DD	SGS-CH		g/t	40	16	4	1.92	1.93	1
TR	SGS-CH		g/t	224	13	7	0.48	0.46	-3
CH	SGS-BO	Ag	g/t	6	2	4	41.85	39.95	-5
HCORE	SGS-BO		g/t	1,761	11	19	49.67	48.95	-1
RC	SGS-BO		g/t	1,251	13	6	25.25	24.28	-4
TR	SGS-BO		g/t	39	4	3	15.55	15.43	-1
HCORE	SGS-BO	S	%	4,500	1,031	12	1.70	1.73	2
RC	SGS-BO		%	6,104	1,878	8	1.92	1.93	0
CH	SGS-CH		%	13	4	10	0.17	0.15	-11
DD	SGS-CH		%	42	27	2	4.05	4.04	0
TR	SGS-CH		%	235	2	3	0.77	0.78	1

Table 12.14 – Avala Laboratory Repeat Data (Au, Ag and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias %
TR	ALS-RO	Au	g/t	31	25	7	1.47	1.43	-3
CH	SGS-BO		g/t	165	15	5	0.46	0.46	0
HCORE	SGS-BO		g/t	9,123	1,355	4	1.37	1.36	-1
QCORE	SGS-BO		g/t	6	3	6	4.72	4.74	0
RC	SGS-BO		g/t	10,405	2,613	6	1.41	1.40	-1
TR	SGS-BO		g/t	508	146	3	1.30	1.30	0
CH	SGS-CH		g/t	29	8	3	0.55	0.55	-1
DD	SGS-CH		g/t	82	30	4	0.86	0.86	-1

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias %
TR	SGS-CH		g/t	600	89	3	0.72	0.72	0
HCORE	SGS-BO	Ag	g/t	4,079	26	3	22.99	23.13	1
RC	SGS-BO		g/t	2,856	35	3	23.33	23.33	0
TR	SGS-CH		g/t	254	5	8	19.60	20.14	3
HCORE	SGS-BO	S	%	9,787	2,252	4	1.72	1.72	0
RC	SGS-BO		%	13,241	3,742	2	1.92	1.92	0
CH	SGS-CH		%	23	2	1	0.91	0.93	2
DD	SGS-CH		%	91	53	1	3.24	3.24	0
TR	SGS-CH		%	516	8	3	0.72	0.72	0

Table 12.15 – Avala Laboratory Split data (Au, Ag and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias %
TR	ALS-RO	Au	g/t	23	13	6	1.70	1.68	-1
CH	SGS-BO		g/t	92	10	5	0.48	0.49	0
HCORE	SGS-BO		g/t	5,030	638	5	0.79	0.79	0
RC	SGS-BO		g/t	6,900	1,582	6	0.87	0.87	0
TR	SGS-BO		g/t	231	53	4	0.74	0.74	0
CH	SGS-CH		g/t	13	2	1	1.21	1.23	2
DD	SGS-CH		g/t	40	14	4	0.48	0.48	0
TR	SGS-CH		g/t	279	33	3	0.74	0.74	0
HCORE	SGS-BO	Ag	g/t	1,983	6	2	33.17	33.72	2
RC	SGS-BO		g/t	1,470	10	1	23.26	23.54	1
TR	SGS-CH		g/t	118	2	1	30.15	30.45	1
HCORE	SGS-BO	S	%	4,989	1,128	2	1.73	1.73	0
RC	SGS-BO		%	6,932	1,929	2	1.87	1.87	0
DD	SGS-CH		%	44	29	2	2.82	2.81	0
TR	SGS-CH		%	252	3	1	0.59	0.60	2

Umpire (external) check pair results are listed below in Table 12.16. Precision was acceptable for Au, Ag and S for most pairs.

Table 12.16 – Avala Umpire duplicate data (Au, Ag and S)

Lab Orig	Lab Check	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias %
ALS-RO	GEN-PE	Au	g/t	22	14	13	1.33	1.39	4
SGS-CH	ALS-VA		g/t	291	69	8	0.76	0.76	0
SGS-CH	GEN-PE		g/t	300	70	10	0.75	0.78	3
SGS-BO	ALS-VA		g/t	6,126	5,952	16	1.04	1.05	1
SGS-BO	GEN-PE		g/t	5,634	5,436	15	0.99	0.99	0
SGS-BO	ALS-VA	Ag	g/t	1,512	147	21	60.92	63.87	5
SGS-BO	GEN-PE		g/t	1,218	134	14	64.80	63.27	-2
SGS-CH	ALS-VA	S	%	290	45	4	3.53	3.52	0
SGS-CH	GEN-PE		%	50	43	27	3.65	3.73	2
SGS-BO	ALS-VA		%	3,246	2,113	12	2.92	2.87	-2
SGS-BO	GEN-PE		%	2,853	1,911	14	2.90	2.86	-1

Overall an acceptable level of precision was demonstrated for the Au and S samples, although most of the Ag pairs were too low grade to permit a meaningful comparison. Precision was acceptable and no significant between laboratory bias was noted for the external check (umpire) samples.

12.2.3.2 DPM Drilling

DPM Au and S field duplicate results (Table 12.17) are more precise than the Avala results and no significant bias was noted. There are no Ag duplicate data available for review. Figure 12.4 and Figure 12.5 show the scatterplots for the Au and S half core field duplicates respectively.

Table 12.17 – DPM Field Duplicate Data (Au and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias %
CH	SGS-BO	Au	g/t	181	47	26	0.79	0.79	0
HCORE	SGS-BO		g/t	1,803	203	12	0.78	0.78	0
QCORE	SGS-BO		g/t	183	84	15	1.45	1.47	2
TR	SGS-BO		g/t	305	64	23	0.41	0.41	-1
HCORE	SGS-BO	S	%	1,605	155	12	1.50	1.50	0
QCORE	SGS-BO		%	183	39	3	1.47	1.48	0

Figure 12.4 - Scatterplot – DPM Half Core Au Field Duplicates

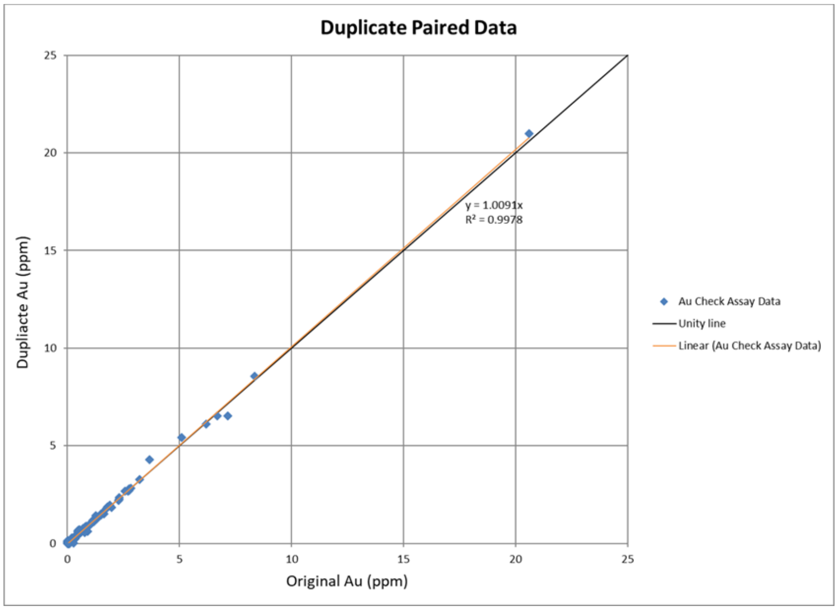
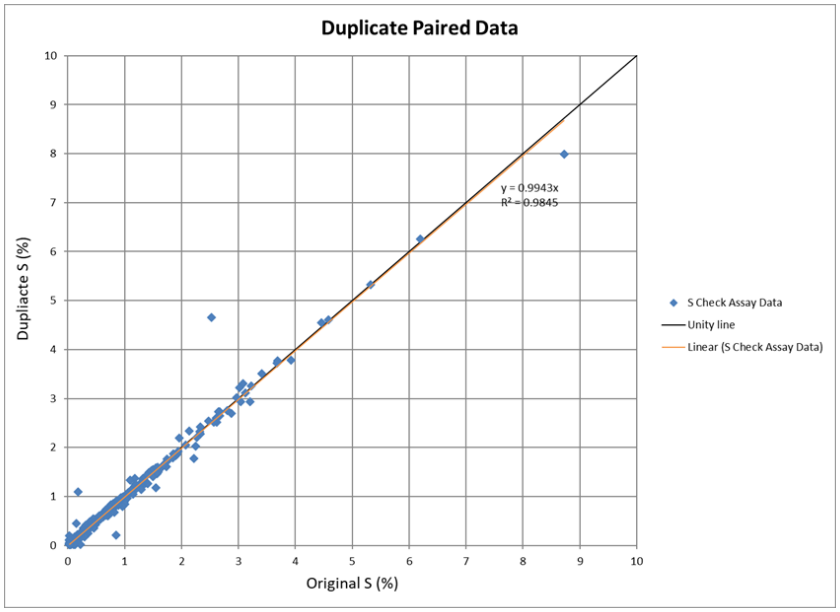


Figure 12.5 - Scatterplot – DPM Half Core S Field Duplicates



Laboratory duplicate pairs are precise with no significant bias (Table 12.18 toTable 12.20).

Table 12.18 – DPM Lab Duplicate Data (Au)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias
HCORE	SGS-BO	Au	g/t	84	18	5	0.95	0.95	0%
TR	SGS-BO		g/t	17	2	18	0.27	0.22	-19%

Table 12.19 – DPM Lab Repeat Data (Au and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias
CH	SGS-BO	Au	g/t	68	21	5	0.72	0.72	0%
HCORE	SGS-BO		g/t	813	109	4	0.61	0.62	0%
QCORE	SGS-BO		g/t	76	40	3	1.40	1.40	0%
TR	SGS-BO		g/t	144	32	4	0.30	0.30	-1%
HCORE	SGS-BO	S	%	723	64	2	1.70	1.70	0%
QCORE	SGS-BO		%	82	16	1	2.25	2.25	0%

Table 12.20 – DPM Lab Split Data (Au and S)

Sample Type	Lab	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias
CH	SGS-BO	Au	g/t	75	21	4	0.68	0.68	0%
HCORE	SGS-BO		g/t	767	92	4	0.87	0.87	0%
QCORE	SGS-BO		g/t	76	31	2	1.58	1.60	1%
TR	SGS-BO		g/t	128	23	3	0.37	0.38	2%
HCORE	SGS-BO	S	%	609	57	2	1.64	1.65	1%
QCORE	SGS-BO		%	77	16	2	1.41	1.41	0%

Umpire (external) check pair results are listed below in Table 12.21. Results were precise for Au and S with no significant bias. Ag pairs were less precise with a bias of 7 to 8% to the original samples (Figure 12.6 and Figure 12.7).

Table 12.21 – DPM Umpire Duplicate Data (Au, Ag and S)

Lab Orig	Lab Check	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias
SGS-BO	ALS-BO	Au	g/t	120	99	12	2.53	2.65	5%
SGS-BO	ALS-RO		g/t	177	175	14	1.77	1.74	-2%
SGS-BO	ALS-BO	Ag	g/t	114	17	22	4.69	4.34	-8%
SGS-BO	ALS-RO		g/t	167	31	36	4.55	4.24	-7%
SGS-BO	ALS-BO	S	%	120	35	6	3.03	3.04	0%
SGS-BO	ALS-RO		%	174	30	11	2.03	2.12	4%

Figure 12.6 – DPM Ag Umpires (SGS-BO / ALS-BO)

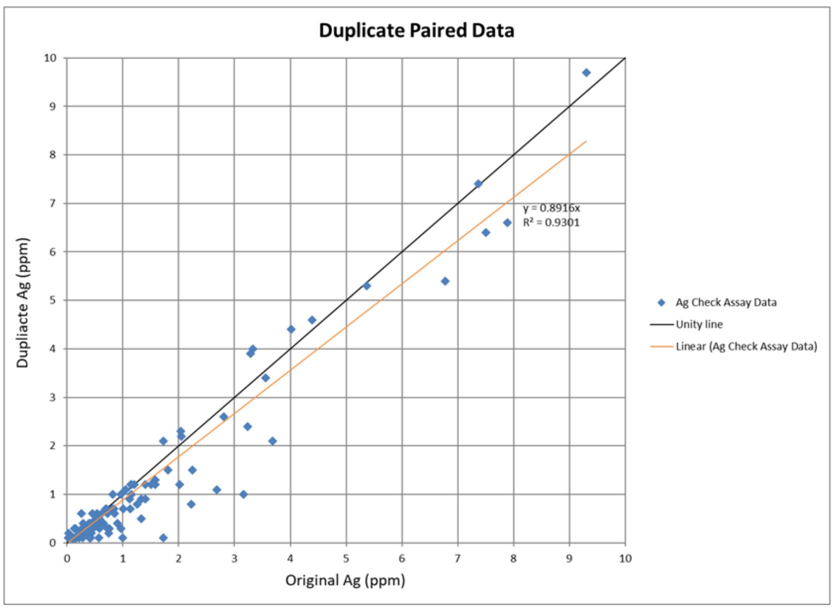
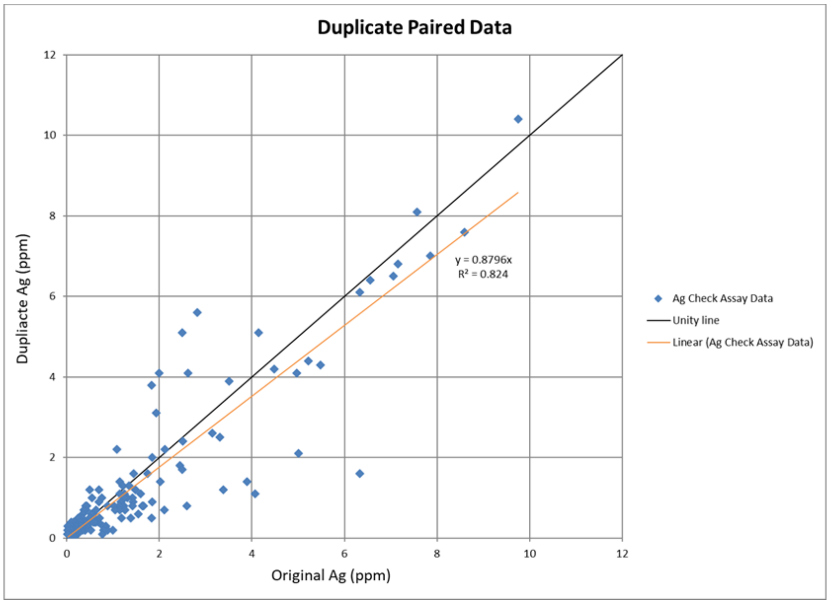


Figure 12.7 – Scatterplot – DPM Ag Umpires (SGS-BO / ALS-RO)



Overall an acceptable level of precision was demonstrated for the Au and S samples with no significant between laboratory bias. No Ag pair data were available for review except for umpire results which had poorer precision and a bias of 7 to 8% to the original results.

12.2.4 Leachwell Samples - 2014

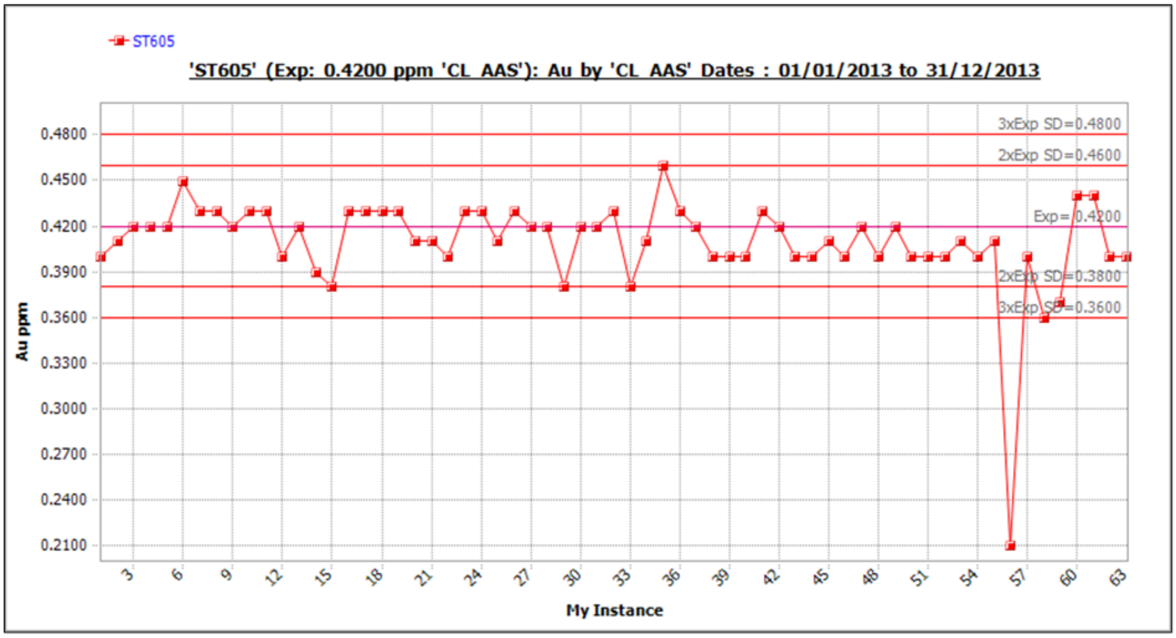
A cyanide Au leach test was completed at SGS Perth in 2014. Samples were analysed by a 50 g fire assay method (FA505) and then processed with an agitated cyanide leach to determine the cyanide extractable Au content. The settled solutions were analysed by AAS (method LWL69J) and the post-leach residue analysed by a 25 g fire assay (FAA303) to determine the undissolved Au content.

CRM with certified cyanide soluble Au values were included with the 2014 cyanide leach samples and results are tabled below. Failures were noted, but no significant bias was observed. The Shewhart control chart for CRM ST605 is shown in Figure 12.8. No significant bias is apparent, and one failure was noted.

Table 12.22 – Leachwell Au CRM data (Absolute bias > 5% and failures highlighted in red)

Laboratory	Std Code	Assay Method	No. of Samples	Mean Au (g/t)	Exp Value Au (g/t)	Mean Bias	No of Failures (> 3 x SD)	Failure %
SGS_PE	ST497	CL_AAS	66	0.98	1.01	-3%	3	5%
SGS_PE	ST589	CL_AAS	66	2.42	2.42	0%	1	2%
SGS_PE	ST605	CL_AAS	63	0.41	0.42	-2%	1	2%

Figure 12.8 – Shewhart Plot Showing Results for Cyanide Soluble Au Results for CRM ST605



The laboratory duplicate results are summarised in Table 12.23. Precision was acceptable and no significant bias was noted with the laboratory duplicates.

Table 12.23 – Avala 2014 Leachwell duplicate data (Au)

Type	Method	Element	Unit	Pairs (total)	Count of pairs (>10 x DL)	CV(AVR) %	Mean Orig	Mean Dup	Bias
LABREP	FAA303	Au	g/t	3,811	2,994	5	0.62	0.62	0%
LABREP	LWL69J		g/t	53	31	6	0.53	0.53	-1%
LABSPLIT	FAA303		g/t	15	11	9	0.36	0.36	-1%

12.3 Twin Hole Analysis

A review of twin hole data was undertaken by DPM, to assess the comparability of RC drillholes and twin diamond drillholes, in order to understand if RC drillholes are an unbiased dataset, usable for resource estimation.

To date, 36 drillhole pairs have been completed at the Bigar Hill prospect, 26 pairs on the Korkan prospect, 7 pairs at the Kraku Pester prospect and 2 at the Korkan West prospect. Diamond twin holes have been designed either to pair an RC drillhole, to pair and extend the depth of drilling of an RC hole (for instance if water was noted during RC drilling) or to pair a drillhole in order to collect material for metallurgical testwork. The spatial locations of drillhole pairs are considered sufficient and have a fair distribution across the deposit footprint. Twins sample spacing was observed to be closest in the Korkan prospect (5.9m apart, on average) whilst furthert apart at Bigar Hill (10.6m apart, on average).

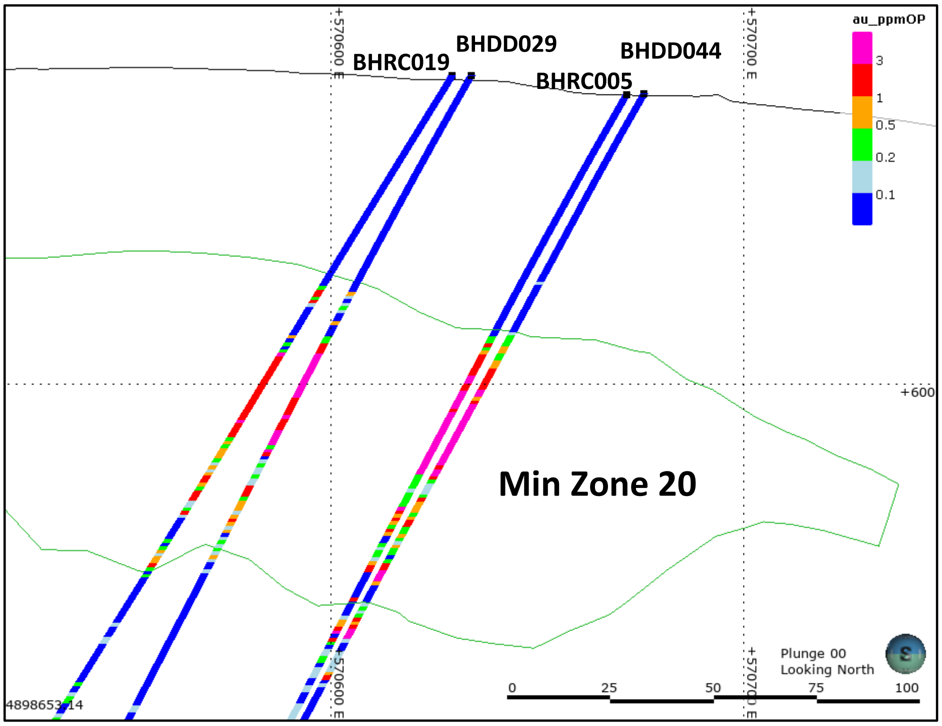
To ensure meaningful comparison of sample representivity, twin samples greater than a distance of 5m were not considered true pairs and were omitted from data analysis. Sample populations were compared using basic grade statistics, visual comparisons, downhole strip logs, histograms, probability and Q-Q plots. Furthermore, scatter plots were prepared by matching whole interval composite pairs.

Table 12.24 – Twin Drillholes Pairs by Deposit, for the Timok Gold Project

Drill Type	No. Pairs			
	Bigar Hill	Korkan	Korkan West	Kraku Pester
Diamond Core vs. Diamond Core	1	1	2	0
Reverse Circulation vs. Reverse Circulation	1	0	0	0
Reverse Circulation vs. Diamond Core	34	25	0	7

When grouped, the Bigar Hill stratigraphic mineralisation hosted in Mineralised Domains 20, 30 & 40 follow very close to a linear line when populations are plotted on Log Q-Q- plots, indicating that the distributions compare well. An indicative cross section showing twin holes at the Bigar Hill deposit is shown Figure 12.9. The twin hole drilling program for RC resource holes within the Minzone 10 domain are insufficient to draw reliable conclusions from, due to the fact that many of the holes were drilled sub-parralell to the domainant trend in mineralsaition, resulting in erratic results that are not suitable for comparison studies.

Figure 12.9 – Cross Section of Twinned Drillhole Pairs from Minzone 20 at the Bigar Hill Deposit

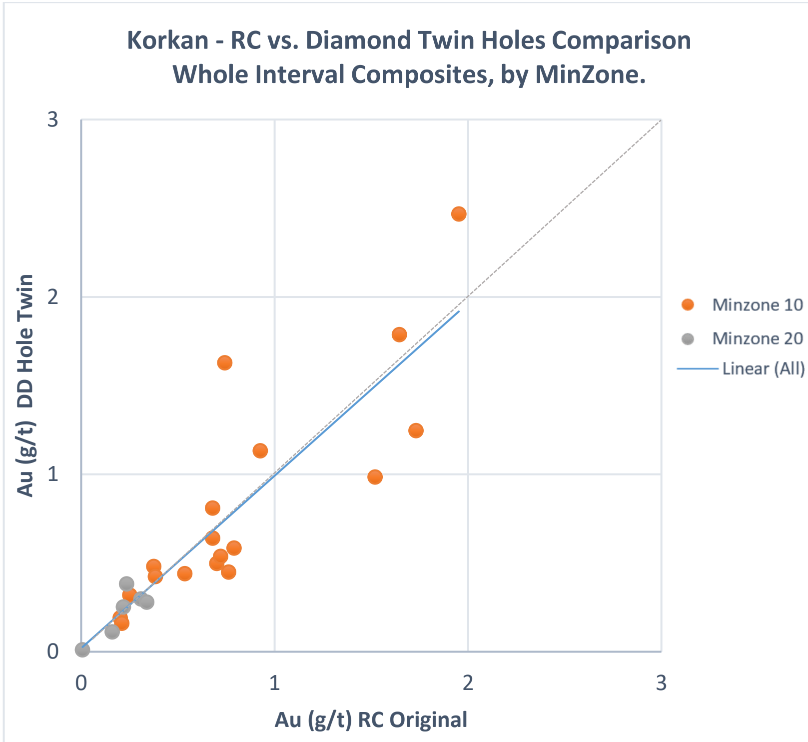


(DPM, 2021)

At the Korkan deposit, RC drillholes and diamond twins of stratigraphic controlled mineralisation plot very close to a linear line when plotted on Log Q-Q plots, indicating that the population distributions compare well. Above 3 g/t, diamond drillholes show a slight bias toward higher values although the number of samples above this value are low, meaning reliable conclusions cannot be made.

Whole interval composites were created across the entire intercept of each mineralisation domain for RC and Diamond Twin holes pair data and values plotted on scatter charts. No distance buffer was applied during this stage of analysis. The results indicate a fair correlation between drillhole intervals, with increasing differences as grade increases. The Korkan performance is better than at Bigar Hill, which is likely due to closer placement of the diamond twin holes at this prospect.

Figure 12.10 – Whole Interval Composites from Korkan, Comparing RC and Diamond twin Holes Intervals.



(DPM, 2021) Linear trend of entire data set in blue

A reccomendation would be to perform additional diamond twins within the Bigar Hill Minzone 10, targeting those RC holes that are correctly orientated to test narrow vein mineralisation perpendicular to mineralisation.

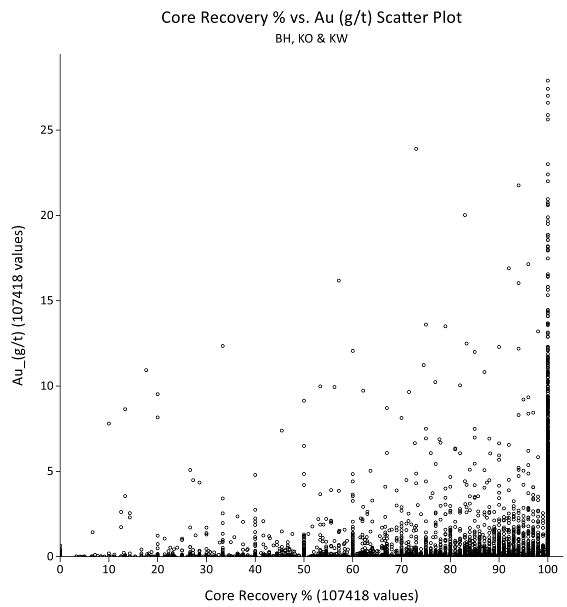
12.4 Sample Bias Analysis

Where multiple sampling methods are used, it is common to compare the frequency distributions of one method versus another to check for biases. Different sampling approaches may possess a particular bias within the dataset which should be considered during resource estimation.

A review of sample bias was conducted by DPM to investigate for potential bias within sample data collected by diamond and RC drilling, as well as channel sampling and trenching. Sample intervals were flagged by lithology, weathering state and mineralisation domains and a theorectical RC recovery value assigned.

For diamond drillholes from the Bigar Hill, Korkan and Korkan west Mineral Resource datasets, an average recovery of 98% was calculated. Figure 12.11 below shows Au (g/t) grade values plotted against core recovery, for all drillholes. No correlations between lower recoveries and increased gold grades were noted during reviews. Lower diamond core recoveries were noted within the overburden domain, which is to be expected considering the unconsolidated nature of this horizon.

Figure 12.11 – Diamond Core recovery (%) vs. Au (g/t) Assay Values

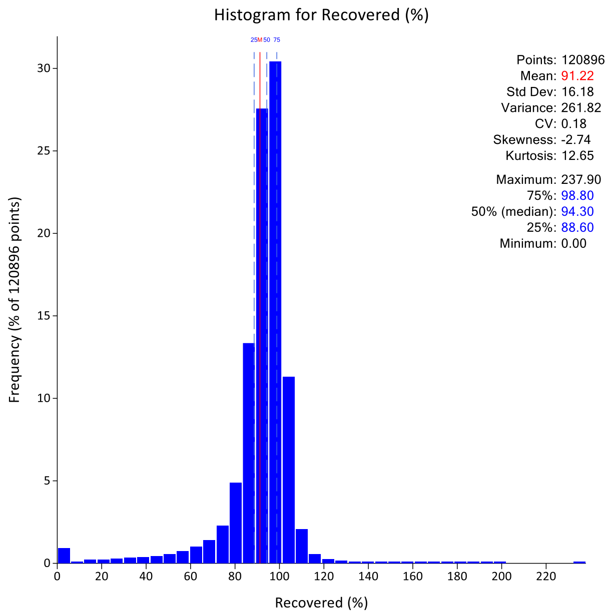


(DPM, 2021)

RC recovery was calculated by dividing actual sample weight (split + reject) by the theoretical sample weight. For RC drillholes from the Bigar Hill, Korkan and Korkan west Mineral Resource datasets, an average recovery of 91.2 % was calculated Figure 12.12.

RC data was grouped by into areas of very low recovery (<20%), low to moderate recovery (20-70%) and very high recovery (>130%) for detailed examination. Areas of very low recovery makeup 1.3% of the dataset and were generally found in the top 5 to 10m of each hole and were noted to be associated with overburden (regolith) were the unsolidated material did not permit a sample to be taken.

Figure 12.12 – Histogram of Theoretical Recovery (%) based on Recorded Sample Weights



(DPM, 2021)

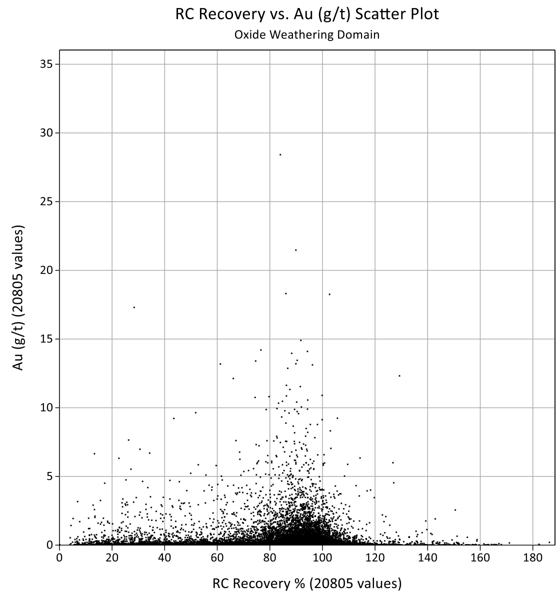
A review of low to moderate recoveries indcated that they are partly associated with less consolidated areas of S1 sandstones, which are known to be interbedded with clay and mudstones. Elsewhere, sporadic areas of low recovery in limestones are likely associated with karstification and karst infill, which will inherently be weaker ground relative to its surroundings.

The recovery data indicates the presence of intervals of excessive (>130%) recovery and such intervals makeup <0.4% of the total population. Upon review of the of the depth of high recovery intervals, it appears that the majority (60%) is associated with the top 10m below surface, coincident with overburden and areas of intense surface weathering.

When reported by weathering state, RC recovery is lowest in oxide weathering domain averaging 80.9%, likely due to the friable nature of the oxide weathering and the overlap of the overburden domain, which almost entirely sits within it. The mean RC recoveries were 92.5% in transitional and in sulphide/fresh domains 93.6%.

RC recovery data was plotted against Au (g/t) grade values for all drillholes. There is no evidence that anomalously low or high sample weights are associated with high (or low) gold grades. Grade vs. RC recovery scatter plots were split by weathering domains. For intervals within the oxide domain, the grade ranges at lower recoveries are similar to those at higher recoveries (Figure 12.13), indicating that no particular bias exists, i.e. comparable grades are found within both higher and lower recovery intervals.

Figure 12.13 – Scatter Plot of RC Recoveries vs. Au g/t Grade Values for the Oxide Weathering Domain



(DPM, 2021)

Channel and trench sample data was reviewed in 3D and one metre sample interval weights were plotted against gold grade. The results approximate a normal distribution and no correlation can be observed between excessive sample weights and Au content.

There is no apparent relationship between dimaond drill core or RC recovery and gold content at Timok. However, mineralised sections, beneath the overburden model with < 70% recovery should be considered for re-drill and sections with <50% should be designated automatic re-drill. This was generally the accepted practice during the 2008-2013 RC drilling campaigns however there are a low number of intervals that have not been re-drilled. A reccomendation would be to review these intervals and re-drill them in preperation for future Mineral Resource updates.

12.5 Qualified Person's Opinion

Overall, there are no concerns with cross contamination and an acceptable level of precision and accuracy was demonstrated for the Au and S samples. Most of the Avala Ag CRM and duplicates were too low grade for a meaningful comparison, but where analysis could be made, there were no fatal flaws. No Ag CRM or duplicates were included by DPM and therefore there is no control on Ag accuracy or precision. However, the contribution of Ag to the contained metal in the Mineral Resource is low and therefore the risk to the Project due to the lack of Ag QC samples is negligible. Anaylsis for sample bias and review of the twin hole results indicates that drilling assay data is an unbiased dataset. The QP concludes that the drillhole data is appropriate for Mineral Resource estimation at the Project.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section covers the testing programs conducted on representative samples from the following Timok deposits: Bigar Hill, Korkan, and Korkan West.

No testing for Kraku Pester related samples were undertaken or included in this Report.

This section includes summaries of past and more recent testwork that were used to develop the process flowsheet and plant design for treating the oxide and transitional ore zones, and conceptual treatment method and metallurgical assumptions used in the sulphide mineral resources estimation process. These testwork programs will be referenced throughout this section of the PFS report.

It is important to note that planning and supervision of the testwork was carried out by DPM. Testwork results were reviewed by both DPM and DRA metallurgists.

DRA provided input during review of the testwork results reported in 2020 and related to the process flowsheet development for the different ore types.

A summary of the testwork was prepared by DPM. Interpretation of the testwork results to develop the process design basis was carried out by DRA.

In the QP's opinion:

- Metallurgical test work completed to date has been appropriate to evaluate and develop appropriate process routes and metallurgical assumptions for the various Timok MRE;
- Metallurgical testing data supports the metal recovery assumptions contained in the LOM plans and metal recovery schedules;
- Samples selected to prepare the metallurgical composites are considered to be representative, and reflects future feedstock to the process plant;
- The QP is not aware of any processing factors or deleterious elements that could have a significant impact on potential economic extraction.

13.1 Historical Testing (2011-2013)

Several metallurgical testwork programs have been undertaken on samples selected from the Project deposits. Avala initiated the scoping-level metallurgical study for the Project in 2011. The primary objective of the scoping-level study in 2011/2012 was to determine the potential recovery of gold and identify potential processing flowsheet options. The focus of these programs was predominantly cyanide leaching, including refractory gold recovery enhancement techniques such as pressure oxidation (POX). The results of these testwork programs have been described within a previous NI 43-101 report ("Timok Gold Project, Serbia, Technical Report and Mineral Resources Estimates for Avala Resources Ltd. (TSX. V. AVZ), Australian Mining Consultants (AMC), AMC (UK) Report No. AMC 413006, 14 October 2013) and are not repeated in detail within this Report.

In essence, combinations of flotation, refractory concentrate treatment (roasting, pressure oxidation or bio-oxidation), with cyanide leaching of oxidised concentrate and flotation tailings, was necessary to achieve reasonable gold recoveries (72 to 76% overall) at prohibitive capital and operating costs. The use of cyanide as a lixiviant for the treatment of Timok ores was not viewed favourably with Serbia's imminent entry into the European Union. A simplified approach aiming to maximise recovery to a flotation concentrate for third party treatment, either domestically or internationally, offers the lowest on-site capital and operating costs and this approach was adopted for the 2014 PEA study design.

In 2013, metallurgical testwork focused on demonstrating that milling and flotation to produce a gold rich sulphide concentrate (for treatment by others) could form the basis of a viable process flowsheet.

13.1.1 Phase 1 – Testwork Program

In general, the Phase 1 testwork program (SGS UK) focused on selective flotation of sulphides and gold associated with non-sulphide gangue to produce a gold-rich concentrate including the optimisation of flotation feed grind size, reagent types and other flotation parameters.

13.1.2 Phase 2 – Testwork Program

The Phase 2 program (SGS UK) further explored the fine grind milling and flotation approach to further confirm the veracity of the production of a saleable gold-rich sulphide concentrate. A wide range of variables were tested including laboratory flotation procedures, flotation froth removal rates and final grind sizes. In general, the optimum flotation reagent types and dosage rates derived from the Phase 1 program were employed for the Phase 2 program in order to minimise testwork variables.

In addition, testwork was initiated to establish the veracity of a beneficiation by size approach where a barren reject fraction could be separated from a high-grade fraction after attritioning.

13.1.3 Phase 3 – Testwork Program

Samples were dispatched to SGS Lakefield (Canada) for bench scale and Woodgrove Technologies Inc. (Woodgrove) "Mini-SFR pilot plant" flotation testwork. Unfortunately, the latter tests were not successful due to procedural issues with the newly commissioned pilot plant equipment, but the SGS laboratory testwork results were instructive.

13.1.4 Phase 4 – Testwork Program

An additional "Extra Flotation Testing", program was initiated in late 2013 to compare the performance of an alternative (less expensive) flotation reagent regime and confirm final concentrate analyses for marketing purposes. Additional high intensity scrubbing testwork was also undertaken with some encouraging results.

13.2 Sample Selection

Detailed metallurgical testwork during 2013 focused on the two largest deposits, Bigar Hill and Korkan, although some work was also completed on samples from the Kraku Pester deposit.

General information regarding the composite samples used for the 2013 flotation (Phase 2) and scrubbing/attrition testwork programs is presented in Table 13.1.

Table 13.1 – Metallurgical Testwork Sample Summary

Composite ID	Deposit	No. of Drill Holes	Intersection (m)	Weight (kg)	Au (g/t)	TS (%)
2013 Flotation Testwork						
Met13_KO_01	Korkan	7	30	62.9	1.51	1.48
Met13_BH_01	Bigar Hill	5	52	113.0	1.45	3.14
Met13_KP_01	Kraku Pester	2	20	50.6	1.41	4.36
2013 Scrubbing/Attrition Testwork						
Met13_KO_02	Korkan	4	30	57	4.47	3.14
Met13_KO_03	Korkan	2	45	77.9	1.07	3.02
Met13_BH_02	Bigar Hill	1	39	43.3	3.48	4.27
Met13_BH_03	Bigar Hill	4	37	55.7	2.52	2.43
Met13_BH_04	Bigar Hill	4	40.5	57.7	0.97	2.09

13.3 Mineralogical Characterisation

Mineralogical characterisation testing was undertaken by SGS UK in 2012 and key aspects of the studies include the following:

- Sulphur grades are relatively low. Head assays for the Phase 2 program composite samples (multi-hole and multi-interval) indicated sulphur to gold ratios of approximately 1.5, 0.87, and 2.80 for the Bigar Hill, Korkan and Kraku Pester samples, respectively.
- X-ray diffraction (XRD) analyses indicate that:
  - Almost the entire sulphides content is present as pyrite, with less than 10% (relative) classed as other sulphides including chalcopyrite and pyrrhotite.
  - Gangue is dominantly quartz, calcite, dolomite (Korkan) and feldspars (Kraku Pester), but some samples also showed significant levels of clays and micaceous minerals.
- QEMSEM analysis indicated that mean pyrite grain sizes of 25 µm, 19 µm and 17 µm were applicable for the Bigar Hill, Korkan, and Kraku Pester composite sub-samples, respectively.
- Approximately 34 to 50% of the free and liberated sulphide particles are under 25 µm, whereas 29 to 38% are above 25 µm in size. The total proportion of pyrite classified as free or liberated was 45 to 92%. The remainder was classified as middlings, where composite particles with quartz/feldspar and calcite represented the main occurrences.

- Pyrite exposure (defined as greater than 50% exposed) was reported as 82%, 74% and 59% for the Bigar Hill, Korkan, and Kraku Pester composite sub-samples, respectively. These levels of exposure should render sulphide particles amenable to flotation, but samples with pyrite exposure values closer to 50% can be expected to exhibit slower flotation kinetics.
- Dynamic secondary ions mass spectrometry (D-SIMS) examinations undertaken in 2012 indicated the presence of substantial quantities of sub-microscopic and solid solution gold within the tested sample.
- Gold was observed within pyrite, chalcopyrite, and iron oxide host minerals. Three different pyrite types were observed (coarse, porous and fine) where each displayed varying gold grades.

The mineralogical characterisation studies are considered to largely explain the flotation performance observed for the various ore type samples during the combined flotation testing results where it may be concluded that:

- Pyrite grain size is relatively fine and a corresponding fine flotation feed size will be required for optimum sulphur recoveries.
- Whilst gold is associated with pyrite, there is a significant proportion associated with the gangue mineral such as silicates. Bright phase analysis targeting gold values would be useful to quantify non-sulphide mineral associations.
- Gold grades varied within the three pyrite types and, as it can be expected that each pyrite type displays differing flotation rates, provided some insight into the flotation concentrate gold grade kinetic profile.

13.4 Comminution

Previous flotation testing (verified by the mineralogical investigations) had demonstrated a relatively fine flotation feed P_{80} size of less than 20 µm was necessary to achieve adequate recoveries.

A number of bench scale comminution characterisation tests were conducted in 2012 on selected samples from the Bigar Hill, Korkan, Kraku Pester, and Korkan East deposits.

The comminution testwork included:

- SAG power index (SPI): A measure of the hardness of the mineralization from a SAG or AG milling perspective.
- Bond ball mill work index (BWi): Standard traditional test used for the assessment of ball milling hardness.
- Modified Bond ball mill work index (ModBond): A simplified procedure conducted in open circuit with slurry rather than the dry Bond method. It requires calibration to the standard Bond test and is not considered appropriate for detailed comminution design activities.

The salient data from these tests is presented in Table 13.2.

Table 13.2 – Comminution Characterisation Testwork Results Summary

Deposit	Unit	Bigar Hill					Korkan	Kraku Pester
Sample ID		**BHDD005**	**562052**	**562053**	**562054**	**MET13-BH-01**	**MET13-KO-01**	**MET13-PE-01**
SPI Testing								
Product P_{64} size	µm		226	221	225	1,700	1,700	306
SPI Testing	min		9.4	80.4	69.5	8	27.1	19.5
Standard Bond Ball Index								
Closing screen size	µm	106						
Product P_{80} size	µm	86						
Bond ball mill Wi	kWh/t	11.9						
Modified Bond Ball Index								
Product P_{80} size	µm	N/A	203	301	336	106	157	120
MB Gpr	g/rev	1.48	1.27	0.51	0.26	1.82	1.55	1.74
WI calculated	kWh/t	12.63	14.43	23.09	27.00	10.20	12.07	10.72

The testwork results indicate:

- SAG milling amenability testing is limited to the SPI methodology, but indicates a range but generally soft to moderate characteristics.
- Bond work index parameters have been predominantly determined by the SGS ModBond procedure. Direct comparison to the standard Bond Ball Mill Work Index results is available for one sample (BHDD005) and indicates a good correlation.

13.5 Flotation

As outlined in Section 13.1, four (4) testwork programs were completed on deposit samples where flotation formed a major component of the scope.

The following sections describe the general features of the flotation testwork elements of these programs, summarise the reported results and outline the metallurgical basis for the PEA flotation circuit design.

13.5.1 Phase 1 Flotation Testing

The Phase 1 testwork program (SGS UK) investigated various parameters to demonstrate the Ultra Fine Grind (UFG) / flotation concept including some optimisation of flotation feed grind size, reagent types and other flotation parameters.

The program included laboratory batch flotation and comparative gravity concentration testwork on several samples representing Bigar Hill, Korkan, and Kraku Pester ore types.

The program also included the following related testing:

- Detailed head assays and mineralogical characterisation.
- Gravity recovery testing. Comparative flotation testing demonstrated superior gold recoveries and grade relationships and gravity recovery testing was abandoned during the program.
- Batch flotation testing at various grind sizes and conditions, including the investigation of several reagent addition regimes.

Results of this testing showed that high gold and sulphur recoveries could be obtained, albeit at high concentrate mass pull weights (>40% by weight).

Bigar Hill was shown to be the best rougher flotation performer of the samples tested, followed by Korkan and then Kraku Pester. Results included:

- Bigar Hill cumulative gold and sulphur recoveries of 86.8% and 96.9%, respectively to a concentrate weight of 24.5%.
- Korkan cumulative gold and sulphur recoveries of 71.8% and 80.2%, respectively to a concentrate weight of 17.9%.
- Kraku Pester cumulative gold and sulphur recoveries of 60.4% and 73.2%, respectively to a concentrate weight of 18.4%.

The testwork concluded:

1. Direct correlation between Au and S recovery as a function of concentrate mass pull.
2. Increase in liberation size resulted in slightly improved selectivity.
3. Irrespective of the pre-concentration method used (gravity or flotation), or the reagent suite, results broadly speaking lie on the same mass pull versus recovery curve.

The metallurgical performance of the ore appears to be deemed to be mineralogically constrained due to the gold being predominantly hosted in pyrite at an extremely fine grain size of 16 µm.

There are also pyrite minerals associated with non-sulphide gangue; mainly quartz calcite and other carbonates.

13.5.2 Phase 2 Flotation Testing

The Phase 2 testwork program (SGS UK) further explored fine grinding and flotation approach to generally confirm the veracity of the production of a saleable gold-rich sulphide concentrate. A wide range of flotation parameters were tested including laboratory flotation procedures, flotation froth removal rates and flotation feed grind sizes.

In general, the optimum flotation reagent types and dosage rates derived from the Phase 1 program were employed for the Phase 2 program in order to minimise testwork variables.

These reagents included proprietary flotation collectors developed specifically for the flotation of gold-containing oxide-based minerals as well as more common sulphide mineralisation.

The Phase 2 testing program included laboratory batch flotation testwork on "MET13" composite samples representing Bigar Hill (BH-01), Korkan (KO-01) and Kraku Pester (PE-01) ore types. In general, the following flotation related testing was conducted:

- Detailed head assays and mineralogical characterisation.
- Investigation of coarse flotation applicability via four-stage sequential flotation tests at reducing P80 grind sizes, i.e.: 75 µm, 53 µm, 38 µm and 20 µm. The BH test indicated very good sulphur recoveries (to 92%) but relatively poor gold recoveries (up to 68%) as gold-containing oxide particles were not floated during the procedure due to the lack of flotation time at the finer final tested grind size. Similar results were reported for a KO sequential test and this approach was abandoned for the remainder of the program.
- All rougher-scavenger flotation testing was undertaken using the Phase 1 Variability testing FT7 reagent addition regime, i.e.: 100 g/t MaxGold 900 and 100 g/t Aero 3418A collectors, no activation and minor dispersant additions.
- Several physical rougher flotation test methods were investigated where various grind sizes, scrape rates, cell types and general rougher concentrate pulling procedures were adopted.

The Bigar Hill ore type is the least mineralogically constrained in terms of pyrite associations, followed secondly by Korkan and thirdly by Kraku Pester. The metallurgical results reported herein, show that this mineralogy is the main driver on the metallurgical responses, which are not surprisingly, best for Bigar Hill and worse for Kraku Pester.

It was clear from the results that the grind size required for effective liberation of sulphides was circa 75µm for Bigar Hill (slightly finer for Korkan and Kraku Pester), but that ultimately a 20 µm grind is required to maximise rougher gold recovery on Bigar Hill and Korkan ore types.

The Kraku Pester ore type is clearly still mineralogically constrained at 20 µm and so some ultra-fine (10 and 5 µm) grinds were performed on this ore type alone but were unsuccessful. As a consequence, no further testing was conducted on this ore type, and Bigar Hill and Korkan became the main focus of testing.

13.5.2.1 Rougher Tests

Bigar Hill was shown to be the best rougher flotation performer of the samples tested, followed by Korkan and then Kraku Pester, as illustrated in Table 13.3.

The results highlighted that employment of UFG to approximately 20 µm with a slower froth scraping rates resulted in much improved selectivity when compared to the Phase 1 program.

Little further work was subsequently conducted on Kraku Pester samples due to the difficult metallurgical characteristics and relatively small resource base.

Table 13.3 – Rougher Optimum Test Results Summary

Ore Type (Sample)	Sample ID	Mass Pull (%Wt.)	Grade		Recovery	
			Au, g/t	TS, %	%Au	%TS
Bigar Hill	(MET13-BH-01)	24.5	5.50	9.7	86.8	96.9
Korkan	(MET13-KO-01)	17.9	6.32	6.7	71.8	80.2
Kraku Pester	(MET13-KP-01)	24.8	2.98	12.7	51.6	74.7

13.5.2.2 Open Cycle Cleaner Tests

Two (2) cleaner tests were performed on Bigar Hill and Korkan. Results are summarised in Table 13.4.

Table 13.4 – Cleaner Test Results

Sample Description	Bigar Hill (FT8BH)				Korkan (FT8KO)			
	Weight %	%Cum Recovery	Grade		Weight %	%Cum Recovery	Grade	
			Au g/t	Cum Au g/t			Au g/t	Cum Au g/t
Cleaner 1 conc.	1.6	22.4	19.9	19.9	1.00	17.9	26.4	26.4
Cleaner 1-2 conc.	3.41	50.3	21.9	20.9	1.71	29.4	23.7	25.3
Cleaner 1-3 conc.	4.08	60.4	21.4	21.0	2.10	35.7	23.6	25.0
Cleaner 1-4 conc.	4.50	67.0	22.0	21.1	2.46	39.5	15.8	23.6
Cleaner 1-5 conc.	**4.68**	**69.2**	**17.5**	**21.0**	**2.92**	**45.4**	**18.9**	**22.9**
Cleaner Tailings			3.69				4.89	
Rougher Conc.	7.8	77.3	14.1		7.2	59.6	12.2	
Rougher Tailings			0.35				0.64	
Calculated Head	100.0	100.0	1.42		100.0	100.0	1.47	

For the Bigar Hill ore, gold recovery to the cleaner concentrate was 69.2% at a cleaner concentrate grade of 21.0 g/t Au. Whilst for the Korkan ore, the combined gold recovery to the cleaner concentrate was 64% at a cleaner concentrate grade of 22.9 g/t Au.

All cleaner tests were conducted in open circuit so the recoveries stated above reflect gold losses to the cleaner tail stream. In a locked cycle test the cleaner tail would be recycled back to the head of the cleaner circuit (or similar such as a separate cleaner-scavenger circuit) and a proportion could report to the final concentrate, thus improving overall gold recovery.

However, this also implies that the final concentrate weight will increase with consequent reduction in the final concentrate gold grade.

Whilst the BH curve is not typical (it is considered unusual for both grade and recovery to increase on a cumulative basis), a final concentrate weight of approximately 4.5% appears optimum for this sample.

There may have been some capacity to increase final concentrate recovery for the KO sample by increasing the target concentrate weight to around 3.5%, although this would probably have reduced the gold grade to below g/t for this sample.

The average calculated Au head grades for the MET samples used in all Phase and SGS Lakefield flotation tests were 1.47 g/t Au and 1.44 g/t Au for the BH and KO ore types.

It should be noted that although cleaner concentrates in excess of 20 g/t Au were produced, the cleaner tests were conducted on rougher concentrates that were higher grade and lower recovery than indicated in the optimum rougher tests summarised in Table 13.3 above.

13.6 Dundee Precious Metals Testing (2018-2020)

Following an updated MRE in 2018, which significantly increased the proportion of oxide material in the Mineral Resource, DPM completed a number of different testwork phases to develop the flowsheet and establish the metallurgical performance of the oxide and transitional ore types processing using heap leach technology.

13.6.1 Phase 1 Testwork Program

Testing at SGS Lakefield in 2018 focussed on scoping level testing on samples representing the oxide (weathered) and transitional ore types from the Bigar Hill, Korkan and Korkan West deposits. The testwork program included coarse and fine bottle roll leach tests, as well as column leach tests to be able to evaluate both heap leach, and conventional Carbon-In-Leach (CIL) technologies.

Based on the column leach test results, metallurgical recoveries, and process operating costs were generated to be used in open pit optimisations, and to prepare the maiden MRE for the oxide and transitional ore zones.

13.6.2 Phase 2 Testwork Program

Additional metallurgical samples were selected and tested at SGS in 2019.

The results of these tests formed the basis of the PEA undertaken by CSA Global and others in 2019.

13.6.3 Phase 3 Testwork Program

To further support the PFS, additional testing was undertaken at SGS Lakefield in 2019. Drill hole intervals were selected to prepare representative composites of the oxide and transitional ore types from the Bigar Hill, Korkan, and Kraku Pester deposits. The results from these tests were subsequently used to support the process design and metallurgical assumptions for the HLF.

13.6.4 Phase 4 Testwork Program

As part of the PFS, testwork was undertaken by SGS Lakefield in 2020. The testwork program investigated alternative lixiviants (thiosulphate/CN Lite) for the leaching representative oxide samples, whilst flotation was evaluated for treating representative transitional samples.

Additional confirmation flotation testwork was undertaken by XPS in 2020 on representative sulphide samples to confirm that a saleable concentrate could be produced. As an alternate to the PEA base case flowsheet of producing a saleable gold concentrate investigative Neutral Albion Leach (NAL) tests were also carried out on a gold bearing sulphide concentrate as part of the preliminary evaluation of the Albion Process™.

Ore characterisation testing was also carried out by Wardell Armstrong International on representative sulphide samples.

A testwork summary matrix is shown in Table 13.5.

Table 13.5 – Testwork Summary Matrix

Laboratory/ Consultant	Mineralogy	Comminution	Gravity	Coarse Bottle Roll	Column Leach	Flotation	CIL	POX	Albion Process	ABA	Dewatering	Environ mental
SGS Mineral Services UK Ltd						X						
Resource Development Inc.					X		X					
						X	X					
						X	X	X				
SGS Minerals Services UK Ltd	X											
						X						
						X						
	X											
	X											
	X											
DST												X
SGS					X							
Enviroleach												X
SGS Canada Inc.				X	X							
SGS Canada Inc.						X			X	X		
SGS Canada Inc.							X					
SGS Canada Inc.							X					
XPS/ Activation Laboratories Ltd	X		X			X	X			X		
Wardell Armstrong International		X										
XPS/Outotec											X	
XPS/Outotec											X	
XPS/Outotec											X	

13.7 Sample Selection and Representivity

13.7.1 Oxide / Transitional

13.7.1.1 Phase 1 Testwork Program

Samples representing the oxide and transitional ore zones were selected for testing to derive metallurgical parameters for the maiden MRE for the Bigar Hill, Korkan, and Korkan West deposits.

A series of metallurgical twin-holes were completed during December 2017 in order to collect fresh material for testwork.

The sample composites targeted mineralisation within the S1 stratigraphic horizon, which is the dominant host of mineralisation at the Project. Samples were selected based upon logged weathering style, visual estimates of the percentage of oxidation and review of the sulphur assay data. All sample composites are located within conceptual pit shells, used to constrain MREs.

In total, four (4) metallurgical composites were collected and a description is shown as Table 13.6.

Table 13.6 – Metallurgical Composites (MRE)

Composite ID	Deposit	Oxidation State	Description
Met18_KO_01	Korkan	Oxide	Oxidized, sedimentary breccia-conglomerate with quartz and limestone pebble fragments within a sandy matrix. Taken from the S1 unit from within the Korkan Deposit.
Met18_KO_02	Korkan	Transitional	Transitional S1 unit material from the Korkan Deposit, comprised of alternating zones oxide and sulphide mineralisation, of equal proportions. The rock type is comprised of a sedimentary breccia-conglomerate with quartz and limestone pebble fragments within a sandy, or to a lesser extent mudstone matrix.
Met18_BH_01	Korkan	Oxide	Oxidized S1/S2 horizon material from the Bigar Hill Deposit. Coarse to medium grained sandstone with interbedded mudstone laminas within S1 fraction of the sample.
Met18_KW_01	Korkan West	Oxide	Oxidized S1 calcareous, medium/fine grained sandstone from the Korkan West Deposit.

Details of the metallurgical drill holes use to select drill hole intervals for which composites were prepared are shown in Table 13.7.

Table 13.7 – Metallurgical Drill Hole Summary (MRE Samples)

Drill Hole ID	Prospect	Easting	Northing	Elevation	Azimuth	Dip	Target Depth	Composite Ore Type
BHDDMET01	Bigar Hill	570478	4898645	675	280	-55	80	Oxide
BHDDMET02	Bigar Hill	570622	4898611	688	275	-64	175	Sulphide
KODDMET01	Korkan	570227	4900437	610	45	-45	70	Oxide/Trans
KODDMET02	Korkan	570266	4900472	622	45	-70	60	Oxide/Trans
KODDMET03	Korkan	570266	4900473	622	40	-50	65	Oxide/Trans
KWDDMET01	Korkan West	569839	4899342	640	190	-60	70	Oxide

13.7.1.2 Phase 2 Testwork Program

Additional samples representing the oxide and transitional ore zones were selected for testing to provide additional statistical representation to support the 2019 PEA. Details of the metallurgical drill holes are shown in Table 13.8.

Table 13.8 – Metallurgical Drill Hole Summary (2019 PEA Samples)

Drill Hole ID	Prospect	Easting	Northing	Elevation	Azimuth	Dip	Target Depth	Composite Ore Type
BHDDMET003	Bigar Hill	570146	4898529	672	270	-65	70	Oxide + Transitional
BHDDMET004	Bigar Hill	570302	4898239	718	270	-55	240	Sulphide
BHDDMET005	Bigar Hill	570143	4898689	640	255	-65	60	Transitional
KWDDMET002	Korkan West	569802	4899367	653	245	-60	90	Oxide
KODDMET009	Korkan West	570222	4899140	685	220	-50	60	Oxide
KODDMET004	Korkan	570309	4900620	648	55	-55	100	Transitional
KODDMET010	Korkan	570435	4900442	673	40	-55	150	Sulphide

13.7.1.3 Phase 3 Testwork Samples

Additional samples representing the oxide and transitional ore zones were selected for testing to provide additional statistical representation to support the PFS.

Details of the metallurgical drill holes are shown in Table 13.9.

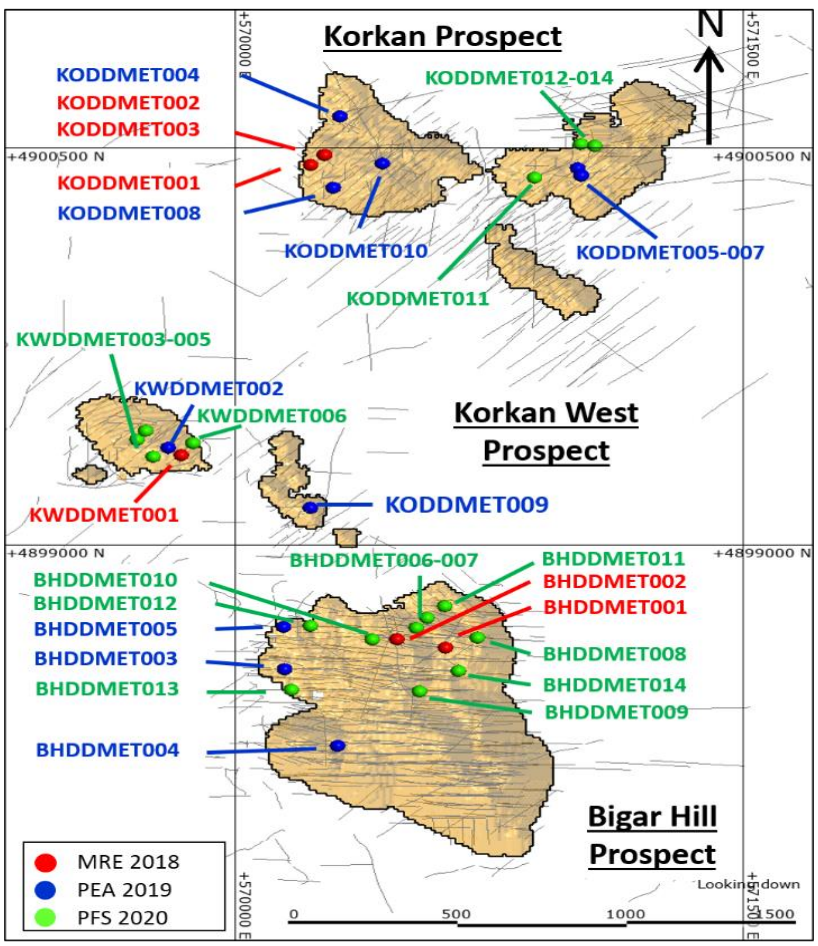
Table 13.9 – Metallurgical Drill Hole Summary (PFS Samples)

Drill Hole ID	Prospect	Easting	Northing	Elevation	Azimuth	Dip	Target Depth	Composite Ore Type
BHDDMET006	Bigar Hill	570535	4898686	667	-65	270	80	Oxide/Trans
BHDDMET007	Bigar Hill	570566	4898724	657	-60	300	110	Oxide
BHDDMET008	Bigar Hill	570715	4898649	667	-65	270	120	Sulphide

Drill Hole ID	Prospect	Easting	Northing	Elevation	Azimuth	Dip	Target Depth	Composite Ore Type
BHDDMET009	Bigar Hill	570544	4898445	739	-64	295	237	Oxide/Trans
BHDDMET010	Bigar Hill	570404	4898642	663	-65	270	100	Oxide/Trans
KODDMET006	Korkan	571023	4900397	668	-50	45	30	Oxide
KODDMET007	Korkan	571021	4900396	668	-88	90	20	Oxide
KODDMET008	Korkan	570290	4900350	642	-56	45	148	Oxide/Trans
KODDMET009	Korkan West	570222	4899140	685	-50	220	60	Oxide
KODDMET010	Korkan	570435	4900442	673	-60	40	151	Oxide + Sulphide
KODDMET012	Korkan	571061	4900509	628	-60	40	100	Sulphide

A plan view showing the location of the metallurgical drill holes generated during the MRE, PEA, and PFS stages in each of the respective deposits is shown in Figure 13.1.

Figure 13.1 - Drill Holes Location / PFS Samples



13.7.2 SULPHIDE

13.7.2.1 Comminution Samples (2020)

A total of 27 intervals of half drill core, weighing a total of 167.3 kg, were submitted to Wardell Armstrong International in September 2020 for testing. The material submitted represented two (2) different deposits Bigar Hill and Korkan.

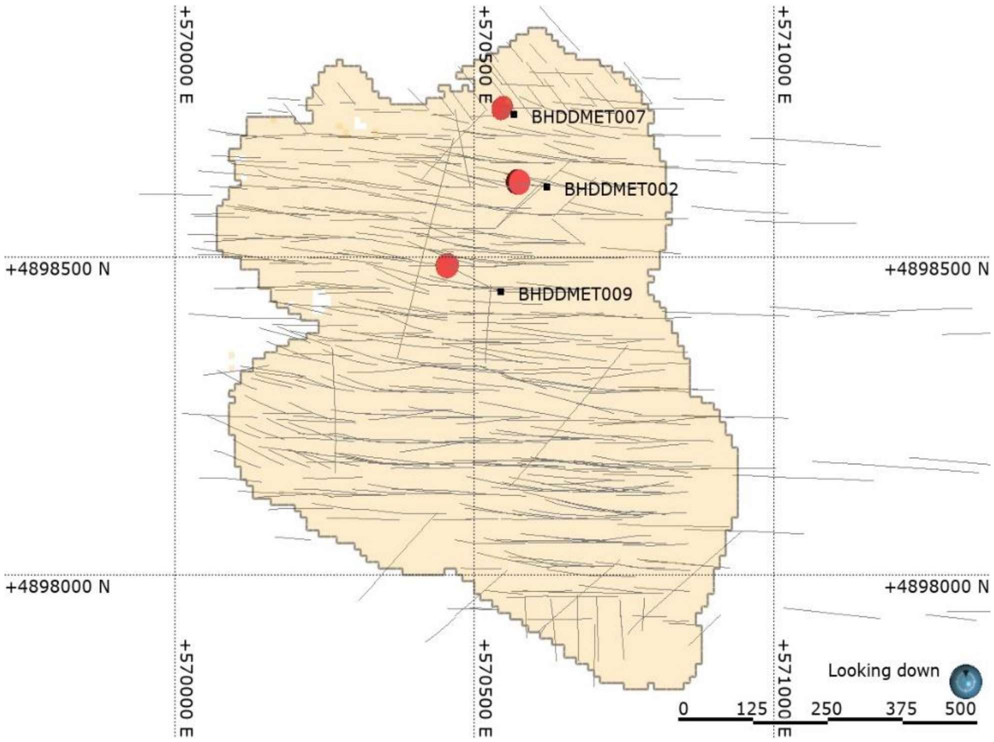
A summary of the drill hole intervals selected to prepare the Bigar Hill composite (BH_SU_20_02) are shown in Table 13.10.

The location of the drill holes from which sample intervals were selected to prepare the BH_SU_20_02 composite is shown in Figure 13.2.

Table 13.10 – Metallurgical Drill Hole Summary (Comminution Sulphide Samples)

COMP ID	HOLE ID	FROM	TO	SAMPLE TYPE	Interp Unit	LI1	SA_OX
BH_SU_20_02	BHDDMET002	107	108	DDH1/2_S	S2	SSS	FRS
BH_SU_20_02	BHDDMET002	108	109	DDH1/2_S	S2	SSS	FRS
BH_SU_20_02	BHDDMET002	109	110	DDH1/2_S	S2	SSS	FRS
BH_SU_20_02	BHDDMET002	110	111	DDH1/2_S	S2	SSS	FRS
BH_SU_20_02	BHDDMET002	111	112	DDH1/2_S	S2	SSS	FRS
BH_SU_20_02	BHDDMET007	44	45	DDH1/2_S	SLS	SLS	FRS
BH_SU_20_02	BHDDMET002	112	113	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET002	113	114	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET002	114	115	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET002	115	116	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET002	116	117	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET002	117	118	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET002	118	119	DDH1/2_S	S1	SSS	FRS
BH_SU_20_02	BHDDMET009	220	221	DDH1/2_S	S1 - BBX	SBX	FRS
BH_SU_20_02	BHDDMET009	226	227	DDH1/2_S	S1 - BBX	SBX	FRS

Figure 13.2 –Drill Hole Location / BH_SU_20_02 Comp ID



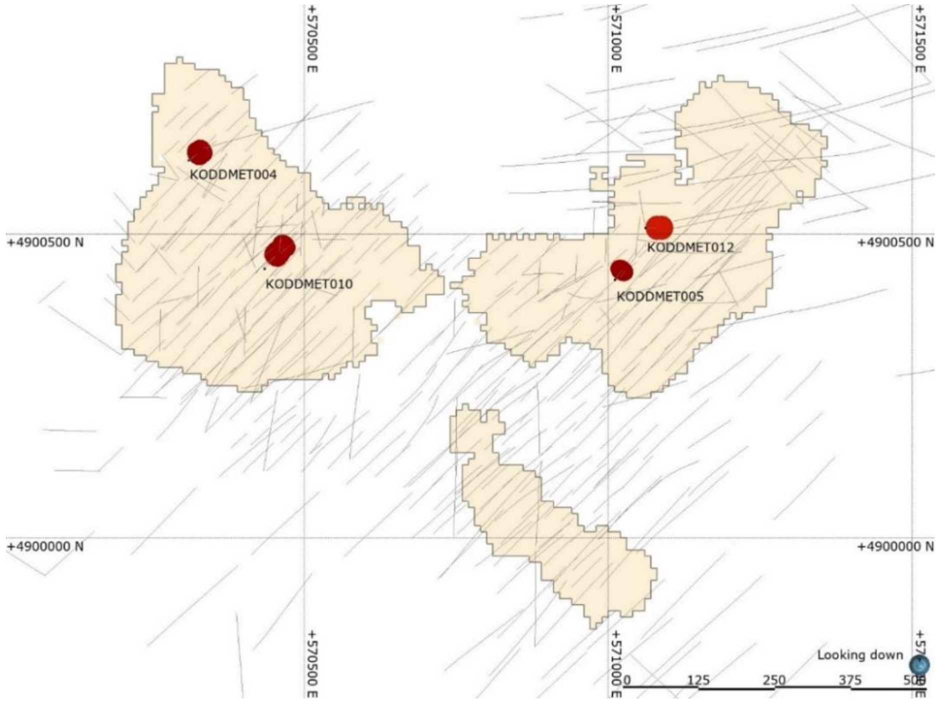
A summary of the drill hole intervals selected to prepare the Korkan composite (KO_SU_20_02) are shown in Table 13.11.

Table 13.11 – Metallurgical Drill Hole Summary (Comminution Sulphide Samples)

COMPID	HOLEID	FROM	TO	SAMPLE TYPE	Interp Unit	LI1	SA_OX
KO_SU_20_02	KODDMET005	32	33	DDH1/2_S	KLS	XMO	FRS
KO_SU_20_02	KODDMET012	55	56	DDH1/2_S	KLS	SLS	FRS
KO_SU_20_02	KODDMET012	78	79	DDH1/2_S	KLS	XMO	FRS
KO_SU_20_02	KODDMET004	32	33	DDH1/2_S	S1	SXP	FRS
KO_SU_20_02	KODDMET004	36	37	DDH1/2_S	S1	SXP	FRS
KO_SU_20_02	KODDMET004	37	38	DDH1/2_S	S1	SXP	FRS
KO_SU_20_02	KODDMET004	46	47	DDH1/4_S	S1	SXP	FRS
KO_SU_20_02	KODDMET010	50	51	DDH1/4_S	S1	SMU	FRS
KO_SU_20_02	KODDMET010	51	52	DDH1/4_S	S1	SMU	FRS
KO_SU_20_02	KODDMET010	52	53	DDH1/4_S	S1	SMU	FRS
KO_SU_20_02	KODDMET010	63	64	DDH1/4_S	S1	SSS	FRS
KO_SU_20_02	KODDMET010	89.7	91	DDH1/4_S	BBX	XPO	FRS

The location of the drill holes from which sample intervals were selected to prepare the KO_SU_20_02 composite is shown in Figure 13.3.

Figure 13.3 – Drill Hole Location / KO_SU_20_02 Comp ID



13.7.2.2 Flotation Composite Samples (2020)

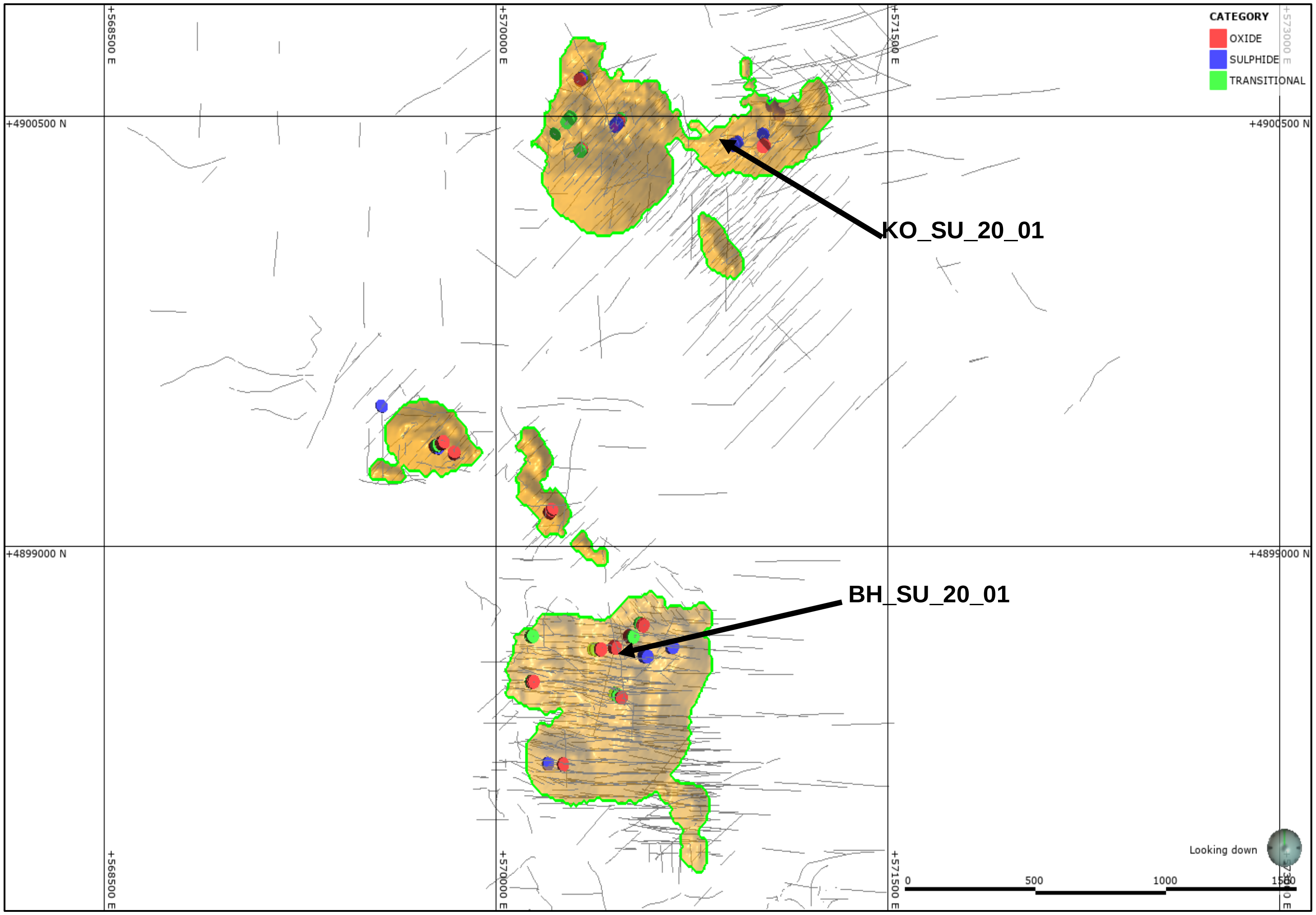
Two (2) sulphide samples were selected for testing; one (1) each from Bigar Hill and Korkan deposits. The same description is shown in Table 13.12.

Table 13.12 – Metallurgical Composites (MRE)

Composite ID	Deposit	Oxidation State	Au (g/t)	S (%)	Description
BH_SU_20_01	Bigar Hill	Sulphide	2.06	1.9	Fresh, S1 and S2 medium/fine grained sandstone and brecciated sandstone from the Bigar Hill. 0.1-0.2% pyrite, typically found in the matrix of the breccia and sandstone grains. S1/S2 Resource Domain.
KO_SU_20_01	Korkan	Sulphide	1.20	1.5	Fresh, coarse grained limestones with intercalated monomict, matrix supported limestone breccia. Hosting disseminated pyrite and pyrite replacement in matrix, with a network of late overprinting carbonate veinlets. 0.5-1% pyrite. S1/SLS Resource Domain.

Figure 13.4 shows the location of all metallurgical samples taken to date (MRE, PEA, and PFS) including the location of the drill holes from which sample intervals were selected to prepare the two (2) sulphide composites.

Figure 13.4 – Location of Sulphide Mineralisation Drill Holes



13.8 Head Assays

13.8.1 Heap Leach Amenability Composite Samples

13.8.1.1 Phase 1 Testwork Composites

a. **Screen Fire Assays**

Screened metallics head assays were performed on each of the composite samples to determine the gold head grade. The analyses were performed on a representative sub-sample of the -2.0 mm material from each sample which had been pulverised to 100% passing 75 µm. Results of the screen fire assays are given in Table 13.13.

Table 13.13 – Screen Fire Analyses (MRE Composites)

Composite I.D.	Calc. Head Grade Au g/t	+106 µm		-106 µm		%Au Distribution	
		Mass %	Au g/t	Mass %	Au g/t	+106 µm	-106 µm
MET18_KO_01	1.44	3.14	0.22	96.9	1.48	0.5	99.5
MET18_KO_02	1.73	3.05	0.32	96.9	1.78	0.6	99.4
MET18_KW_01	1.04	2.94	0.34	97.1	1.07	1.0	99.0
MET18_BH_01	1.86	2.58	0.49	97.4	1.90	0.7	99.3

Results in Table 13.13 indicate that gold values range from 1.04 ppm Au in the KW_01 sample to 1.73 ppm Au in the KO_02 sample. Most of the gold is distributed in the -106 µm size fraction.

b. **Comprehensive Head Assays**

Comprehensive head assays for the four (4) metallurgical composites are shown in Table 13.14.

Table 13.14 – Head Assays (MRE Composites)

Element	Unit	MET18_KO_01	MET18_KO_02	MET18_KW_01	MET18_BH_01
Au	g/t	1.44	1.73	1.04	1.86
Ag	g/t	2	<2	<2	3
As	%	0.008	0.010	0.007	0.016
Hg	g/t	0.9	1.2	1.1	2.8
S_T	%	0.02	0.48	0.01	0.04
$S^{=}$	%	< 0.01	0.41	< 0.01	< 0.01
S^{o}	%	< 0.05	< 0.05	< 0.05	< 0.05
SO_4	%	< 0.1	< 0.1	< 0.1	< 0.1
C_T	%	5.08	5.73	6.58	4.05
Corg	%	< 0.05	< 0.05	< 0.05	< 0.05
TOC	%	< 0.05	0.14	0.06	0.07
CO_3	%	31.3	26.5	41.0	23.3

Table 13.14 shows that silver levels in the composite head samples are low. Whereas sulphide sulphur levels in the transitional sample are higher compared to oxide zone at 0.5%.

13.8.1.2 Phase 2 Testwork Composites

Screened metallics head assays were performed on each of the composite samples to determine the gold head grade. The analyses were performed on a representative sub-sample of the -2.0mm material from each sample which had been pulverised to 100% passing 75 µm.

Results of the screen fire assays are given in Table 13.15.

Table 13.15 – Screen Fire Analyses (PEA Composites)

Composite I.D.	Calc. Head Grade Au g/t	+106 µm		-106 µm		%Au Distribution	
		Mass %	Au g/t	Mass %	Au g/t	+106 µm	-106 µm
BH_P1_01	1.47	1.89	0.44	98.1	1.49	0.6	99.4
BH_P1_02	0.47	1.79	0.26	98.2	0.48	1.0	99.0
BH_P1_03	0.98	2.36	0.68	97.6	0.99	1.6	98.4
BH_P1_04	5.32	2.95	6.65	97.0	5.28	3.7	96.3
KW_P1_01	1.15	2.26	0.52	97.7	1.17	1.0	99.0
KW_P1_02	0.89	2.35	0.27	97.6	0.91	0.7	99.3
KO_P1_01	0.77	1.99	0.46	98.0	0.78	1.2	98.8
KO_P1_02	3.43	2.67	2.41	97.3	3.46	1.9	98.1
BH_P1_01	1.47	1.89	0.44	98.1	1.49	0.6	99.4
BH_P1_02	0.47	1.79	0.26	98.2	0.48	1.0	99.0

Results in Table 13.15 indicate that gold values range from 0.89 ppm Au in the KW_P1_02 sample to 5.32 ppm Au in the BH_P1_04 sample. Most of the gold is distributed in the -106µm size fraction.

a. **Comprehensive Head Assays**

Comprehensive head assays for the metallurgical composites are shown in Table 13.16.

Table 13.16 – Head Assays (PEA Composites)

Element	Unit	BH_P1_01	BH_P1_02	BH_P1_03	BH_P1_04	KW_P1_01	KW_P1_02	KO_P1_01	KO_P1_02
Au	g/t	1.47	0.47	0.98	5.32	1.15	0.89	0.77	3.43
Ag	g/t	<3	<3	<3	<3	<3	<3	<3	<3
As	%	0.8	0.4	0.6	1.4	1.0	0.8	0.6	0.4
Hg	g/t	0.014	0.038	0.020	0.018	0.014	0.006	0.006	0.110
S_T	%	3.4	4.1	6.1	2.9	2.4	0.4	2.0	10.5
$S^=$	%	0.45	0.13	0.40	2.34	0.10	0.04	0.04	1.77

Element	Unit	BH_P1_01	BH_P1_02	BH_P1_03	BH_P1_04	KW_P1_01	KW_P1_02	KO_P1_01	KO_P1_02
S^o	%	0.39	0.11	0.32	2.10	0.09	< 0.05	< 0.05	1.50
SO_4	%								
Corg	%	6.97	5.58	9.87	5.40	7.50	5.50	11.1	5.72

Silver values in the composite head samples was below the detection limit. Sulphide sulphur values range from 0.04% for the oxide composites to 2.4% for the BH_P1_04 sulphide composite.

13.8.1.3 Phase 3 Testwork Composites

a. **Screen Fire Assays**

Screened metallics head assays were performed on each of the composite samples to determine the gold head grade. The analyses were performed on a representative sub-sample of the -2.0mm material from each sample which had been pulverised to 100% passing 75µm.

Results of the screen fire assays are given in Table 13.17.

Table 13.17 – Screen Fire Analyses (PFS Composites)

Composite I.D.	Calc. Head Grade Au g/t	+106 µm		-106 µm		%Au Distribution	
		Mass %	Au g/t	Mass %	Au g/t	+106 µm	-106 µm
BH_PFS_01	1.10	2.85	0.26	97.2	1.1	0.7	99.3
BH_PFS_02	1.58	2.88	0.34	97.1	1.6	0.6	99.4
BH_PFS_03	3.78	2.99	0.25	97.0	3.9	0.2	99.8
BH_PFS_04	0.38	3.01	0.06	97.0	0.4	0.5	99.5
BH_PFS_05	1.57	2.83	0.60	97.2	1.6	1.1	98.9
BH_PFS_06	2.12	2.64	1.13	97.4	2.1	1.4	98.6
BH_PFS_07	1.58	2.43	0.26	97.6	1.6	0.4	99.6
BH_PFS_08	2.00	2.89	0.35	97.1	2.1	0.5	99.5
BH_PFS_09	0.76	2.86	0.08	97.1	0.8	0.3	99.7
KO_PFS_01	0.43	2.78	0.06	97.2	0.4	0.4	99.6
KO_PFS_02	1.63	2.91	0.13	97.1	1.7	0.2	99.8
KO_PFS_03	0.80	2.60	0.08	97.4	0.8	0.3	99.7
KO_PFS_04	0.46	4.58	0.13	95.4	0.5	1.3	98.7
KO_PFS_05	0.62	2.83	0.39	97.2	0.6	1.8	98.2
KO_PFS_06	1.33	2.88	0.42	97.1	1.4	0.9	99.1
KW_PFS_01	1.20	3.00	0.30	97.0	1.2	0.7	99.3
KW_PFS_02	1.04	2.66	0.59	97.3	1.1	1.5	98.5
KW_PFS_03	0.97	2.99	0.14	97.0	1.0	0.4	99.6

Results in Table 13.17 indicate that gold values range from 0.38 ppm Au in the BH_PFS_04 sample to 3.78 ppm Au in the BH_PFS_03 sample. Most of the gold is distributed in the -106µm size fraction.

b. **Comprehensive Head Assays**

Comprehensive head assays for the metallurgical composites are shown in Table 13.18.

Table 13.18 – Head Assays (PFS Composites)

Element	Unit	BH_PFS _01	BH_PFS _02	BH_PFS _03	BH_PFS _04	BH_PFS _05	BH_PFS _06	BH_PFS _07	BH_PFS _08	BH_PFS _09
Au	g/t	1.10	1.58	3.78	0.38	1.57	2.12	1.58	2.00	0.76
Ag	g/t	< 4	< 4	< 4	< 4	< 4	< 4	< 2	< 4	< 4
As	%	0.010	0.011	0.030	0.016	0.002	0.003	0.021	0.007	0.013
Hg	g/t	2.1	1.2	7.9	0.7	0.5	< 0.3	4.8	1.3	1.2
S_T	%	0.03	< 0.01	1.41	0.09	0.18	< 0.01	0.33	0.05	0.38
$S^=$	%	< 0.05	< 0.05	1.17	0.09	0.16	< 0.05	0.29	0.05	0.32
S^o	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
SO_4	%	< 0.1	< 0.1	0.5	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
C_T	%	6.29	3.82	0.97	3.87	9.69	3.15	4.99	6.76	4.50
Corg	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
TOC	%	< 0.05	< 0.05	< 0.05	0.08	0.13	< 0.05	0.08	< 0.05	0.11
CO_3	%	35.1	20.5	4.81	17.6	52.2	14.9	21.4	37.4	23.3
Element	**Unit**	**KO_PFS _01**	**KO_PFS _02**	**KO_PFS _03**	**KO_PFS _04**	**KO_PFS _05**	**KO_PFS _06**	**KW_PFS _01**	**KW_PFS _02**	**KW_PFS _03**
Au	g/t	< 4	< 4	< 4	< 4	< 4	< 4	< 2	< 4	< 4
Ag	g/t	0.010	0.017	0.004	0.008	0.006	0.005	0.009	0.010	0.004
As	%	1.7	1.0	1.5	0.4	1.3	0.9	1.2	0.9	0.7
Hg	g/t	0.07	0.04	0.18	0.02	< 0.01	< 0.01	< 0.01	< 0.01	0.02
S_T	%	0.05	< 0.05	0.18	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
$S^=$	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
S^o	%	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
SO_4	%	6.47	0.13	5.69	6.32	6.98	8.2	4.95	5.78	7.17
C_T	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
Corg	%	< 0.05	0.06	< 0.05	< 0.05	< 0.05	0.05	< 0.05	< 0.05	< 0.05
TOC	%	35.2	0.52	31.7	28.3	40.0	44.5	22.8	32.3	39.3
CO_3	%	0.010	0.017	0.004	0.008	0.006	0.005	0.009	0.010	0.004

Silver values in the composite head samples was below the detection limit. Sulphide sulphur values range from <0.05% for the oxide composites to 1.17% for the BH_PFS_03 transitional composite.

13.8.2 FLOTATION COMPOSITE SAMPLES

13.8.2.1 Transitional Zone Composites

Comprehensive head assays for the two (2) transitional composites tested at SGS Lakefield for flotation evaluation are shown in Table 13.19.

Table 13.19 – Head Assays (Transitional Zone Composites)

Element	Unit	BH_PFS_09	KO_P1_01
Au	g/t	0.81	0.65
Ag	g/t	0.8	<0.5
Cu	%	< 0.01	0.013
Zn	%t	< 0.01	< 0.01
S_T	%	0.015	0.01
$S^=$	%	0.41	0.05

Table 13.19 shows that silver levels in the transitional composite head samples range from 0.5 to 0.8 g/t. Sulphide sulphur levels in the transitional composites range from 0.05% to 0.41%.

13.8.2.2 Sulphide Zone Composites

Comprehensive head assays for the two transitional composites tested at XPS for flotation evaluation are shown in Table 13.20.

Table 13.20 shows sulphur levels in the sulphide composites range from 1.62% to 1.84%.

Table 13.20 – Head Assays (Sulphide Zone Composites)

Element	Unit	BH_PFS_09	KO_P1_01
Au	g/t	2.35	1.26
Ag	g/t	-	-
Cu	%	<0.005	<0.005
Zn	%t	0.02	<0.01
S_T	%	1.84	1.62
$S^=$	%	-	

13.9 Mineralogical Characterisation

13.9.1 Oxide / Transitional Zones

a. **Bigar Hill Mineralisation**

Native gold (0.5-5 µm) hosted in:

- Pyrite;
- Goethite after pyrite;
- Goethite soakage from pyrite weathering;
- Calcite in E W trending fault.

b. **Korkan Mineralisation**

- Native gold (2-3 µm);
- Hosted in calcite (vein/pervasive soaking).

c. **Korkan West – Bigar Hill Mineralisation**

- Native gold (0.5-4 µm);
- Hosted in goethite (pseudomorphs after pyrite);
- A model on the Eh pH evolution responsible for Au precipitation:

  1. Pyrite with invisible gold precipitates;
  2. As the oxidation potential increases pyrite minerals are oxidised to goethite. Because of this oxidation process thiosulphate anions are released, resulting in nanoscale leaching of gold from the goethite mineral into solution.
  3. Eh still increases (being only at weakly oxidative) Au thiosulfate complexes become unstable and gold is precipitated as micrometer sized native gold in Fe oxides/ oxyhydroxides phase.

Figure 13.5 shows the Eh-pH model for evolution of gold precipitation.

Figure 13.5 – Eh/pH Diagram

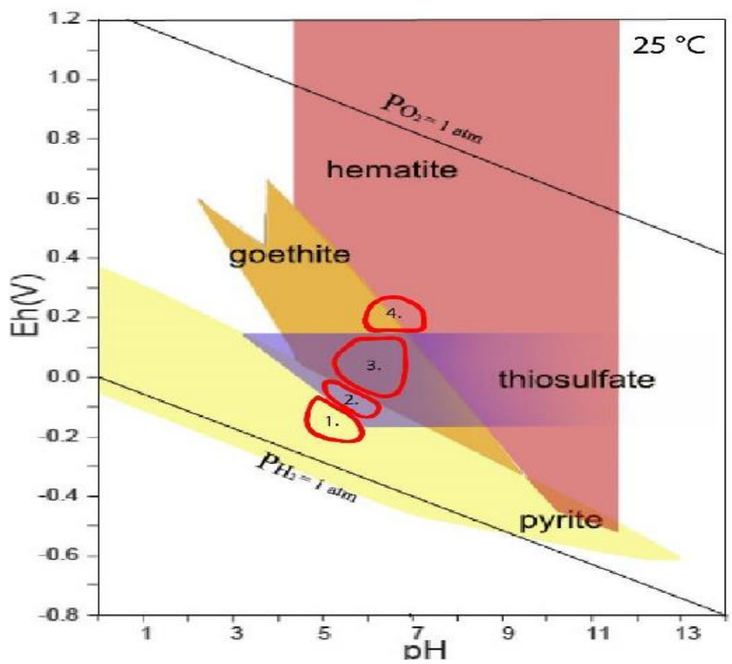
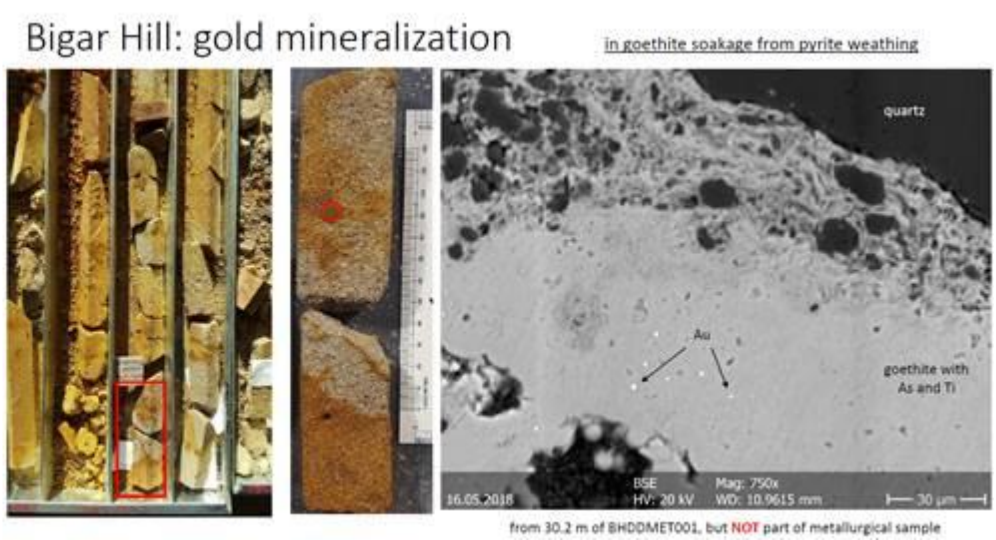


Figure 13.6 – Gold Mineralisation



13.9.2 SULPHIDE ZONE

Examined lithologies include sedimentary sandstones, siltstones, limestones, breccias of these rocks, and a few samples of andesite, monzonite, and BBX fault gouge.

The major observation is that, regardless of the original lithology or mineralogy, alteration minerals consistently include chlorite, biotite, lesser muscovite, and secondary quartz, and chlorite/biotite/muscovite, always, enclose the sulfide ore minerals. Native Au was not observed.

- Fluid/ rock reaction dissolved and/or replaced primary minerals & liberated Fe.
- Fluids sulphidized available Fe forming alteration minerals and Au bearing pyrite.
- As facilitates incorporation of Au in pyrite yielding ~consistent pyrite As/Au ratios.
- SEM analyses of host rock calcite and clays detected trace to minor Fe (and other metal cations available to form pyrite.
- SEM analyses determined that "dark to pale green chlorite" is chamosite, clinochlore, & kaolinite dickite.
- Pyrite, marcasite, and arsenopyrite all contain Au; pyrite is the common host.
- Pyrite cores and high relief rims were observed, but both may contain Au.
- Pyrite framboids and both euhedral and anhedral pyrite may or may not contain Au.
- Proximity to ore fluid appears more important than pyrite form.
- As the ore mineral appears to be Au bearing pyrite rather than Au, the ore fluid did not have to saturate Au and pyrite precipitation was key.
- Intense fluid rock reaction that liberated and sulfidized Fe produced high grade ore.

13.10 Comminution Characterisation

Wardell Armstrong International (WAI) was commissioned by DPM to undertake a programme of comminution testing on two (2) samples of sulphide mineralisation from the Project.

Each sample was subjected to a suite of testing consisting of:

- Bond Low Energy Impact Testing (CWi); A measure of crushability.
- SMC Testing (SMC); A measure of the competency of the ore from a SAG/AG milling perspective.
- Bond Abrasion Index Testing (Ai); a measure of abrasiveness.
- Bond Rod Mill Work Index (RWi); Standard traditional test used for the assessment of ball milling hardness.
- Bond Ball Mill Work Index (BWi). Standard traditional test used for the assessment of ball milling hardness.

The salient data from these tests is presented in Table 13.21.

Table 13.21 – Comminution Testwork Summary

Deposit	Unit	Bigar Hill	Korkan
Sample ID		BH_SU_20_02	KO_SU_20_02
Ore s.g.	t/m^3	2.58	2.57
Bond Impact Work Index	kWh/t	17.2	12.97
SMC Test			
Dwi	kWh/m^3		
A		64	60.3
b		2.26	1.27
A x b		144.64	76.581
ta		1.47	0.73
SCSE	kWh/t	6.13	7.53
Bond Abrasion Index	g	0.3075	0.1646
Bond Rod Mill Work Index	kWh/t	10.16	12.38
Bond Ball Mill Work Index	kWh/t	14.84	12.75

The testwork results available indicate:

- Bond Low Energy Impact testing showed Crusher Work Index values of 12.97 kWh/t and 17.20 kWh/t for the Korkan sulphide and Bigar Hill sulphide samples respectively. When characterised using standard classification criteria, the Korkan sulphide sample was shown to be "medium" with respect to crushability whilst the Bigar Hill sulphide was shown to be "difficult";
- SMC testing reported A*b values of 76.6 for the Korkan Sulphide and 144.6 for the Bigar Hill Sulphide whilst SAG Circuit Specific Energy (SCSE) values ranged from 6.13 kWh/t for the Bigar Hill Sulphide to 7.53 kWh/t for the Korkan Sulphide. When compared against values within the JK Tech database, both samples were shown to be relatively "soft" with both SCSE values within 17.4% of the lowest results;
- Bond Abrasion Index testing produced Abrasion Index values of 0.1646 for the Korkan Sulphide and 0.3075 for the Bigar Hill Sulphide. Using standard classification criteria, both samples were characterised as being "slightly abrasive";
- Bond Rod Mill Work Index values ranged from 10.16 kWh/t for the Bigar Hill Sulphide to 12.38 kWh/t for the Korkan Sulphide with both samples considered "medium" based on standard classification criteria; and
- Bond Ball Mill Work Index testing gave Work Index values of 12.75 kWh/t for the Korkan Sulphide and 14.84 kWh/t for the Bigar Hill Sulphide. When again classified using standard criteria, this showed the Korkan Sulphide material to be "medium" with respect to fine ore grindability whilst the Bigar Hill Sulphide was identified as being "hard".

13.11 Gravity Tests

In order to determine the degree of Gravity Recoverable Gold (GRG) 2-stage bulk gravity tests were undertaken using a 3" Knelson concentrator. The two (2) grind sizes selected were 212 µm and 75 µm.

Results of the gravity tests for the Bigar Hill sulphide composite are shown in Table 13.22.

Table 13.22 – Gravity Test Results (Bigar Hill)

Description	Ore Mass (g)	Au Assay (g/t)	Au Recovery (%)
Starting Mass (g)	20,000	2.71	
Mass Removed from Stage 2 Con (g)	157.11	23.55	6.84%
Mass in Stage 2 Tailings (g)	19,842.89	2.54	93.16%
Mass Removed from Stage 3 Con (g)	98.98	18.18	3.32%
Mass in Stage 3 Tailings (g)	19,743.91	2.57	93.75%
Mass of Con #2 + Con #3	256.09	21.48	10.16%

Major observations:

- The GRG content was low at 10.16%;
- The gold in tailings is distributed preferentially in the fine size fractions;
- The gold that was recovered is also distributed towards finer sizes;
- The gold grade of the concentrate significantly increases at the finer mesh sizes, for both the concentrate and tailings.

Results of the gravity tests for the Korkan sulphide composite are shown in Table 13.23.

Table 13.23 – Gravity Test Results (Korkan)

Description	Ore Mass (g)	Au Assay (g/t)	Au Recovery (%)
Starting Mass (g)	20,000	1.24	
Mass Removed from Stage 2 Con (g)	147.61	3.19	1.89%
Mass in Stage 2 Tailings (g)	19,852.39	1.23	98.11%
Mass Removed from Stage 3 Con (g)	57.1	1.95	0.45%
Mass in Stage 3 Tailings (g)	19,795.29	1.22	97.30%
Mass of Con #2 + Con #3	204.71	2.84	2.34%

Major observations:

- GRG content was very low at 2.34%;
- The gold in tailings is well distributed amongst each size fraction;

- The gold that was recovered is distributed towards coarse size fractions;
- Grades increase at the finer fractions with a lower mass recovery.

The bulk gravity results show that there is limited GRG, and thus does not warrant including a gravity circuit in the flowsheet for treating the sulphide ore types.

13.12 Heap Leach Amenability Tests

13.12.1 Phase 1 Testwork

13.12.1.1 Coarse Bottle Roll Leach Tests

Coarse sample bottle roll testing was conducted to identify the maximum gold recovery achievable from each of the samples at crush sizes typical of conventional heap leach operations.

A series of tests were performed to investigate the effect of crush size on leach performance at a fixed cyanide concentration of 0.5 g/L. Coarse sample bottle roll leach tests were carried out for a leach duration of fourteen days; various crush sizes (P_{100} -50 mm, -16 mm and -6.3 mm) were tested.

A summary of the gold recovery achieved during the coarse sample bottle roll test programme are given in Table 13.24.

Gold extraction from testing conducted on the different master composites at the optimum crush size of 100% -16mm; after 14 days of leaching were:

- Korkan – oxide ore: 93.2%;
- Korkan - transitional ore: 53.1%;
- Bigar Hill - oxide ore: 93.7%;
- Korkan West – oxide ore: 75.5%.

Results did show a slight improvement in gold extraction between a crush size of 100% -50mm, and that achieved at 100% -16mm. There was no real increase in gold extraction arising from crushing to the finer crush size of 100% -6.3mm.

Table 13.24 – Summary of Coarse Sample Bottle Roll Test Results

Composite ID	Deposit	Ore Zone	Ore Crush Size (mm)	COBR Test No.	Reagent Consumption kg/t of CN Feed		Au % Extraction (CN) Hours / Days								CN Residue Au g/t	Head Au, g/t	
					NaCN	CaO	4 h	1 d	2 d	5 d	7 d	9 d	12 d	14 d		CN Calc	CN Direct
MET18_KO-01	Korkan	Oxide	-6.35	1	0.51	0.76	76.7	86.6	88.9	85.6	94.2	90.0	89.5	95.6	0.07	1.60	1.44
			-16	2	0.93	0.70	65.9	81.2	85.3	84.2	88.7	90.6	83.1	93.2	0.11	1.62	
			-25	3	0.90	0.70	53.7	75.7	78.0	79.1	91.2	86.8	85.8	93.4	0.10	1.52	
MET18_BH-01	Bigar Hill	Oxide	-6.35	4	0.27	1.14	54.6	79.7	79.7	80.9	91.3	88.9	88.2	93.6	0.11	1.65	1.86
			-16	5	0.92	0.98	46.8	78.3	87.1	81.0	95.4	91.2	88.5	93.7	0.11	1.75	
			-25	6	0.95	0.92	32.2	66.3	82.0	82.7	83.9	89.6	86.5	93.9	0.11	1.72	
MET18_KW-01	Korkan West	Oxide	-6.35	7	0.46	0.71	41.2	61.6	68.4	69.6	76.3	73.1	72.5	75.9	0.26	1.08	1.04
			-16	8	0.68	0.62	30.2	56.3	68.2	69.3	73.4	76.7	77.7	75.5	0.26	1.06	
			-25	9	0.71	0.67	24.0	51.0	65.0	66.7	69.5	68.0	69.9	74.1	0.28	1.08	
MET18_KO-02	Korkan	Transitional	-6.35	10	0.32	0.71	37.7	46.0	51.5	42.7	51.0	50.3	51.1	55.1	0.80	1.77	1.73
			-16	11	0.86	0.77	30.5	42.3	47.9	44.5	49.8	51.3	50.4	53.1	0.71	1.51	
			-25	12	0.98	0.83	11.6	43.7	48.6	45.5	46.8	50.9	53.7	54.3	0.73	1.60	

13.12.1.2 Column Leach Tests

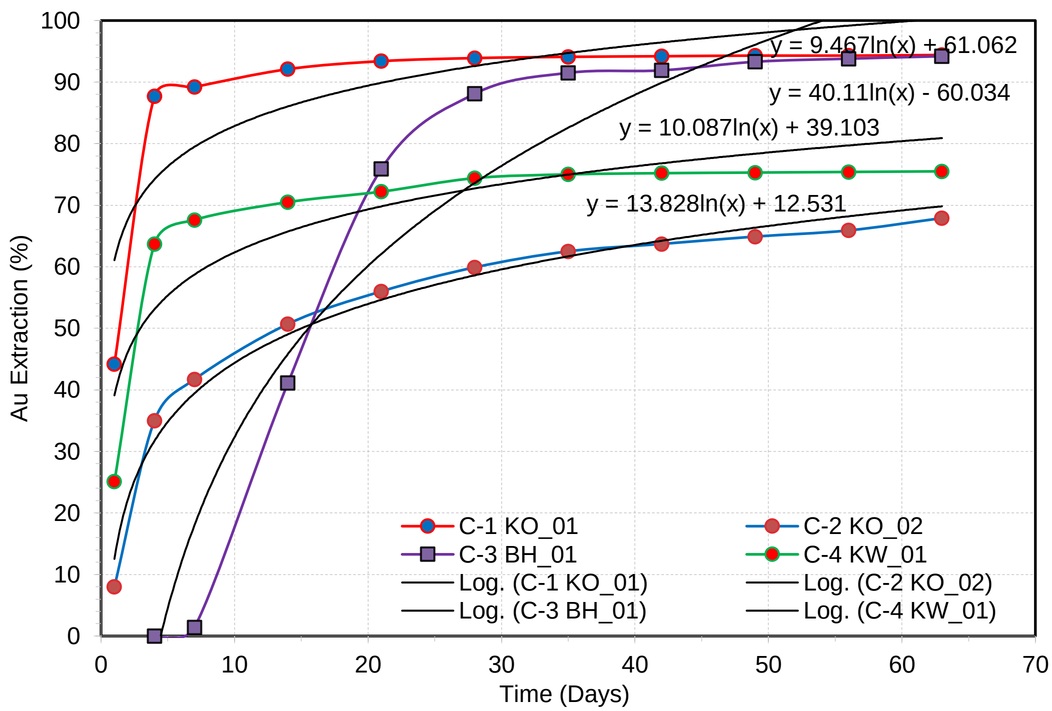
Column leach tests were undertaken to provide confirmation of the achievable metal recoveries and leach rates from each of the samples under heap leaching conditions. A total of four (4) tests were conducted, one on each of the master composite samples at a crush size of 80% -12.5 mm.

Samples were leached for a total of 63 days, using a 0.5 g/L cyanide solution at a target solution application rate of 10 L/m²/hr. The pregnant leach solution was passed through activated carbon to adsorb the gold. The activated carbon was changed on days 1, 4 and 7, and weekly thereafter.

The results of the column leach tests are summarised in Table 13.25. The results show good correlation between the gold extraction based on back calculated head, and those based on carbon assays and solids leach residue.

The column leach curves for the four (4) master composites are shown in Figure 13.7.

Figure 13.7 – Column Leach Kinetics



The test results showed gold recoveries to carbon ranged from 67.9% (KO-02), to 94.4% (KO-01).

Analysis of the leach kinetics during the tests showed the gold to be fast leaching with between 92.1% (KO-01) and 70.5% (KW-01) of the total gold recovered within the first fourteen days of leaching. Gold leach kinetics for the transitional ore zone were slower, and leaching was still ongoing after 63 days of leaching.

During column tests lime consumption was moderate, ranging from 0.88 to 1.21 kg/t whilst the consumption of cyanide was low, ranging from 0.21 to 0.36kg/t.

A comparison of column leach versus coarse bottle roll leach test results are summarised in Table 13.26. Results show a good correlation between column and coarse bottle gold extractions; except for the Korkan transitional ore sample.

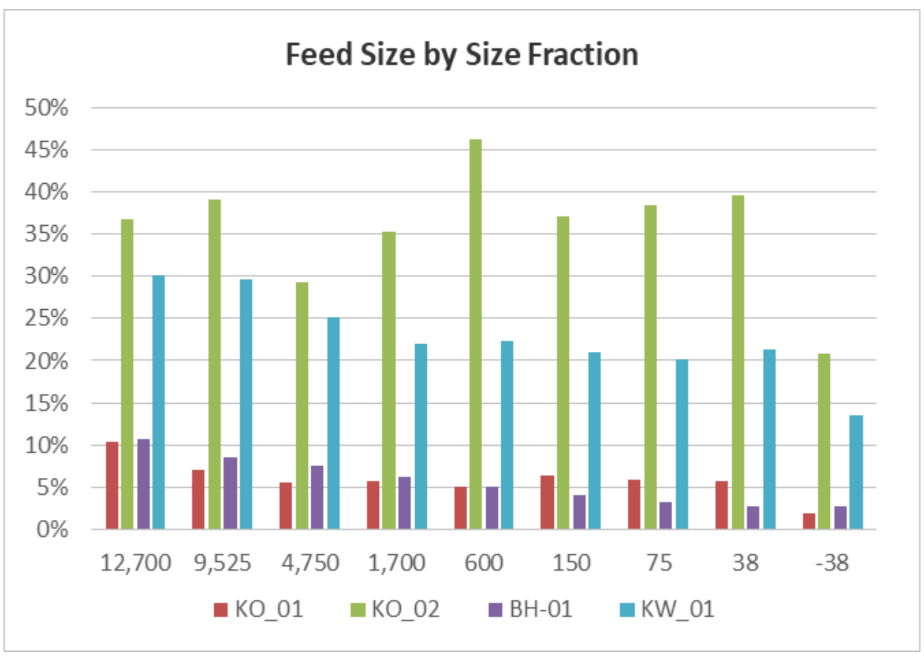
As discussed in Section 13.7.1.1 the coarse bottle roll leach tests were still leaching when the test was terminated after 14 days of leaching.

13.12.1.3 Size by Size Analysis

Sub-samples of both the column leach feed and leach residues from the four samples were submitted for size-by-size analysis for gold to determine the distribution of metal within each sample and to allow metal recoveries by size to be calculated.

The gold distribution in the leach residue as a function of size is graphically represented in Figure 13.8.

Figure 13.8 – Size by Size Analysis



Size by size recovery curves show a decrease in gold extraction in the coarse size fractions (+19.0mm). This suggests that a finer crush size of 100% -12.5 mm could result in higher gold leach extractions.

Table 13.25 – Summary of Column Leach Tests Results

Sample ID	Crush Size 80% mm	Agglom. Stage	Head Assay Au g/t	Calculated Head Au g/t	Extracted Grade Au g/t	Tails Grade Au g/t	%Au Leach Recovery (based on)		
							Measured Head	Calculated Head	Carbon/Residue
Korkan Oxide	-12.9	No	1.48	1.54	1.46	0.08	98.6	94.8	94.4
Korkan Transitional	-12.7	No	1.72	1.96	1.34	0.62	77.9	68.4	67.9
Bigar Hill Oxide	-12.3	No	1.87	2.01	1.90	0.11	101.6	94.5	94.2
Korkan West Oxide	-12.6	No	1.11	1.14	0.87	0.27	78.4	76.3	75.5

Table 13.26 – Comparison of Column Leach Test vs. Coarse Bottle Roll Leach Test Results

Sample ID	Ore Type	Leach Time Days	Crush Size mm	Column Leach Recovery (based on)			Leach Time Days	Crush Size mm	Bottle Roll Recovery Au %
				Measured Head	Calculated Head	Carbon/ Residue			
				Au %					
Korkan	Oxide	63	-12.5	98.6	94.8	94.4	14	-16	93.2
	Transitional		-12.5	77.9	76.3	67.9		-16	53.1
Bigar Hill	Oxide		-12.5	101.6	71.6	94.2		-16	93.7
Korkan West	Oxide		-12.5	78.4	70.5	75.5		-16	75.5

13.12.2 Phase 2 Testwork

13.12.2.1 Coarse Bottle Roll Leach Tests

Coarse ore bottle roll tests were conducted to identify the maximum gold recovery achievable from each of the samples, at crush sizes typical of conventional heap leach operations.

A series of tests were performed to investigate the effect of crush size on leach performance at a fixed cyanide concentration of 0.5g/L. Coarse sample bottle roll leach tests were carried out for a leach duration of thirty days; various crush sizes (P100 -50 mm, -25 mm, -12.5 mm, and -6.3 mm) were tested.

A summary of the gold recovery achieved during the coarse sample bottle roll test programs are given in Table 13.27.

The effect of crush size on gold extraction is shown graphically in Figure 13.9.

Figure 13.9 – Crush Size Effect

Results do show that high gold extraction for the oxide ore zone is achievable at a coarse crush size of 100% -50mm, whereas the transitional ore zone requires a finer crush size of 100% -25 mm.

Gold extractions for the sulphide ore zone samples were <10%, signifying this ore zone is not amenable to processing via heap leach technology.

13.12.2.2 Column Leach Tests

Column leach testing was undertaken to provide confirmation of the achievable metal recoveries and leach rates from each of the samples under heap leaching conditions. A total of three columns were carried out, one on each of the master composite samples at a crush size of 80% -12.5 mm.

A 0.5 g/L cyanide solution was used at a target solution application rate of 10 L/m²/hr. The pregnant leach solution was passed through activated carbon to adsorb the gold. The activated carbon was changed on days 1, 4 and 7, and weekly thereafter. Oxide ore samples (BH_P1_01) were leached for a total of 70 days, whilst the transitional ore sample (KO_P1_01) was leached to a maximum of 91 days.

The results of the column leach tests are summarised in Table 13.28. The results show good correlation between the gold extraction based on back calculated head, and that based on carbon assays and solids leach residue.

A comparison of column leach versus coarse bottle roll leach test results are summarised in Table13.29. Results show a good correlation between column and coarse bottle gold extractions.

Table 13.27 – Summary of Coarse Sample Bottle Roll Test Results

Composite ID	Ore Zone	Test #	Crush Size mm	Reagent Consumption kg/t of CN Feed		% Gold Extraction Days										COBR Tailing Au g/t	Head Grade Au g/t COBR	
				NaCN	CaO	1/4	1	2	5	7	9	15	21	26	30		Calc.	Direct
BH_P1_01	Transitional	COBR-1	-16	1.21	0.84	32	41	43	47	51	52	56	55	58	59	0.59	1.44	1.47
BH_P1_02	Oxide	COBR-2	-16	1.30	1.60	44	63	67	76	80	78	87	80	86	89	0.06	0.56	0.47
BH_P1_03	Transitional	COBR-3	-16	1.23	0.96	30	44	49	53	57	57	61	60	62	64	0.37	1.02	0.98
		COBR-4	-38	1.73	0.91	24	36	37	41	44	45	50	46	51	54	0.33	0.71	
		COBR-5	-6.35	0.68	1.17	32	45	49	52	56	58	63	60	64	62	0.38	1.01	
BH_P1_04	Sulphide	COBR-6	-16	1.97	1.61	3	4	4	5	6	7	7	8	8	7	4.78	5.16	5.32
KW_P1_01	Oxide	COBR-7	-16	1.30	0.60	45	62	67	71	74	78	89	88	90	87	0.15	1.12	1.15
		COBR-8	-38	1.75	0.83	44	62	68	72	77	80	89	84	86	85	0.19	1.24	
		COBR-9	-6.35	0.56	0.89	49	60	65	66	68	76	84	81	88	86	0.17	1.20	
KW_P1_02	Oxide	COBR-10	-16	1.14	0.44	45	64	71	71	72	77	117	83	94	82	0.16	0.89	0.89
		COBR-11	-38	1.53	0.26	33	56	68	68	75	79	57	88	86	81	0.21	1.08	
		COBR-12	-50	1.69	0.01	32	60	70	74	73	73	84	76	75	83	0.16	0.92	
KO_P1_01	Oxide	COBR-13	-16	0.47	0.90	50	66	73	72	77	77	84	82	84	82	0.14	0.74	0.77
		COBR-14	-38	1.32	-0.48	42	60	68	69	72	67	80	78	78	79	0.16	0.76	
		COBR-15	-6.35	0.46	0.88	45	61	66	67	67	66	81	78	82	82	0.14	0.78	
KO_P1_02	Sulphide	COBR-16	-16	1.21	1.46	2	2	3	3	4	5	5	5	7	6	3.19	3.39	3.43
		COBR-17	-38	2.51	0.84	2	2	2	2	3	5	6	7	8	8	3.98	4.31	
		COBR-18	-6.35	1.19	1.46	2	2	3	3	4	5	5	5	7	6	3.15	3.34	

Table 13.28 – Summary of Column Leach Test Results

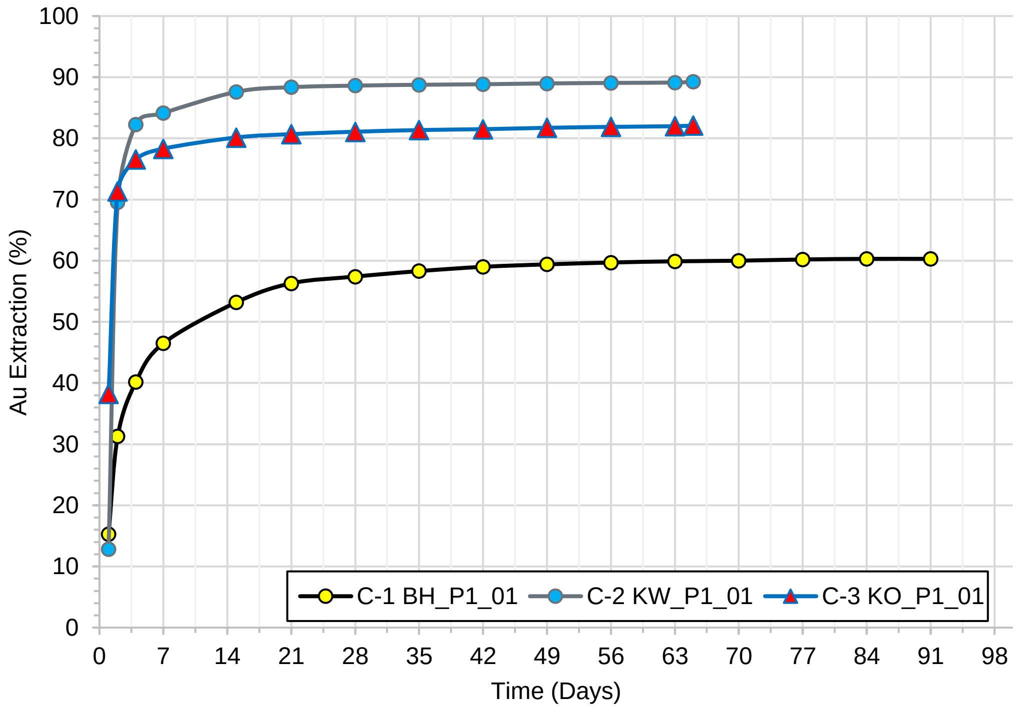
Sample ID	Crush Size 80% mm	Agglom. Stage	Head Assay Au g/t	Calculated Head Au g/t	Extracted Grade Au g/t	Tails Grade Au g/t	%Au Leach Recovery (based on)		
							Measured Head	Calculated Head	Carbon/Residue
Bigar Hill Transitional	-11.3	No	1.35	1.36	0.83	0.53	61.5	61.0	60.3
Korkan West Oxide	-12.3	No	1.15	1.15	1.04	0.11	90.4	90.4	89.3
Korkan Oxide	-12.3	No	0.77	0.80	0.66	0.14	85.7	82.5	82.1

Table13.29 – Comparison of Column Leach Test vs. Coarse Bottle Roll Leach Test Results

Sample ID	Ore Type	Leach Time Days	Crush Size P_{80} mm	Column Leach Recovery (based on)			Leach Time Days	Crush Size mm	Bottle Roll Recovery Au %
				Measured Head	Calculated Head	Carbon/ Residue			
				Au %					
Bigar Hill	Transitional	91	-11.3	61.5	61.0	60.3	30	-5/8	59
Korkan West	Oxide	65	-12.3	90.4	90.4	89.3		-5/8	87
Korkan			-12.3	85.7	82.5	82.1		-5/8	82

Column leach kinetics are shown in graphically in Figure 13.10. The gold leach kinetics for the oxide samples are quite rapid with approximately 80% of the gold leached in the first 4 days. The transitional sample shows less rapid leach kinetics and leaching plateaus out after 60 days, with a gold recovery of 60%, compared with 85-90% for oxide material.

Figure 13.10 – Column Leach Kinetics



13.12.3 PHASE 3 TESTWORK

The Phase 3 testwork program was carried out to further support the heap leach design requirements for the PFS. Column leach testing was undertaken to provide confirmation of the achievable metal recoveries and leach rates from each of the samples under heap leaching conditions.

A total of twenty-three (23) column tests were conducted investigating the following parameters:

- Oxidation state;
- Head grade;
- Crush size;
- Agglomeration.

A 0.5 g/L cyanide solution was used at a target solution application rate of 10 L/m^2/hr. The pregnant leach solution was passed through activated carbon to adsorb the gold. The activated carbon was changed on days 1, 4 and 7, and weekly thereafter. Oxide ore samples were leached for a total of 84 days, whilst transitional ore samples were leached to a maximum of 112 days.

The results of the column leach tests are summarised in Table 13.31 (oxide samples) and Table 13.32 (transitional samples). The results show good correlation between the gold extraction based on back calculated head, and that based on carbon assays and solids leach residue.

A comparison of column leach versus coarse bottle roll leach test results are summarised in Table 13.33. Results show reasonably good correlation between column and coarse bottle gold extractions.

Column leach kinetics for the oxide and transitional samples, at the optimum crush size of -25mm, are shown in graphically in Figures 13.11 and 13.12. The gold leach kinetics for the oxide samples are quite rapid with approximately >80% of the gold leached after 14 days. The transitional sample shows less rapid leach kinetics with >60% leached after 20 days.

Average reagent consumptions for the 24 composites are shown in Table 13.30.

Table 13.30 – Reagent Consumptions

Ore Zone	Reagent Consumption, kg/t	
	NaCN	Lime
Oxide	0.30	0.70
Transitional	0.52	0.81

The average cyanide and lime consumptions are considered to be low for both ore zones, compared to other heap leach operations.

Some of the columns (KW_PFS_01 and KO_PFS_03) demonstrated percolation issues and required agglomeration with cement. The composites exhibited a high percentage of clay in the column feed material.

Table 13.31 – Summary of Column Leach Test Results (Oxide)

Column ID	Sample ID	Deposit	Ore Zone	Crush Size P_{80} mm	Agglom Stage	Head Assay Au g/t	Calc. Head Au g/t	Extracted Grade Au g/t	Tails Grade Au g/t	%Au Leach Recovery (based on)		
										Measured Head	Calc. Head	Carbon/ Residue
C-4	BH_PFS_01 (-1")	Bigar Hill	Oxide	21.7	No	1.20	1.28	1.18	0.10	98.3	92.2	91.9
C-5	BH_PFS_02 (-1")			20.0	No	1.50	1.68	1.51	0.17	100.7	89.9	89.7
C-7	BH_PFS_04 (-1")			19.8	No	0.41	0.48	0.38	0.10	92.7	79.2	79.4
C-9	BH_PFS_06 (-1")			19.4	No	2.21	2.20	1.81	0.39	81.9	82.3	82.2
C-11	BH_PFS_08 (-1")			21.8	No	1.55	1.61	1.49	0.12	96.1	92.5	92.3
C-13	KO_PFS_01 (-1")	Korkan	Oxide	22.0	No	0.40	0.41	0.33	0.08	82.5	80.5	79.8
C-14	KO_PFS_02 (-1")			18.7	No	1.51	1.71	1.60	0.11	106.0	93.6	93.4
C-16	KO_PFS_04 (-1")			20.0	No	0.46	0.47	0.43	0.04	93.5	91.5	92.0
C-17	KO_PFS_05 (-1")			20.6	No	0.50	0.66	0.55	0.11	110.0	83.3	83.6
C-18	KO_PFS_06 (-1")			21.0	No	1.25	1.33	1.23	0.10	98.4	92.5	92.4
C-19	KW_PFS_01 (-1")	Korkan West	Oxide	20.9	Yes	1.18	0.99	0.80	0.19	67.8	80.8	81.2
C-20	KW_PFS_02 (-1")			20.4	No	1.06	1.16	1.01	0.15	95.3	87.1	87.2
C-21	KW_PFS_03 (-1")			21.0	No	0.94	1.04	0.91	0.13	96.8	87.5	87.9
C-22	BH_PFS_02 (-1.5")	Bigar Hill	Oxide	30.3	No	1.56	2.05	1.88	0.17	120.5	91.7	91.8
C-24	KW_PFS_01 (-1.5")	Korkan West	Oxide	31.2	Yes	1.1	1.14	0.86	0.28	78.2	75.4	75.6

Table 13.32 – Summary of Column Leach Tests (Transitional Samples)

Column ID	Sample ID	Deposit	Ore Zone	Crush Size P_{80} mm	Agglom Stage	Head Assay Au g/t	Calc. Head Au g/t	Extracted Grade Au g/t	Tails Grade Au g/t	%Au Leach Recovery (based on)		
										Measured Head	Calc. Head	Carbon/ Residue
C-6	BH_PFS_03 (-1")	Bigar Hill	Trans	19.8	Yes	3.91	3.74	3.28	0.46	83.9	87.7	87.7
C-8	BH_PFS_05 (-1")			21.6	No	1.72	1.87	1.17	0.70	68.0	62.6	62.8
C-10	BH_PFS_07 (-1")			19.5	No	1.46	1.61	1.14	0.47	78.1	70.8	70.8
C-12	BH_PFS_09 (-1")			20.9	No	0.83	0.80	0.47	0.33	56.6	58.8	59.1
C-15	KO_PFS_03 (-1")	Korkan	Trans	19.8	No	0.73	0.93	0.74	0.19	101.4	79.6	79.6
C-23	KO_PFS_03 (-1.5")			30.9	Yes	0.80	0.60	0.37	0.23	46.3	61.7	61.8

Table 13.33 – Comparison of Column Leach Test vs. Coarse Bottle Roll Leach Test Results

Sample ID	Method	Coarse Bottle Roll Leach			Column Leach				
	Ore Type	Leach Time Days	Crush Size mm	Bottle Roll Recovery %Au	Leach Time Days	Crush Size P_{80} mm	Column Leach Recovery, %Au (based on)		
							Assay Head	Calc. Head	Carbon/ Residue
BH_P1_01	Transitional	30	-16	59	91	11.3	61.5	61.0	60.3
KW_P1_01	Oxide			87	63	12.6	90.4	90.4	89.3
KO_P1_01	Oxide			82		12.6	85.7	82.5	82.1
BH_PFS_03	Transitional		-25	80	112	19.8	83.9	87.7	87.7
BH_PFS_05				62		21.6	68.0	62.6	62.8
BH_PFS_07				71		19.5	78.1	70.8	70.8
BH_PFS_09				60		20.9	56.6	58.8	59.1
KO_PFS_03				74		19.8	101.4	79.6	79.6

Figure 13.11 – Column Leach Kinetics (Oxide Samples)

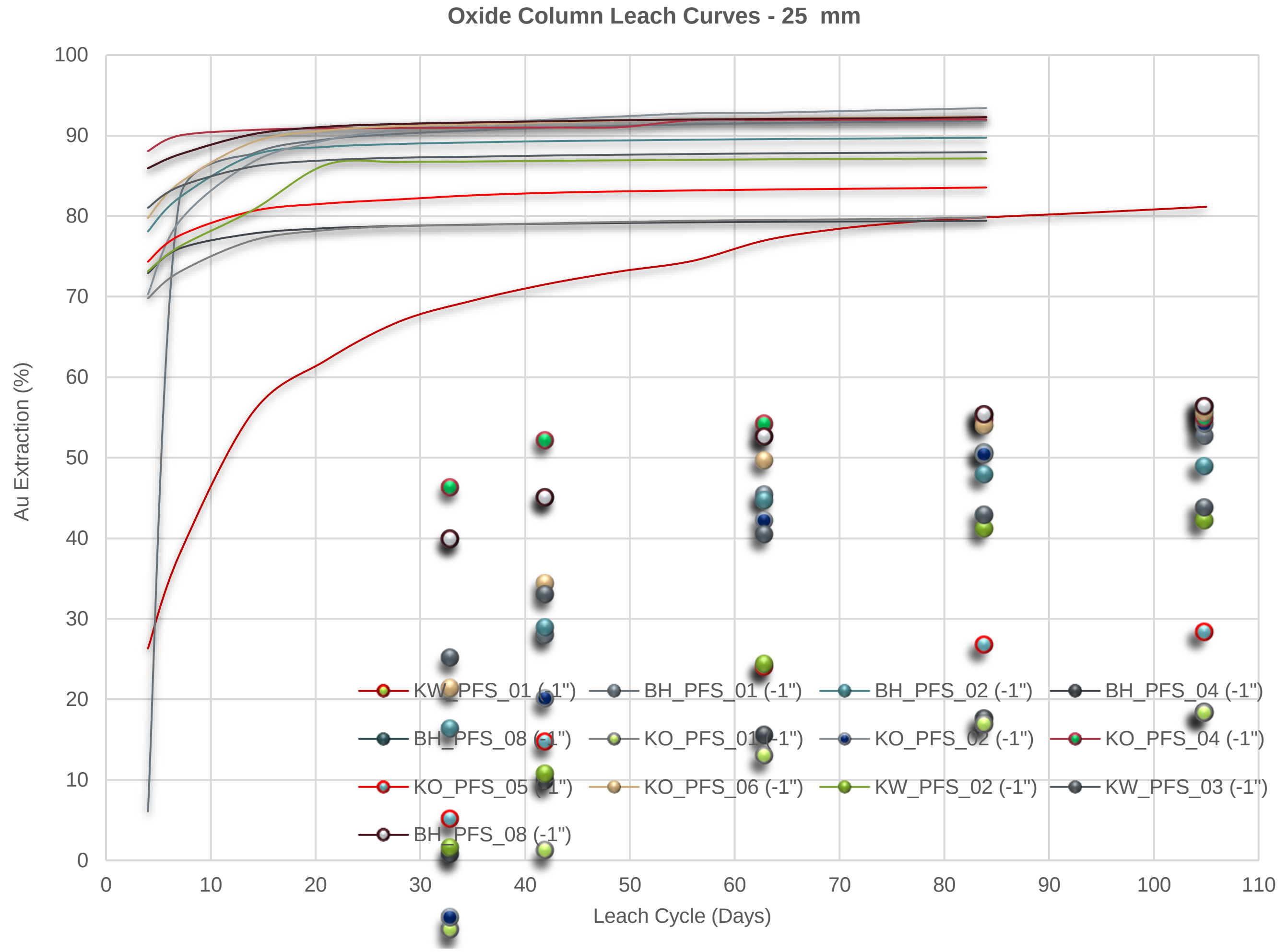
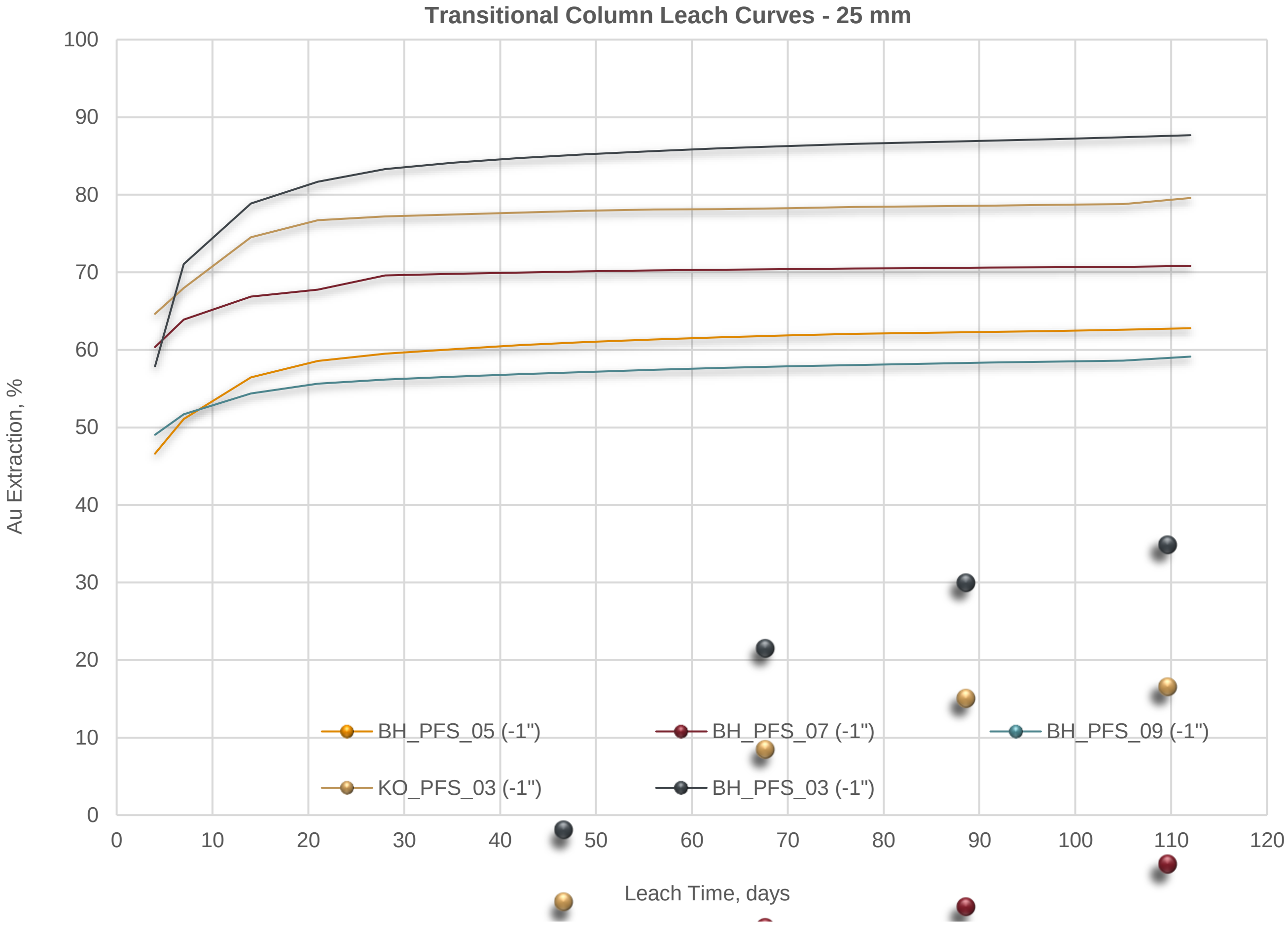


Figure 13.12 – Column Leach Kinetics (Transitional Samples)



13.12.3.1 Effect of Crush Size

a. **Oxide Zone**

The effect of crush size on gold leach extraction for oxide ore zone samples tested are shown in Tables 13.34 to 13.36.

Table 13.34 – Gold Extraction (- 16 mm)

Testwork Phase	Column ID	Composite ID	Deposit	Oxidn. State	Leach Cycle days	%Au Extn
1	C-1	KO_01 (-5/8")	Korkan	Oxide	63	94.4
	C-3	BH_01 (-5/8")	Bigar Hill		63	94.2
	C-4	KW_01 (-5/8")	Korkan West		63	75.5
2	C-2	KW_P1_01 (-5/8")	Korkan West		70	89.3
	C-3	KO_P1_01 (-5/8")	Korkan		70	82.1
		Average			**66**	**87.1**

Table 13.35 – Gold Extraction (- 25 mm)

Testwork Phase	Column ID	Composite ID	Deposit	Oxidn. State	Leach Cycle days	%Au Extn
3	C-4	BH_PFS_01 (-1")	Bigar Hill	Oxide	84	91.9
	C-5	BH_PFS_02 (-1")			84	89.7
	C-7	BH_PFS_04 (-1")			84	79.4
	C-9	BH_PFS_06 (-1")			84	82.2
	C-11	BH_PFS_08 (-1")			84	92.3
	C-13	KO_PFS_01 (-1")	Korkan		84	79.8
	C-14	KO_PFS_02 (-1")			84	93.4
	C-16	KO_PFS_04 (-1")			84	92.0
	C-17	KO_PFS_05 (-1")			84	83.6
	C-18	KO_PFS_06 (-1")			84	92.4
	C-19	KW_PFS_01 (-1")	Korkan West	Oxide	105	81.2
	C-20	KW_PFS_02 (-1")			84	87.2
	C-21	KW_PFS_03 (-1")			84	87.9
		Average			**86**	**87.2**

Table 13.36 – Gold Extraction (- 38 mm)

Testwork Phase	Column ID	Composite ID	Deposit	Oxidn. State	Leach Cycle days	%Au Extn
3	C-22	BH_PFS_02 (-1.5")	Bigar Hill	Oxide	84	91.8
	C-24	KW_PFS_01 (-1.5")	Korkan West	Oxide	105	75.6
		Average			**95**	**83.7**

Gold leach extraction for the oxide zone appears to be independent of crush size over the range of crush sizes tested.

Based on the results of column leach tests carried out at different grind sizes an optimum crush size of 100% passing 25mm has been selected as optimum for the oxide ore zone.

b. **Transitional Zone**

The effect of crush size on gold leach extraction for transitional zone samples tested are shown in Tables 13.37 to 13.39.

Table 13.37 – Gold Extraction (- 16 mm)

Testwork Phase	Column ID	Composite ID	Deposit	Oxidn. State	Leach Cycle days	%Au Extn
1	C-2	KO_02 (-16 mm)	Korkan	Trans	63	67.9
2	C-1	BH_P1_01 (-16 mm)	Bigar Hill	Trans	91	60.3
		Average			**77**	**64.1**

Table 13.38 – Gold Extraction (- 25 mm)

Testwork Phase	Column ID	Composite ID	Deposit	Oxidn. State	Leach Cycle days	%Au Extn
3	C-6	BH_PFS_03 (-25 mm)	Bigar Hill	Trans	112	87.7
	C-8	BH_PFS_05 (-25 mm)				62.8
	C-10	BH_PFS_07 (-25 mm)				70.8
	C-12	BH_PFS_09 (-25 mm)				59.1
	C-15	KO_PFS_03 (-25 mm)	Korkan			79.6
		Average			**112**	**72.0**

Table 13.39 – Gold Extraction (- 38 mm)

Testwork Phase	Column ID	Composite ID	Deposit	Oxidn. State	Leach Cycle days	%Au Extn
3	C-23	KO_PFS_03 (-38 mm)	Korkan	Trans	112	**61.8**

Gold leach extraction is less dependent on crush size; a decrease in gold recovery was obtained at a coarse crush size of -38mm.

Based on the results of column leach tests carried out at different grind sizes an optimum crush size of 100% passing 25mm has been selected as optimum for the transitional ore zone.

13.12.3.2 Head Grade Effect

a. **Oxide Zone**

The relationship between head grade and gold leach extraction, as a function of crush size, is shown in Figures 13.13 and 13.14.

Figure 13.13 – Head Grade vs. %Au Extraction (16 mm)

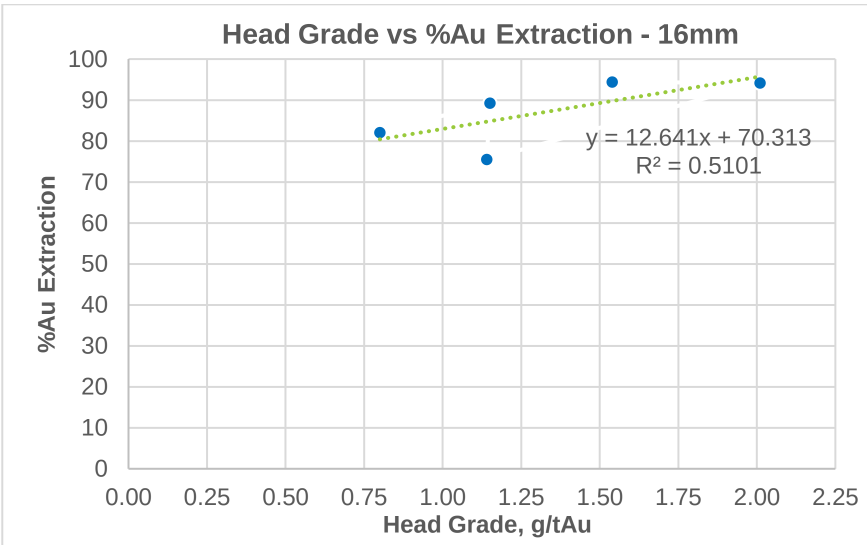
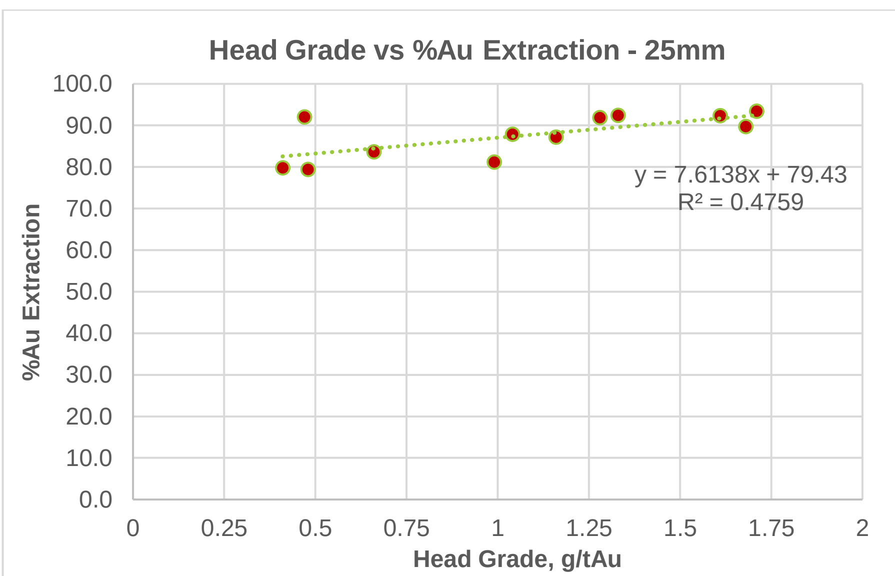


Figure 13.14 – Head Grade vs. %Au Extraction (25 mm)



Figures 13.13 and 13.14 show that there is a direct relationship between head grade and gold leach extraction for the oxide zone samples; crushed at -16 mm and -25 mm.

b. **Transitional Zone**

Figure 13.15 – Head Grade vs. %Au Extraction (25 mm)

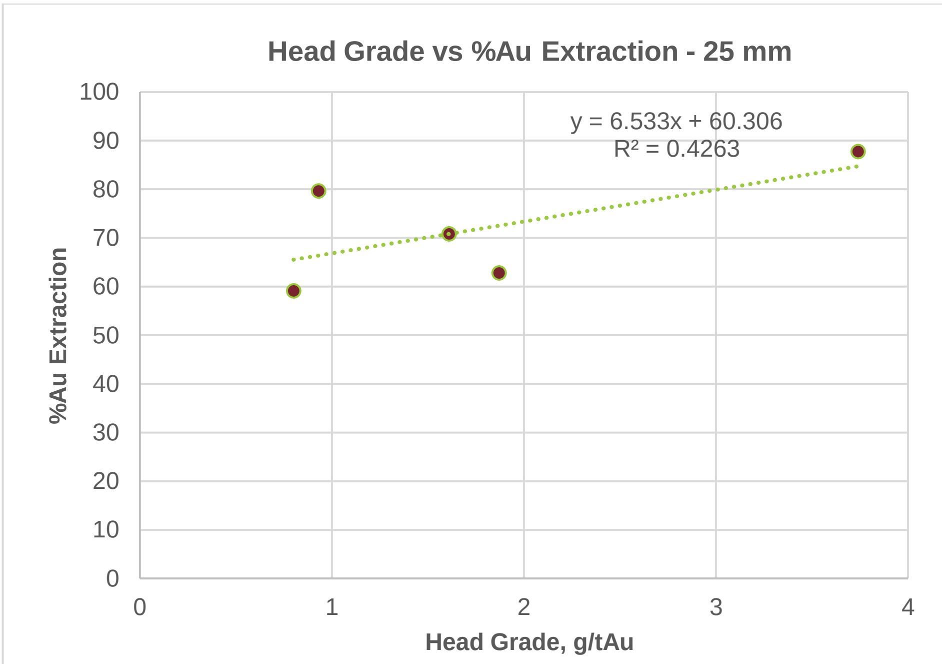


Figure 13.15 show that there appears to be a relationship between head grade and gold leach extraction for the transitional zone samples is not as strong as that exhibited for the oxide zone samples.

Gold leach extraction for the transitional zone is more related to the %TS content.

13.13 Flotation

13.13.1 Transitional Composites

In order to evaluate the metallurgical performance of flotation on transitional composites representing the Bigar Hill and Korkan deposits rougher kinetic, open cycle cleaner, and locked cycle tests were carried out.

13.13.1.1 Rougher Kinetic Tests

Rougher kinetic tests were carried out at a target P_{80} of 120 µm, for a float residence time of 30 min. Optimal grind size and flotation times were derived from testing undertaken on the sulphide composites. Alternative promoter reagents were investigated. Results of rougher kinetic tests are shown in Table 13.40.

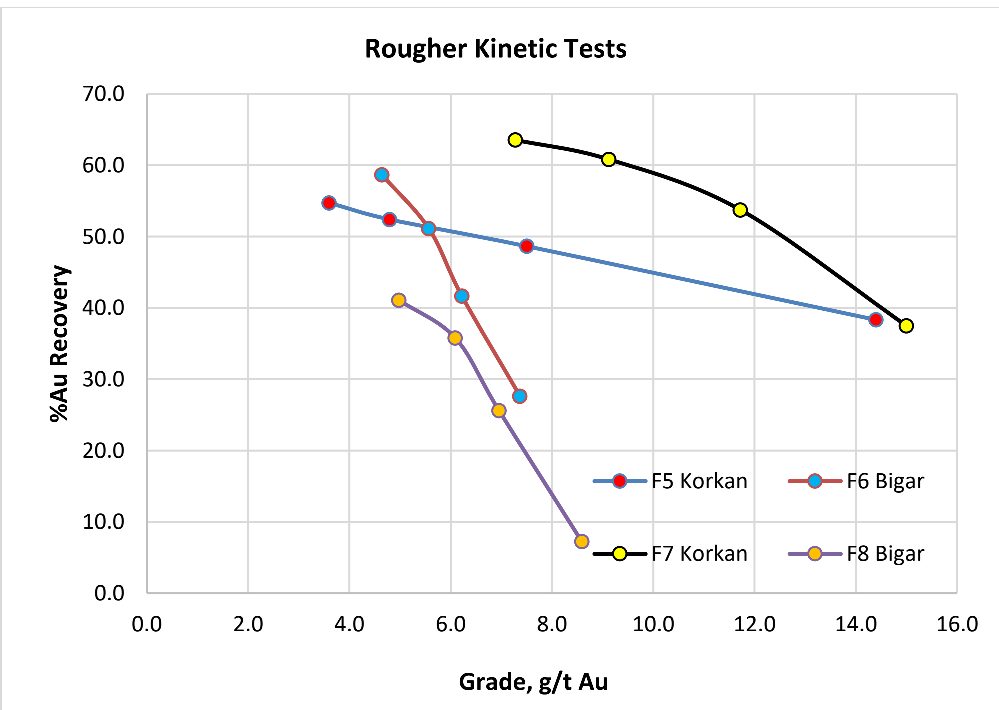
Table 13.40 – Kinetic Rougher Test Results

Float Test	Comp ID	Grind Size P_{80} µm	Collector Type	Mass Pull %Wt.	Grade		Recovery	
					Au g/t	%TS	Au	%TS
F5	KO_P1_01	126	AF 208	9.84	3.60	0.48	54.73	83.93
F6	BH_PFS_09	127	AF 208	10.42	4.64	3.41	58.69	79.87
F7	KO_P1_01	121	AF 404	8.04	7.28	1.21	63.55	91.33
F8	BH_PFS_09	128	AF 404	5.12	4.98	2.77	41.09	65.13

Rougher results show that gold and total sulphur recoveries to the rougher concentrate ranged from 41 to 64%, and 65 to 91% respectively. Rougher grade-recovery curves are shown in Figure 13.16.

Flotation performance for Korkan was better compared to that obtained for Bigar Hill; using different collector types.

Figure 13.16 – Rougher Test Results



13.13.1.2 *Open Cycle Cleaner Tests*

Open cycle cleaner tests were carried out adopting a single stage of cleaning and a regrind stage. The target regrind size for the rougher/scavenger concentrate was a P_{80} of 20µm.

Results of the open cycle cleaner tests are shown in Table 13.41.

Table 13.41 – Open Cycle Cleaner Test Results

Float Test	Comp ID	Grind Size	Regrind	Mass Pull %Wt.	Grade		Recovery	
		P_{80} µm			Au g/t	%TS	Au	%TS
F9	BH_PFS_09	120	20	1.80	15.8	16.4	36.7	70.8
F10	KO_P1_01	120	20	0.27	71.9	10.6	33.6	55.7
F11	KO_P1_01	120	20	0.83	26.9	3.9	35.0	48.6

Results of the open cycle cleaner tests show that overall gold recovery to the cleaner concentrate is quite low at around 35%, at a concentrate grade ranging from 16 to 72g/t Au. The minimum gold grade for a saleable pyrite concentrate is 20 g/t.

13.13.1.3 Locked Cycle Tests

Locked cycle tests were carried out on each of the transitional composites adopting the optimal test conditions derived from the open cycle cleaner tests. Results of the locked cycle tests are summarised in Table 13.42.

Table 13.42 – Locked Cycle Cleaner Tests

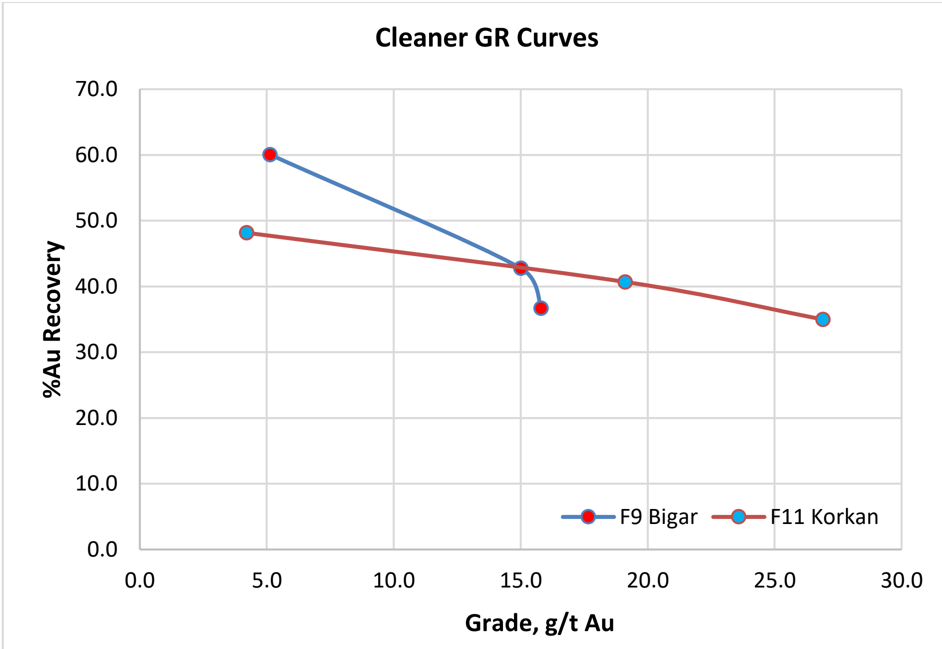
Composite ID	Test ID	Product	Mass Pull Wt. %	Assays, %, g/t		% Distribution	
				Au	S	Au	S
BH_PFS_09	LCT 1	1st Cl Conc	2.5	10.9	11.4	32.8	70.4
KO_P1_01	LCT 2	1st Cl Conc	1.2	20.4	2.8	38.7	73.3

Locked cycle test results show that gold recovery to the cleaner concentrate was low ranging between 33 and 39%, at cleaner concentrate grades ranging from 11 to 20.4 g/t Au. This indicates that not all of the gold in the feed is associated with sulphides.

The cleaner concentrate grade for the Bigar Hill composite does not meet the required specification for a saleable concentrate, whilst the final concentrate grade for the Korkan composite is right on the limit of being acceptable.

Cleaner grade-recovery curves are shown in Figure 13.17.

Figure 13.17 – Open Cycle Cleaner Grade-Recovery Curves



13.13.2 Sulphide Composites

Additional testwork to support the base case process option developed from the 2019 PEA was carried out at XPS on composites representing the sulphide zone. Two (2) composite samples (60 kg each) representing the sulphide zones at Bigar Hill (BH_SU_20_01) and Korkan (KO_SU_20_01) deposits were submitted to XPS for testing.

The objective of the testwork program was to optimise the rougher flotation conditions for the Bigar Hill and Korkan deposits in terms of primary grind size, reagent type, and reagent quantity. Mineralogical support is included for size by size analysis of flotation tailings.

After optimisation of rougher conditions open cycle cleaner tests were executed to optimise the regrind size and chemical conditions, and to demonstrate the effect of multiple cleaning stages in open circuit. This was subsequently followed by locked cycle testing to quantify closed circuit grade and recovery expectations.

The combined float tail was subjected to cyanidation leaching to determine the degree of cyanide extractable gold associated with non-sulphides.

13.13.3 Rougher Tests

Rougher optimisation tests were carried out on the Bigar Hill composite evaluating different grind sizes, and reagent suites to determine the optimum grind size, and reagents to carry forward into the open cycle cleaner tests.

Results of the rougher optimisation tests are shown in Table 13.43.

Table 13.43 – Batch Rougher Test Results

Test ID	Deposit	Circuit Config.	Grind Size P_{80} µm	Float Time min	Reagent Suite	Rougher/Scavenger Concentrate			
						% Mass Pull	Grade (ppm)	Recovery, %	
							Au	Au	S
FT001	Bigar Hill	Ro/Sc	150	45	Standard	20.8	6.84	56.1	85.5
FT002		Ro/Sc	100	45	Standard	22.7	6.25	54.5	80.5
FT003		Ro/Sc	75	45	Standard	26.8	5.36	56.3	81.3
FT004		Ro/Sc	53	45	Standard	31.3	5.12	59.1	84.1
FT005		Ro/Sc	38	45	Standard	34.8	4.75	64.2	85.5
FT006		Ro/Sc	150	45	Reagent Dosage	23.6	6.47	58.8	83.9
FT007		Ro/Sc	150	55	Reagent Dosage	26.6	6.19	63.5	83.2
FT008		Ro/Sc	150	55	Reagent Dosage	26.3	6.54	65.0	83.1
FT 009		Ro/Sc	53	79	Standard	31.7	5.00	63.1	88.8
FT 010		Ro/Sc	38	55	Standard	37.7	3.88	61.8	87.7
FT 011		Ro/Sc	150	55	$CuSO_4$	27.9	5.52	62.8	91.2

Results in Table 13.43 show that the optimal primary grind size for flotation was a P_{80} of 120 µm. This coarse primary grind size promoted selectivity between sulphides (pyrite) and non-sulphide gangue (mainly silicates).

The rougher optimisation tests also showed the requirement to include the addition of copper sulphate in the reagent suite to assist with the activation of tarnished sulphides.

Figure 13.18 and Figure 13.19 clearly show that gold recovery and total sulphur recovery are directly related to concentrate mass pull.

Figure 13.18 – Au Recovery vs. Mass Pull

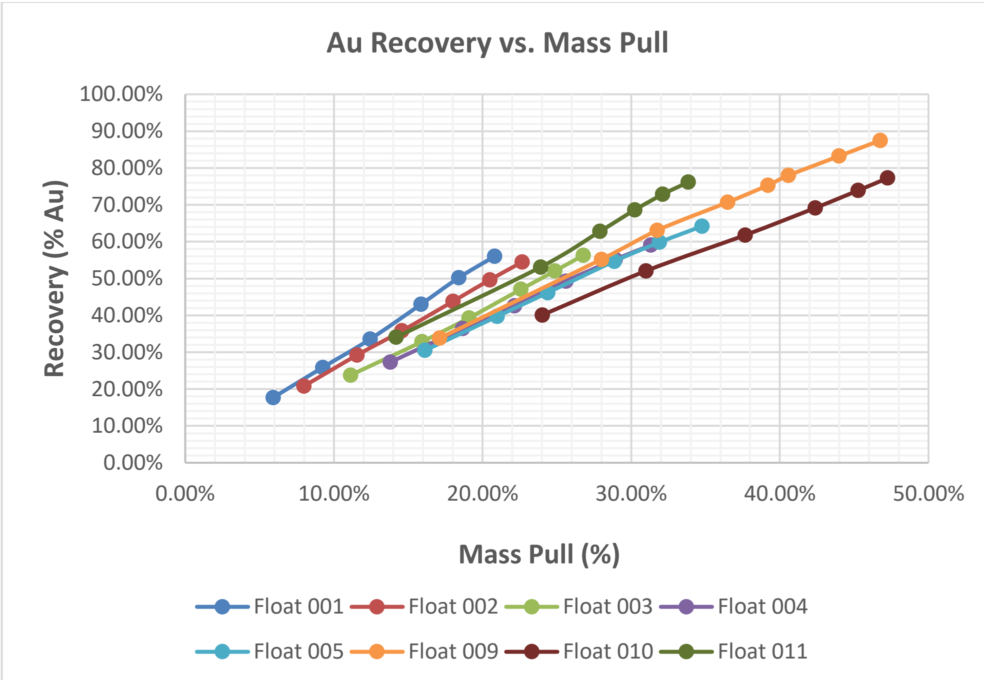
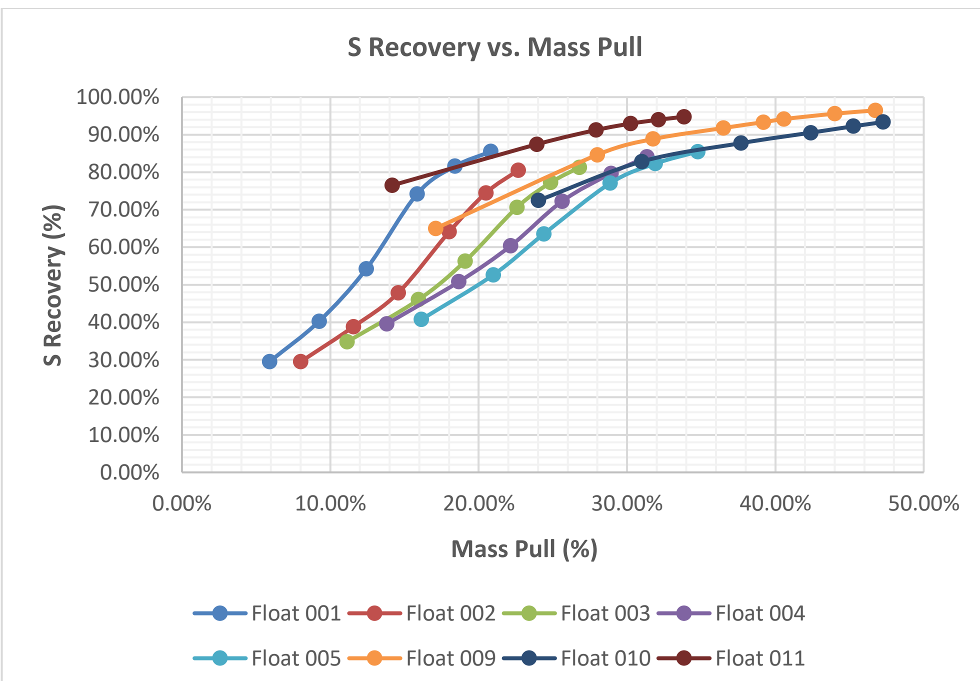
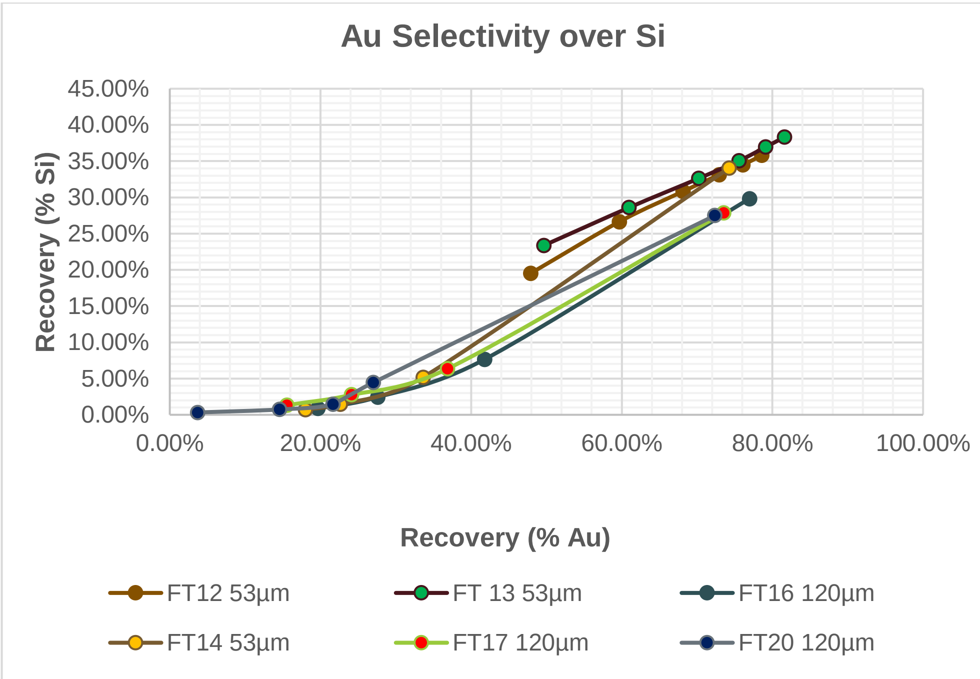


Figure 13.19 – TS Recovery vs. Mass Pull



In order to achieve high gold and sulphur recoveries laboratory float residence times are very long, due to poor selectivity between pyrite and non-sulphide gangue. This is clearly shown in Figure 13.20 with poor selectivity of gold over non-sulphide gangue. As the grind size coarsens, the selectivity of gold and total sulphur over non-sulphide gangue significantly improves.

Figure 13.20 – Gold Selectivity of NSG



13.13.4 Open Cycle Cleaner Tests

A series of open cycle cleaner tests were carried out investigating:

- 1st cleaner scavenger stage;
- Regrind stage;
- Silicate depressants.

Open cycle cleaner grade-recovery curves for Bigar Hill and Korkan composites are shown in Figure 13.21 and Figure 13.22.

Optimum results were achieved for tests carried out with a rougher concentrate regrind stage; P_{80} of 20 µm, and with the addition of EDTA as a gangue depressant.

Figure 13.21 – Open Cycle Cleaner Grade-Recovery Curves (Bigar Hill)

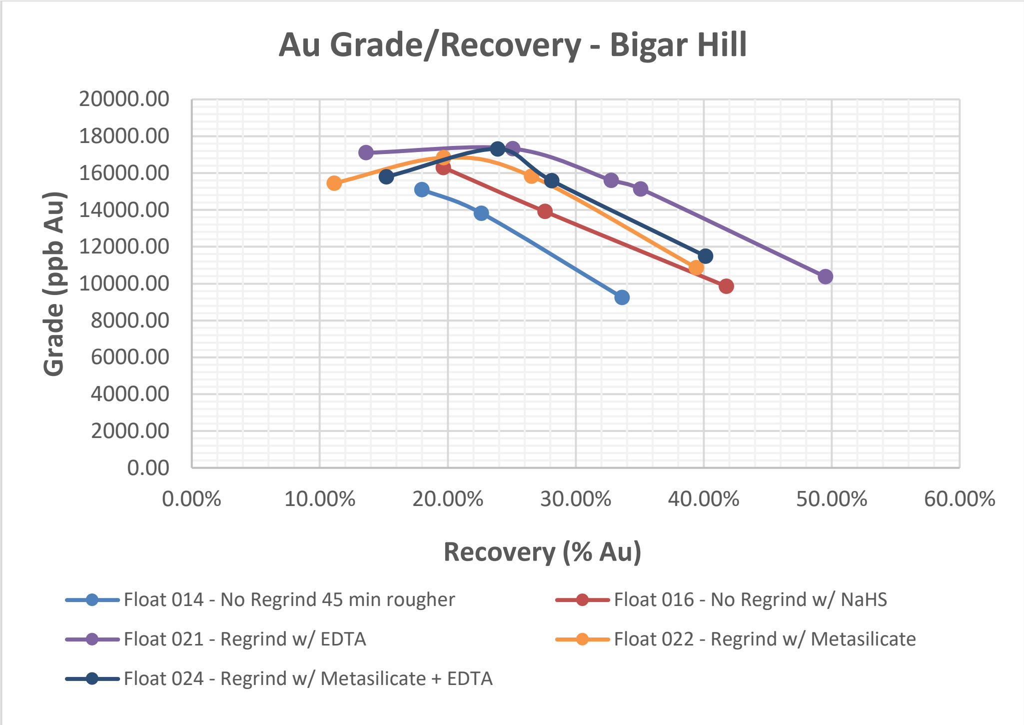
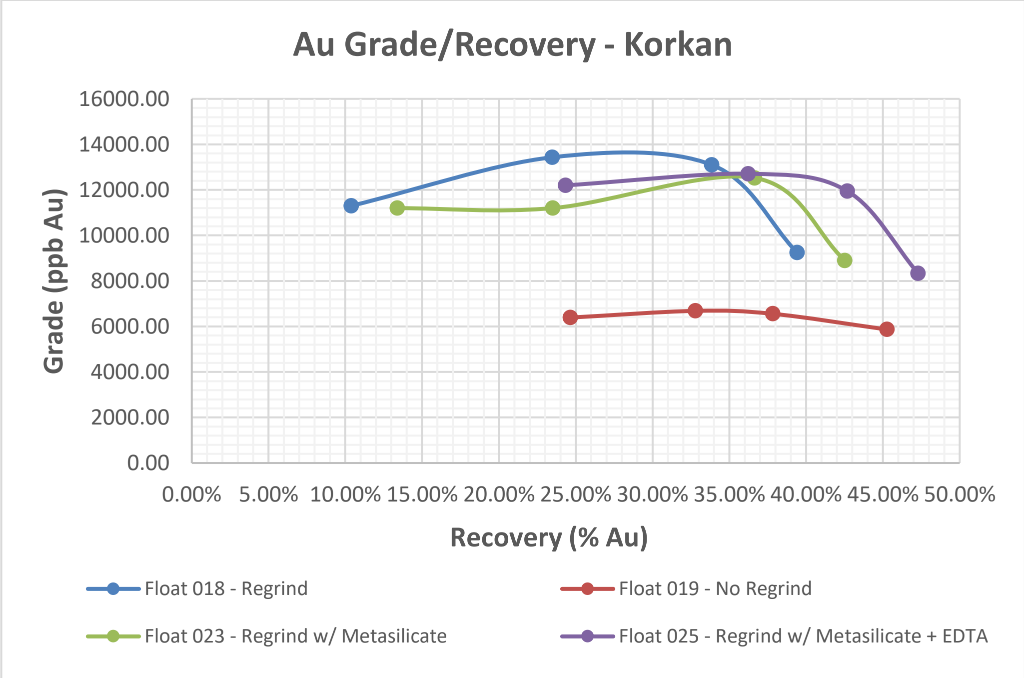


Figure 13.22 – Open Cycle Cleaner Grade-Recovery Curves (Korkan)



13.13.5 LOCKED CYCLE TESTS

Based on optimal test conditions derived from the open cycle cleaner tests a single Locked Cycle Test was carried out on each of the sulphide composites representing the Bigar Hill and Korkan deposits.

Results of the LCT's are summarised in Tables 13.44 and 13.45.

Table 13.44 – Locked Cycle Test Results (Bigar Hill)

Metallurgical Balance Cycles 4-6	% Mass Distn.	Grade (%)					% Distribution				
		Au (ppm)	Fe	Mg	S	Si	Au	Fe	Mg	S	Si
Heads Fresh Feed	**100.00**	2.59	2.27	0.19	1.48	26.47	100.0	100.0	100.0	100.0	100.0
Cleaner Concentrate	**8.79**	**12.46**	13.79	0.23	13.79	16.35	**42.3**	53.3	10.9	82.1	5.4
Cleaner Scavenger Tail	23.69	3.63	2.35	0.27	0.73	23.88	33.2	24.4	34.8	11.8	21.4
Rougher Tailings	67.52	0.94	0.75	0.15	0.13	28.70	24.6	22.3	54.4	6.1	73.2

The final sulphur grade is fairly low for a cleaned pyrite concentrate. The cleaner concentrate still contains a significant amount of silica and talc, despite the use of EDTA as a gangue depressant.

Table 13.45 – Locked Cycle Test Results (Korkan)

Metallurgical Balance Cycles 4-6	% Mass Distn	Grade (%)					% Distribution				
		Au (ppm)	Fe	Mg	S	Si	Au	Fe	Mg	S	Si
Heads Fresh Feed	**100.00**	1.24	1.60	5.88	1.59	7.42	100.0	100.0	100.0	100.0	100.0
Cleaner Concentrate	**5.04**	**10.27**	19.57	3.43	22.81	5.44	**41.8**	61.8	2.9	72.4	3.7
Cleaner Scavenger Tail	18.69	1.31	1.13	6.43	0.80	6.44	19.7	13.2	20.4	9.5	16.2
Rougher Tailings	76.26	0.63	0.52	5.91	0.38	7.79	38.5	25.0	76.6	18.1	80.1

Results of LCT's carried out on the Bigar Hill and Korkan composites are inferior to those obtained on sulphide composites tested by SGS in 2012 and 2013. Both gold recovery and final cleaner concentrate grades are lower. The rougher concentrate grades were significantly lower grade, compared to those achieved in the 2012 work.

It is speculated that the sulphide samples tested at XPS in 2020 have different mineralogical properties compared to those tested by SGS in 2012 and 2013. Mineralogy tends to indicate that the pyrite mineral grain size is finer in the sulphide composite samples tested in 2020, compared to those in 2012/13 testwork phases. An additional geo-metallurgical program is required to obtain a better understanding of variability within the sulphide zones.

13.14 Leaching

13.14.1 CIL Tests (Oxide Composites)

The standard industry lixiviant for extraction of gold from the oxide and transitional ore types is sodium cyanide.

The viability of using alternative lixiviants was investigated; namely thiosulphate and CN Lite.

13.14.1.1 Thiosulphate

Testing of thiosulphate was carried out on oxide composites representing the Bigar Hill and Korkan deposits. Kinetic leach tests were carried out at standard grind size P_{80} of 75µm, for different leach times.

Results of leach tests carried out with thiosulphate are shown in Table 13.46.

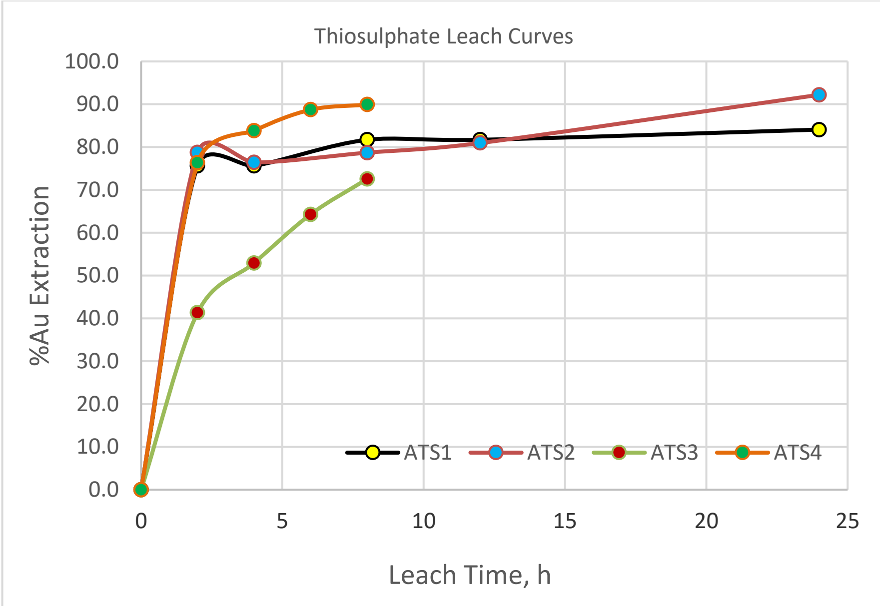
Table 13.46 – Thiosulphate Leach Test Results

Leach Time h	KO-PFS-05 ATS1	BH-PFS-02 ATS2	Leach Time h	KO-PFS-05 ATS3	BH-PFS-02 ATS4
	%Au Extraction			%Au Extraction	
0	0.0	0.0	0	0	0
2	75.6	78.7	2	41	76
4	75.6	76.4	4	53	84
8	81.7	78.7	6	64	89
12	81.7	80.9	8	73	90
24	84.1	92.2			

The kinetic leach curves for the different tests are shown graphically in Figure 13.23.

Results show that high gold extractions were achieved after 24 hours of leaching. The results did show that there is some variability with respect gold extractions achieved under similar test conditions.

Figure 13.23 – Kinetic Leach Curves



13.14.1.2 CNLITE

Leach tests were carried out on oxide composites representing Bigar Hill and Korkan deposits. Comparative leach tests were carried out with CN Lite and NaCN at a standard grind size P_{80} of 75µm, and leach residence time of 48 hours.

In test CIL-3, lab reagent grade NaCN (98% NaCN) was used/added. In test CIL-4, CN Lite (which is 25% NaCN as determined from titration) was used/added. Thus, for CIL-4 and addition rate of 3.57 kg/t of CN Lite was added and given that CN Lite is 25% NaCN, the equivalent addition rate was 0.89 kg/t NaCN. Results of leach tests are summarised in Table 13.47.

Table 13.47 – Kinetic Leach Test Results (Oxide Composites)

Comp ID	Test ID	Lixiviant	%Au Extn.	Reagent Consumption	
				NaCN, kg/t	Lime, kg/t
BH_PFS_02	CIL-3	NaCN	93.3	0.34	0.40
	CIL-4	CNLite	93.8	0.36	0.05
KO_PFS_05	CIL-5	NaCN	89.0	0.23	0.36
	CIL-6	CNLite	90.5	0.26	0.00

Results in Table 13.47 show that gold leach extractions were lower than that achieved with sodium cyanide. Given that CN Lite contains 25% sodium cyanide the reagent consumption with respect to cyanide consumption were effectively the same for the comparative tests.

13.14.2 CIL Tests (Transitional Composites)

The base case process developed from the PEA for treating transitional ores was heap leach technology. Whole ore cyanidation leach tests were carried out to evaluate treating transitional ores using conventional CIL.

Tests were carried out on composites representing the transitional ore from Bigar Hill and Korkan deposits. The CIL tests were carried out at a target grind size P_{80} of 75µm, cyanide addition of 0.5 g/L, and leach residence time of 48 hours.

Results of the CIL tests are shown in Table 13.48.

Table 13.48 – CIL Test Results (Transitional Composites)

Comp ID	Test ID	Grind Size P_{80} µm	Lixiviant	%Au Extn.	Reagent Consumption kg/t	
					NaCN	Lime
BH_PFS_09	CIL-7	92	NaCN	56.7	0.41	0.49
KO_P1_01	CIL-8	41	NaCN	86.2	0.15	0.36

Results show that gold leach extraction was moderate, ranging from 57-86%. It should be noted that the Korkan composite was unintentionally ground finer than the target grind size P_{80} of 75µm. Cyanide and lime consumptions were low to moderate.

Gold extraction, obtained by conventional CIL, would be on par with that obtained for treating the transitional material by heap leach technology. A separate trade-off study would be required to evaluate the economics of both technologies, however based on results to date adopting heap leach technology would likely be most economic as there is only a minor increase in gold recovery using CIL versus heap leaching, and there is a relatively small tonnage of transitional material in the Timok resource.

13.14.3 CIL Tests (Transitional LCT Tails)

To evaluate potential cyanideable gold from the flotation tails CIL tests were carried out on the combined scavenger / 1st cleaner scavenger tails. CIL tests were carried out on as is products, at cyanide concentration of 0.5 g/L and for leach time of 48 hours.

Results of the CIL tests are shown in Table 13.49.

Table 13.49 – CIL Test Results (Combined Float Tails)

Comp ID	Test ID	%Au Extn.	Reagent, kg/t	
			NaCN	Lime
BH_PFS_09	CIL 9	60.1	0.22	0.41
KO_P1_01	CIL 10	74.9	0.21	0.23

Results show that gold leach extractions of 60 to 75% were obtained.

13.14.4 Cyanidation Tests (Sulphide LCT Tails)

The combined scavenger and 1st cleaner scavenger tails were subjected to cyanidation leaching to determine the degree of cyanide soluble gold which could potentially be recovered by a conventional CIL.

Results of the kinetic leach tests are shown in Table 13.50 and 13.51.

Table 13.50 – Kinetic Leach Test Results (Bigar Hill)

Sample	Time (hour)	Au (g/t)	% Au in sol	Ag (g/t)	% Ag in sol	Cu (g/t)	% Cu in sol
Dund 21 Assay 0324	0	0	0.0	0	0.0	0	0.0
	1	0.341	38.2	0.1511	36.2	21.980	48.4
	2	0.336	38.5	0.1547	37.9	23.214	52.2
	4	0.341	39.9	0.1586	39.7	24.276	55.8
	8	0.346	41.4	0.1661	42.3	25.574	59.9
	24	0.365	44.4	0.1839	47.5	29.373	69.5
	48	0.390	48.2	0.2004	52.5	32.606	78.2

Table 13.51 – Kinetic Leach Test Results (Korkan)

Sample	Time (hour)	Au (g/t)	% Au in sol	Ag (g/t)	% Ag in sol	Cu (g/t)	% Cu in sol
Dund 21 Assay 0327	0	0	0.0	0	0.0	0	0.0
	1	0.004	0.8	0.0114	13.4	12.248	32.0
	2	0.003	0.6	0.0109	13.1	13.835	37.1
	4	0.003	0.5	0.0109	13.5	15.130	41.4
	8	0.002	0.5	0.0115	14.4	16.747	46.6
	24	0.003	0.7	0.0146	18.4	21.633	60.6
	48	0.003	0.7	0.0172	21.9	24.771	70.3

Results in Tables 13.50 and 13.51 show that an additional 48% and 0.7% gold can be leached from the combined flotation tails.

13.14.5 Ultra-Fine Grinding-CIL Tests

UFG technology followed by conventional CIL was evaluated on sulphide composites representing the Bigar Hill (BH_P1_04) and Korkan (KO_P1_02) deposits.

The sulphide composites were ground to P_{80} of 75µm, and then floated to produce a gold bearing sulphide concentrate.

Results of the bulk rougher float are shown in Table 13.52.

Table 13.52 – Bulk Rougher Float Results

Test ID	Deposit	Float Circuit	Grind Size P80 µm	Reagent Suite	Mass Pull %Wt.	Gold		Sulphur	
						Grade (ppm)	%Rec	Grade (%)	%Rec
F1	Bigar Hill	Ro-Sc	75	PAX	25.0	8.4	80.4	6.6	93.3
F2	Korkan	Ro-Sc	75	PAX	23.6	15.5	64.7	5.6	76.1

Results show that gold and sulphur recoveries to the bulk rougher concentrate ranged from 65 to 80%, and 76 to 93% respectively. Rougher concentrate mass pull was fairly high at circa 25% by weight.

The bulk rougher concentrate was then finely reground ahead of cyanidation leach stage. The regrind sizes for the Bigar Hill and Korkan rougher concentrates were a P_{80} of 18.4 µm and 9.8 µm respectively. The rougher tail also underwent cyanidation leach, on as is material, to determine the amount of cyanide recoverable gold.

Results of the cyanidation leach tests are shown in Table 13.53.

Table 13.53 – Cyanidation Leach Test Results

Product	BH_P1_04	KO_P1_02
	%Au Extraction	
F1+F3 Ro Conc.	36.3	-
F1+F3 Ro Tails	29.8	-
F2+F4 Ro Conc.	-	75.0
F2+F4 Ro Tails	-	58.5

Gold leach extractions from the rougher concentrate ranged from 36% to 75%, whilst for the tail they ranged from 30% to 59%.

13.14.6 ALBION AMENABILITY TESTS

The Albion Process™ was used to evaluate pre-oxidative technologies on the refractory rougher concentrate. A Neutral Albion Leach (NAL) test was carried out on the reground rougher concentrate.

Test conditions for the NAL tests are shown in Table 13.54.

Table 13.54 – Neutral Albion Leach Test Conditions

Test No.	Sample ID	Grind K80 (µm)	Pre-Treatment	Leach			
				Time (h)	Density (%)	NaCN (g/L)	Carbon (g/L)
NAL-1 CIL	F1+F3 Flot Conc (BH_P1_04)	18.4	Albion	48	25	1	15
NAL-2 CIL	F2+F4 Flot Conc (KO_P1_02)	9.8	Albion	48	25	1	15

The NAL-1 showed a sulphur oxidation of 80%, whilst for NAL-2, a sulphur oxidation of 82% was achieved. The NAL residue was subsequently subjected to cyanidation to determine overall gold leach extraction.

Results of the CIL tests on the NAL residue are shown in Table 13.55.

Table 13.55 – Cyanidation Leach Test Results

Test No	Sample ID	Extraction (% Au)	Residue (g/t Au)	Calc Head (g/t Au)
NAL-1 CIL	F1+F3 Flot Conc (BH_P1_04)	88.0	0.62	5.1
NAL-2 CIL	F2+F4 Flot Conc (KO_P1_02)	95.2	0.63	13.2

Based on head assay (NAL Residue) versus leach tailing assays, NAL-1 CIL gold extraction was ~88%. Whilst for NAL-2 CIL gold extraction was ~95%.

The major outcomes from the completed testwork program are the following:

- Oxide and transitional ore types are both amenable to heap leach, and can both be processed through a single stage irrigation;
- Sulphide ore is not amenable to heap leach and has not demonstrated amenability for a simple hydrometallurgical processing;
- Flotation testwork completed to date was unable to generate salable concentrate and further testwork is required.

The testwork interpretation details can be found below.

13.15 Testwork Interpretation

Interpretation of the testwork results was completed by DRA in order to establish a basis of process design for the heap leach plant and define the requirements for further study of the sulphide resource.

13.15.1 Heap Leach Plant

Testwork on the Timok deposit is considered representative of the ore to be processed according to the PFS mine schedule. Testing was conducted on ores distributed throughout the planned pit shells, with respect to both depth and spatial coverage.

The tests provide suitable data coverage for the primary geological rock types, namely;

- Korkan – oxide ore;
- Korkan - transitional ore;
- Bigar Hill - oxide ore; and
- Korkan West – oxide ore.

The sulphide mineralization type is not included in the mine plan due to the poor leach performance. Other methods for gold recovery were considered for the sulphide mineral resources including flotation for production of concentrate and oxidation of the concentrate through the Albion process followed by cyanidation of the oxidation product and flotation tails, CIL adsorption of the gold, and production of the doré bars as a final product.

The majority of the ore samples tested have head grades which are considered representative of the mineral reserves in the PFS mine schedule, and within an acceptable grade range of the design criteria.

13.15.1.1 Heap Leach Crush Particle Size, Expected Recoveries and Agglomeration Requirement

The column leach test results were used to determine the metallurgical gold recoveries and reagent consumption levels.

An ore crush size recommended for design was selected based on the gold recovery and duration of the leach cycle summarised in Table 13.34 through Table 13.39.

A design crush size P100 of 25 mm is recommended for the all types of ore.

There is an opportunity in evaluation if the coarser crush size (P100 of 38 mm) will be acceptable for all types of ore with regards to the leach recovery. The coarser crush size promises potential benefits in simplification of the crushing area flowsheet and reduction of the operating costs.

Insufficient CWi and Ai tests were conducted on the oxide and transitional ore samples to design the crushing plant flowsheet and size the relevant plant and equipment, in addition to predicting the operational cost associated with crushing-plant wear parts. The PFS crushing circuit design is presently based on DRA design and operational experience. Additional testing of ore composites and variability samples is required to develop a sufficient data set.

Based on industry benchmarking, DRA experience, and outcome from the process design reviews the following discounts were applied to the column tests recovery numbers:

- 2% for oxide ore;
- 5% for transitional ore to cater for the data scatter and a number of data points available to date.

A summary of the expected field gold recovery numbers by ore type can be found in Table 13.56.

Table 13.56 – Expected Gold Recoveries (Field), by Ore Type

Ore Type	Ore Tonnage, Million Tonnes	Percentage of Total Ore to Heap Leach, (%)	Gold Grade, (g/t)	Gold Recovery, (%) 1.
Bigar Hill, Oxide	8.76	45.6	1.19	85.1
Korkan, Oxide	3.36	17.5	0.90	86.2
Korkan West, Oxide	3.71	19.3	0.99	83.4
Bigar Hill, Transitional	1.92	10.0	1.09	68.8
Korkan, Transitional	1.20	6.3	1.02	74.6
Korkan West, Transitional	0.25	1.3	0.74	81.0
Oxide Total	15.8	82	1.08	84.9
Transitional Total	3.4	18	1.04	71.8
Oxide and Transitional Total	**19.2**	**100**	**1.07**	**82.6**

Notes:
Column leach testwork number by ore type at –25.4 mm crush size. Discounted 2% for oxide ore, and 5% for transitional ore
Numbers may not add due to rounding

Currently, no information on the oxide and transitional ore regarding 'preg robbing' or 'preg borrowing' behaviour is available. Providing for the expected clay loaded areas within the deposit, this work should be budgeted for the feasibility study (cyanide shake tests).

Present knowledge of the ore body suggests that agglomeration will be required intermittently on some clay loaded ore domains.

It should be noted that an agglomeration circuit has been included on an as required basis, but a greater understanding on the amount and locations of clay zones in the deposit is required.

The knowledge of the clay speciation presently is limited to the clay types within the deposit, and no clay speciation data is available. Currently, no load permeability tests, nor agglomeration tests results required for definition of the agglomerates strengths and required dosage of lime and cement are available. These tests are planned for the feasibility study testwork program.

With regards to the ore stacking, all types of ore can be stacked together based on the current results of the leach tests. Agglomerated ore should be physically separated from non-agglomerated ore on the leach pad where practical.

13.15.1.2 Heap Leach Reagent Consumption

A review of the column leach test sodium cyanide consumption, at the Timok design solution application volume, gave a cyanide consumption level of 0.40 kg/t. A laboratory-to-field factor of 30% was applied to the column leach test results, which gave an expected field sodium cyanide consumption of 0.12 kg/t.

To ensure proper pH control, the expected quick lime addition maximum is 0.32 kg/t (equivalent to 0.30 kg/t of pure calcium oxide). A quicklime availability of 92% calcium oxide was assumed on the recommendation of the selected supplier.

Cement addition will be required for the agglomeration circuit only, and a consumption rate is presently understood at 10 kg/t based on DRA's experience due to the absence of the testwork data. The cement consumption rate should be revised during the next stage of the Project upon receipt of the testwork results and the necessity to agglomerate. Portland Type II cement will be used for agglomeration and is widely available. This cement is sulfate resistant and is the standard choice for agglomeration cement in heap leach operations.

Preparation of a geo-metallurgical model of the Timok deposit is recommended to optimise reagent consumption and gold recovery for the different ore types.

13.15.1.3 Irrigation Rates and Methodology

Irrigation rates, durations, and application flux rates (m^3 of solution / tonne of ore) were determined from the testwork and DRA experience.

Irrigation rates and cycle times were adjusted to accommodate the heap leach stacking requirements. The PEA irrigation rates were re-evaluated targeting on reduction of the application rates and simplification of the solutions management across the heap leach circuit.

Based on the assessment of the leach curves from 28 column tests for the oxide and transitional ore types, oxide ore gold extraction reaches maximum extraction at approximately 75 days and the transitional ore gold extraction approaches the maximum extraction at approximately 80 days.

Analysis of the leach curves shows the two distinct leaching characteristics of the Timok oxide and transitional ore zones:

- A kinetic regime – the leach rate is controlled by the leach solution volume rate and active cyanide concentration and achieves rapid leaching of gold which is easily accessible to the leaching solution.
- A diffusion regime – the leach rate is controlled by diffusion and the leaching of gold/silver electrum particles present. As the surface material oxidizes at a slow rate, new particle

surfaces are exposed to leaching. Both mechanisms contribute to the slow leach rate for diffusion regime.

For a conventional heap leach project, a two-stage irrigation system with an intermediate solution tank could be implemented to reduce irrigation volume. However, the heap leach for Timok is designed as a valley fill due to layout and terrain related constraints. Separation of pregnant and intermediate solutions, though not impossible, would be difficult and would negate the benefits of the two-stage irrigation schedule. Based on the results of the evaluations, a shorter single-stage irrigation schedule is more practical than the two-stage irrigation. The single stage irrigation will maintain the simplicity of the process and will ultimately help to minimise solution management challenges during the heap leach plant operation.

The applicability of a shorter leach cycle was confirmed by numerical analysis of the gold extraction rate increase. The models were based on the work completed and published in a number of papers (Marsden and Botz, 2017 and 2019).

The modelling was applied to both oxide and transitional ore types to project a leach recovery rate through the leach cycle duration as a function of the slow and fast leachable reaction rate.

The results were plotted against the leach cycle duration days and aligned to the experimental data using the sum of squares to determine the accuracy of the curve fit.

The CLT results were sorted and grouped based on ore type and the deposit. The maximum gold extraction achieved rate for each test is used to normalize the datasets for modelling. The average normalized gold extraction rate is calculated for each leaching interval for each deposit and ore type. The oxide ore type reaches a weighted average recovery of 87% (lab - undiscounted) and transitional ore achieves 77.5% (lab - undiscounted) respectively.

By the ninetieth day, the leaching of most types of ore is complete with exclusion of the Korkan transitional ore. This ore type represents approximately 6% of total ore tonnage amenable for the heap leach and cannot be considered as a major driver for the leach cycle time selection.

A comparison of 2019 PEA two stage design against proposed PFS single stage irrigation design demonstrates that a longer leach cycle time and a two-stage irrigation regime gain in an additional 81 ounces of gold annually (approximately $117,000 per annum at gold price of $1,450 per oz); however, an additional power demand and additional cyanide consumption, due to oxidation, negates the revenue generated by the additional gold production and ultimately results in annual loss of approximately $385,000.

Overall, there is no penalty in terms of final or ultimate gold recovery as the ore underneath the newly stacked ore continues to leach. There is a slight delay by 0.11% of gold production for single stage irrigation, however, this could be offset by the simplification of the flowsheet for the single-stage irrigation schedule and the reduction of irrigation volume compared to the two-stage irrigation.

Compared to the double irrigation plant for the same ore tonnage, a smaller lixiviant solution pumping flowrate is required through the heap and associated ponds, tanks and piping network. Solution flow though the adsorption circuit will also decrease. Due to the shorter leach cycle time and reduced flux rate a cyanide consumption per tonne of ore will also decrease for the single stage irrigation option.

Improved operability and maintainability are expected for the single stage irrigation system due to the simplified solution management and smaller number of pumps and valves.

Based on the analysis, a single-stage irrigation schedule with the leach cycle of 90 days for both oxide and transitional ore types has been selected.

13.15.2 Sulphide Material

Based on the current understanding of the sulphide resource, the metallurgical responses observed in the testwork, and current metal prices, an economic processing route for the sulphide material has yet to be determined and the further work is required.

14 MINERAL RESOURCE ESTIMATE

14.1 Definitions

According to the May 10, 2014 version of CIM Definition Standards and the November 29, 2019 CIM MRMR Best Practice Guidelines:

- A *Mineral Resource* is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
- An *Inferred Mineral Resource* is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
- An *Indicated Mineral Resource* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.
- A *Measured Mineral Resource* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

14.2 Data Supplied

Drilling and trench data files were exported from DPM's AcQuire database and supplied in comma-separated-values ("csv") format. Wireframes were received in DXF format, as exported from Leapfrog, and were related to geology, mineralisation, weathering and fault domains.

The date of receipt of the final drill hole database is May 29, 2020.

14.3 MRE Estimation Procedures

The estimation of the Bigar Hill, Korkan and Korkan West Mineral Resources includes the following procedures and steps:

- Validation of the drill hole database received from DPM;
- Drill hole database import into MS Torque, a SQL based database manager integrated into HxGN MinePlan 3D;
- Importing into HxGN MinePlan 3D all geological, mineralisation, weathering and fault domain solids received from DPM in DXF format;
- Generation of basic descriptive statistics for the different mineralisation and geological domains for assessing the statistical parameters;
- Statistical analysis of the sample lengths to determine a suitable length for compositing;
- Creation of Estimation Zones ("ESTZON"), based on a combination of geological and mineralisation information;
- Statistical analysis of bulk density data;
- Geostatistical analysis by ESTZON to assess the spatial continuity of gold assay values;
- Generation of a block model, with a parent block size of 20×20×10 m and sub-blocks of 5×5×5 m, covering all three deposits areas;
- Review of the KNA performed in the 2018 Estimate;
- Setup of interpolation parameters;
- Grade interpolation into the parent block using OK;
- Uniform Conditioning (UC) support change from the parent block size of 20×20×10 m into the SMU size of 5×5×5 m;
- Localized Uniform Conditioning (LUC) by means of SMU grade ranking based on a separate OK ranking model;
- Weathering domains coding in the resulted SMU size block model and interpolation of other elements (S, Ag and Fe) using IDW2;

- Interpolation of density values for mineralised domains using IDW2 and assigning of mean densities values for waste domains by combination of geological and weathering domains information;
- Validation of the MRE using visual inspection, statistical comparisons, swath plots, HERCO analysis and review of grade tonnage curves;
- Selection of optimised pit shells, using the Lerchs-Grossmann Algorithm, to constrain part of the mineralisation that has demonstrated economic viability;
- Classification of the Mineral Resources according to CIM Definition Standards whose definitions are incorporated by reference into NI 43-101.
- Mineral Resource reporting and final statement.

14.4 Drill Hole Database and Data Verification

14.4.1 Drill Hole Database

The drill hole database was supplied to DRA in CSV format as exported from DPM's Acquire database. Received files were first reviewed in Excel format to check for inconsistencies prior to import into MS Torque, the SQL based database manager of HxGN MinePlan 3D. An error with the initial handover files received in April 2020 and these errors were pointed out to DPM. According to DPM, they were related to the unintentional rounding of 'from' and 'to' values, due to the misconfiguration of the export templates within AcQuire. The final and corrected database was received on May 29, 2020 and this date is considered as the effective date for the purpose of the current MRE.

The breakdown of drill-holes by type, and other drilling attributes are summarised in Table 14.1 and Table 14.2. Gold is the only assay field of economical interest. In addition to gold, three (3) accessory elements, sulphur, iron and silver, were considered for the purpose of the MRE, to aid mine planning and mineral process optimisation.

Table 14.1 – Drilling Statistics

Deposit	Drill-hole type	No. of holes	Average Length (m)	Total Meterage (m)	No. of gold assays	Length Assayed (m) for gold	No. of Silver assays	No. of Iron assays	No. of Sulphur assays
Bigar Hill	RC	351	211.0	74,059.0	72,687	74,059.0	27,191	508	68,129
	Diamond	184	211.4	38,904.7	31,885	32,023.0	27,099	9,669	31,738
Korkan	RC	337	165.9	55,914.0	51,233	52,237.0	3,814	1,882	51,233
	Diamond	214	203.5	43,552.7	40,370	40,849.0	18,745	9,711	39,997
Korkan West	RC	17	93.8	1,594.0	1,281	1,594.0	1,281	1,281	1,281
	Diamond	114	139.3	15,883.7	15,630	15,665.0	15,630	15,413	15,630

Table 14.2 – Database Statistics

Description	Database Records, by Prospect			
	Bigar Hill	Korkan	Korkan - West	Total
Collars	535	551	131	**1,217**
Surveys	4,874	5,269	1,207	**11,350**
Assays	104,571	93,657	16,911	**215,139**
Lithology	105,708	94,448	17,090	**217,246**
Alteration	104,189	93,560	16,807	**214,556**
Geotechnical	20,084	23,894	7,782	**51,760**
Density	6,621	7,660	3,099	**17,380**

14.5 Interpretations and Geological Modelling Procedures

14.5.1 Geology

14.5.1.1 Geological Units

DPM generated a series of wireframes using Leapfrog and passed them to DRA for review and use in the MRE. The geological wireframes generated represent different stratigraphic units within the Project. For the three (3) deposits under consideration, these solids constitute:

Bigar Hill:

Overburden, Andesite Sill, Marl, Conglomerates (S2), Sandstones and Conglomerates (S1), Basal Breccia, Jurassic Limestone and Metamorphic Phyllite.

Korkan:

Overburden, Andesite Sill, Hornblende Diorite Porphyry, Marl, Conglomerates (S2), Sandstones and Conglomerates (S1), Basal Breccia, Jurassic Limestone and Metamorphic Phyllite.

Korkan West:

Overburden, Marl, Conglomerates (S2), Sandstones and Conglomerates (S1), Lower Cretaceous Limestone, Jurassic Limestone and Metamorphic Phyllite.

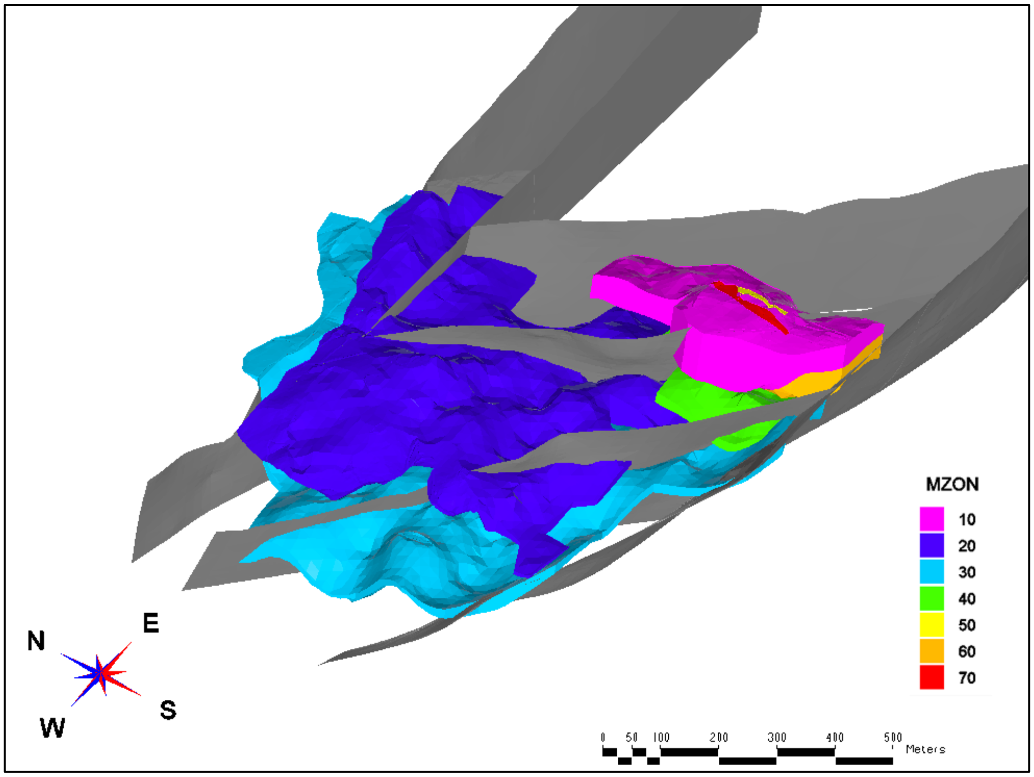
14.5.1.2 Faults

Fault surfaces were also generated by DPM using Leapfrog. Information used to construct fault wireframes are derived from a combination of correlated drill intersections, mapped surfaces position, inferred fault alignments deduced from sharp changes in lithology between drill holes, and other geological indicators.

Figure 14.1 illustrates the interpretation fault relationship at Bigar Hill from an oblique elevated view, looking northeast. The sub-horizontal shapes represent the mineralisation. The northern and southern bounding faults of the interpreted graben structure are clearly suggested.

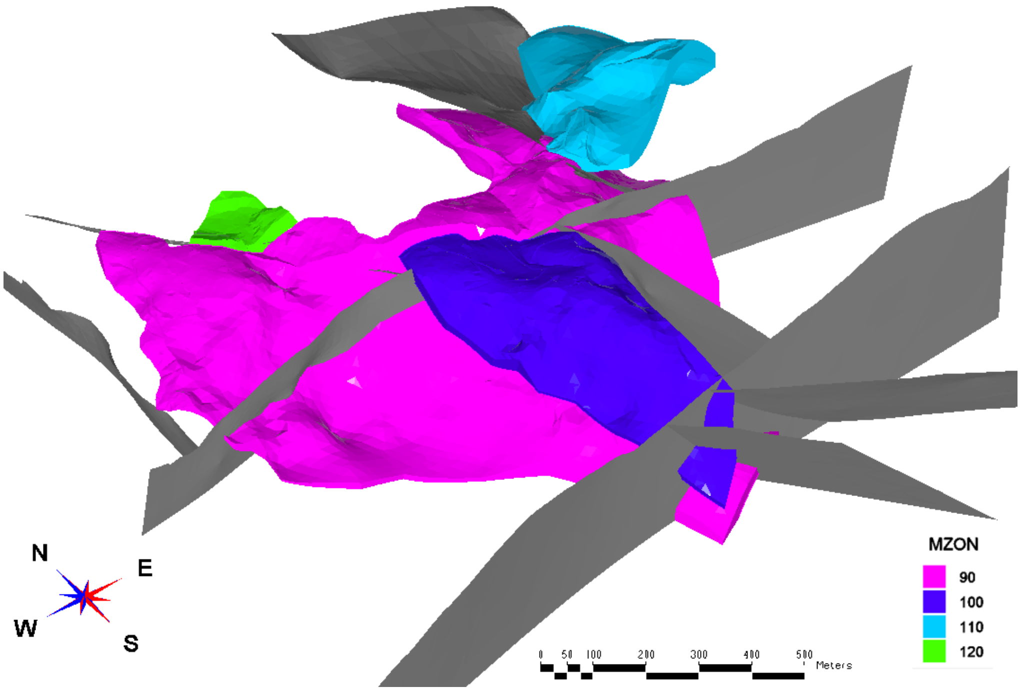
Figures 14.2 and 14.3 depict similar northeast looking oblique views of relationship between faults and mineralisation at Korkan and Korkan West.

Figure 14.1 – Locations of Faults and Mineralised Zones at Bigar Hill



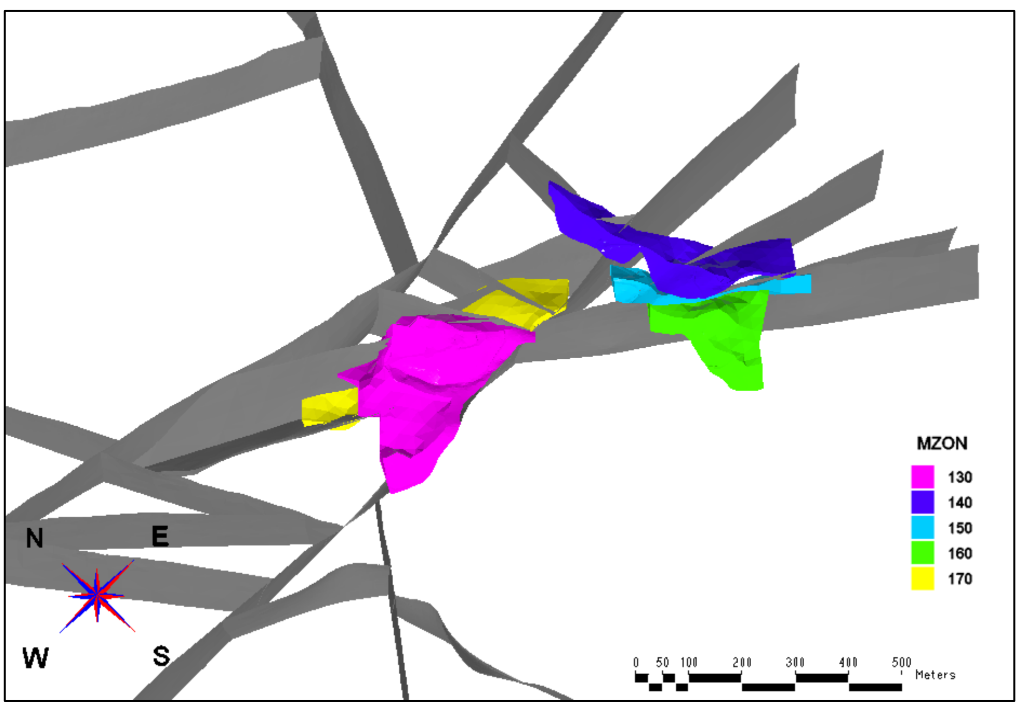
(Source: DRA Global, 2020)

Figure 14.2 – Locations of Faults and Mineralised Zones at Korkan



(Source: DRA Global, 2020)

Figure 14.3 – Locations of Faults and Mineralised Zones at Korkan West



(Source: DRA Global, 2020)

14.5.2 MINERALISATION

Considering the gradational character of the mineralisation and broad stratigraphic control, DPM found it more appropriate to apply looser constraints during definition of mineralised wireframes. For this purpose, assaying values were composited to 1 m and an indicative cut-off grade of 0.1 g/t Au was chosen to guide the generation of grade shells for each mineralised zone.

14.5.2.1 Bigar Hill

At Bigar Hill, elevated gold values are generally concentrated along certain horizons and most particularly in the S1/S2 and S1/limestone contacts. The mineralisation along the S1/S2 contact displays the best continuity, forming a relatively coherent body of above 0.5 g/t Au material. However, the upper and lower contacts of the mineralisation are more generally of a gradational nature, with an upper contact locally well defined but less so the lower extents, thus making the interpretation subjective.

Mineralised Zone 10 ("MZON") is a mineralised domain within the andesite sill. It strikes approximately north-south with a gentle dip to the east. It extends about 300 m in the east-west direction and about 500 m in the north-south direction. It's vertical footprint ranges from near surface to an elevation of approximately 580 m. MZON20 (S1/S2 contact) and MZON30 (S1/Limestone contact) zones jointly extend about 950 m east-west and 850 m north-south. Their vertical footprint ranges from near surface to a depth of 300 m from surface. The S1/limestone mineralisation (MZON30), in particular, pulls sharply upwards against the northern and southern graben boundaries.

MZON40 is the mineralised domain generated for the S2/VOL/MARL contact zone and it strikes roughly north-south with a dip ranging between -20° and -30° and flattening near surface. It extends about 300 m in the north-south direction and 240 m in the east-west direction.

DPM also prepared models of vein zones within the andesite sill referred to MZON50, MZON60, MZON70 and MZON80. Each of these vein zones was modelled separately except MZON80 which had to be back diluted into MZON10 because of it very small size which doesn't allow it to be coded properly in the block model.

14.5.2.2 Korkan

The Korkan prospect is situated on the same stratigraphic level as at Bigar Hill and Korkan West. Stratiform gold mineralisation at Korkan occurs mostly along the unconformable and breccia-like lower S1 and JLS limestone (S1/JLS contact) unit, and in Karst-infill zones at the upper contact of the JLS unit. The S1/BBX/JLS (MZON90) mineralisation forms a broadly hook-shaped body, due to folding, with an east-west extent of about 1,300 m. For the most part of its body, it is dipping to the south at around 25°, but on the north-east, it passes over a crest and then presents a limb dipping to the north. The north limb dips low in the west and up to -50° toward the east. The deepest part of the southern limb goes up to 440 m below surface.

MZON100 is located above the central position of the S1/JLS (MZON90) mineralisation south limb. It strikes roughly 300 m with a down dip extension of about 650 m and a vertical extension 300 m below surface. This zone is interpreted as being an up-thrusted segment of MZON 90.

MZON110 is a mineralised shell mostly located within the sandstones and conglomerates of the S1 and overlying S2 unit. It presents a north-east strike with an extend of about 550 m and a plunge ranging between -30° and -55° to the north, with a vertical extension reaching 270 m below surface.

MZON120 is a mineralised shell with the form of an inversed hook defined entirely within the Jurassic limestone (JLS). It presents two limbs the first oriented north-east with strike of about 270 m. The second limb is north-west oriented with a strike extent of about 250 m. Mineralisation in this domain encompasses primarily infilled karstified zones, interpreted to have formed along NW-SE feeder zones.

14.5.2.3 Korkan West

Korkan West mineralisation is located within two (2) main zones. MZON130 and MZON170 are two (2) mineralised shells located on the western flank of Korkan West. MZON130 is a mineralised body located mostly within the S1 and S2 geological units. It has an irregular shape and is striking roughly north-east with and extent of about 370 m and a plunge to the north ranging between -20° and -45°. Its vertical extent is about 110 m below surface.

MZON170 is a mineralised body principally located at the interface between the S1 and JLS contact. It has a north-west strike and dips to the north between -20° and -30°. MZON140 is a mineralised shell located on the western part of Korkan West and is defined at the interface between the conglomerates (S2) and Sandstones and Conglomerates (S1) geological units. It strikes north-west with a length of about 480 m and a global dip around 15° to the north.

MZON150 is a mineralised shell located within the upper and lower cretaceous limestone geological units. It presents a north-west trend with a length of about 375 m and is dipping about 60-70° to the north. MZON160 is a small mineralisation body entirely located within the upper cretaceous limestone. It is mostly oriented north-west with an irregular shape.

14.5.3 WEATHERING PROFILES

Interpretation of weathered profiles by DPM is based on geological logging information of weathering state, combined with cyanide soluble Au (LeachWELL) assay data and sulphur assaying results. An overview of the approach is discussed below.

14.5.3.1 Visual Logging Data

A re-logging program was undertaken in 2017 by DPM geologists and consisted of recording visual estimation of oxidation percentage within 1 m intervals of core and RC chips. All existing core was retrieved and re-logged whilst RC chips was relogged using chip tray samples or using photographs when chips were unavailable. This work has allowed DPM to demarcate changeover points from oxide to transitional and from transitional to fresh rock mineralisation. Five (5) categorical oxidation codes were also defined based on the style of oxidation. This coding system recorded a progressive diminution of oxidation level in five (5) steps going from completely oxidized to fresh rock code as shown in Table 14.3.

Table 14.3 – Oxidation Codes

Logging Code	Description
SOX	Strongly oxidised
MOX	Moderately oxidised
POX	Partially oxidised
WOX	Weakly oxidised
FRS	Fresh

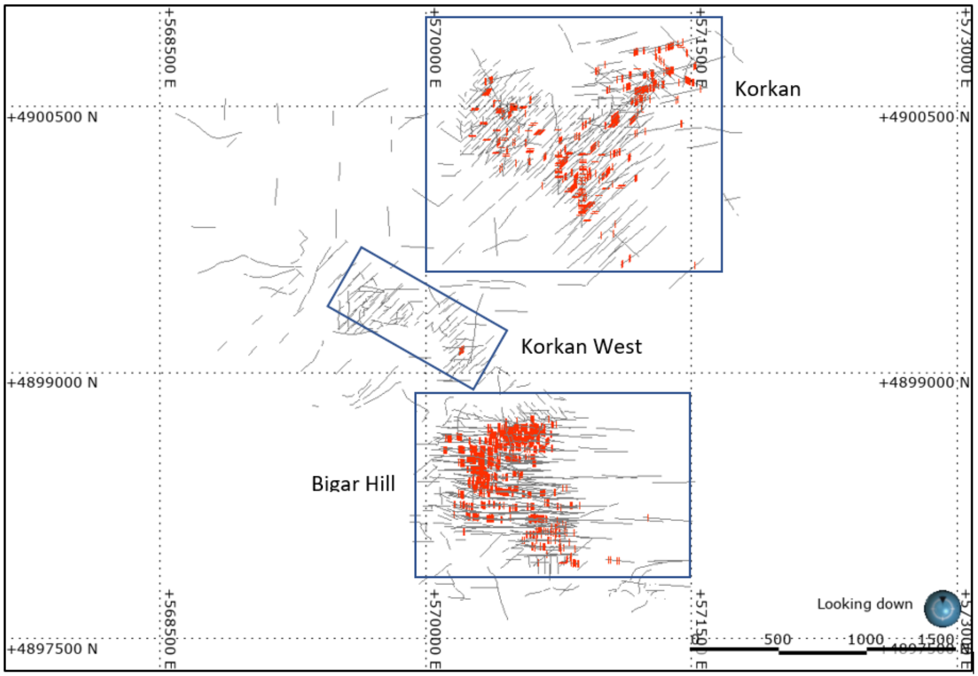
14.5.3.2 LeachWELL Cyanide Au Leach Assay Data

Agitated cyanide-gold leach tests were conducted in 2014 using on 3930 five-metre composite samples taken from diamond and reverse circulation drill holes from Bigar Hill (1810), Korkan (1201) and Kraku Pester (919) prospects. Composites were selected based on a 0.1 g/t contour. The LeachWELL analytical approach is discussed further in Section 12.2.4.

Spatial representivity is reasonable for the Bigar Hill prospect, whilst for Korkan the spatial coverage is poorer. For the Korkan West prospect, only one hole exists with CN leach tests. The figure below shows the spatial locations of composite samples for Bigar Hill, Korkan and Korkan West.

When CN soluble Au is plotted against total sulphur % a broad negative trend can be seen, characterised by an increasing proportion of CN soluble Au as the sulphur content decreases. Above 0.7 AuCN/Au and below 0.5 % Total sulphur, there is a subset of data points which upon visual review, corresponds well to areas of intense oxidation. Below this group a broad group of datapoints between 0.5 to 0.7 AuCN/Au can be defined which appears to align most with more transitional types of mineralisation. This cluster of data is more variable and breaks down above 1% sulphur.

Figure 14.4. – Drillholes Displaying Intervals of LeachWELL Cyanide-Gold Leach Tests



(DPM, 2021)

Estimates of visually logged oxidation percent were plotted against CN soluble Au. Above 0.7 AuCN/Au, there is a grouping of data that forms a cluster of data with increased values of logged oxide percent. This portion of the dataset was reviewed in 3D and found to correspond well to areas of intense oxidation. Beneath this group, there are composites which possess relatively lower visually estimated oxide values whilst still returning elevated AuCN/Au ratios. These composites represent transitional mineralisation which is only partially oxidised. The oxidised mineralisation forms well-defined clusters of data, allowing domains to be created, although the grouping for transitional is broader and not so well developed.

In the cyanidation process, copper–gold deposits containing significant amounts of cyanide soluble copper can lead to high cyanide consumption with low gold extraction. Base-metal carbonate replacement mineralisation is known to exist in the Korkan East prospect, which is interpreted to overprint sediment hosted gold mineralisation. Such mineralisation reaches its highest grades, up to 0.5% Cu over a metre, some 300 m below surface and is comprised of breccia bodies containing semi-massive sulphide mineralisation. Statistical and visual reviews of AuCN/Au ratios in the Korkan east area indicate that there no tendency for decreased levels of CN soluble Au with increasing levels of copper mineralisation.

14.5.3.3 Total Sulphur Assay Data

Sulphur samples are analysed by combustion with an infrared finish ("CSA6V"). This method allows a detection limit down to 0.025% total Sulphur.

14.5.3.4 Methodology

The construction of weathering domains follows the rules as shown below in Table 14.4. To ensure continuity, some inclusions of lower or higher-grade intervals have been made. CN soluble Au ratio data took precedence where available, followed by logged oxide percentages and total sulphur grade. The oxide category attribute was relied on very rarely during modelling.

Table 14.4 – Oxide and Transitional Modelling Criteria

Weathering Domain	Logged Oxide (%)	CNAu/Au Ratio	Total Sulphur Grade (%)	Oxide Category
Oxide	>10	>0.70	<0.5	SOX, MOX, WOX, POX
Transitional	>1	>0.50	<1	MOX, WOX, POX

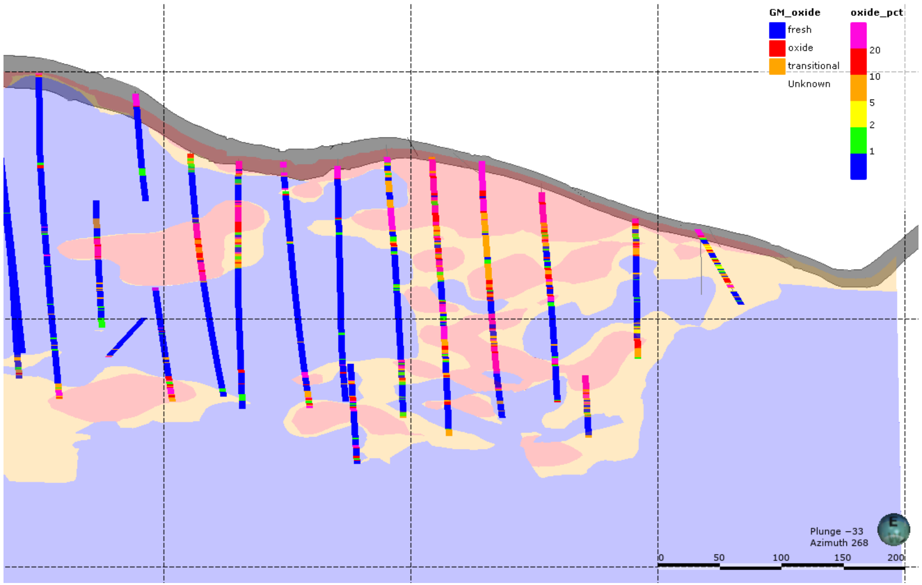
3D models were created using indicator interpolant models combined with surface models. Firstly, a model of the surficial weathering expression is made by modelling weathering change over points where proximal to the topography surface. Beneath the surficial weathering level, weathering that continues below surface is modelled using indicator interpolants. Interpolants were built at a 5m resolution with exact snapping selected. The indicator volumes were informed by a structural trend that mimics the key lithologic contacts for each prospect. Subsequently the two models are merged together, and the oxide weathering domain trimmed to the extents of transitional domain to ensure no overlap.

14.5.3.5 Validation

Visual reviews were undertaken by means of comparing the input datasets against the weathering models. Further review was undertaken in comparison with the geologic model, to ensure key stratigraphic trends were being honoured.

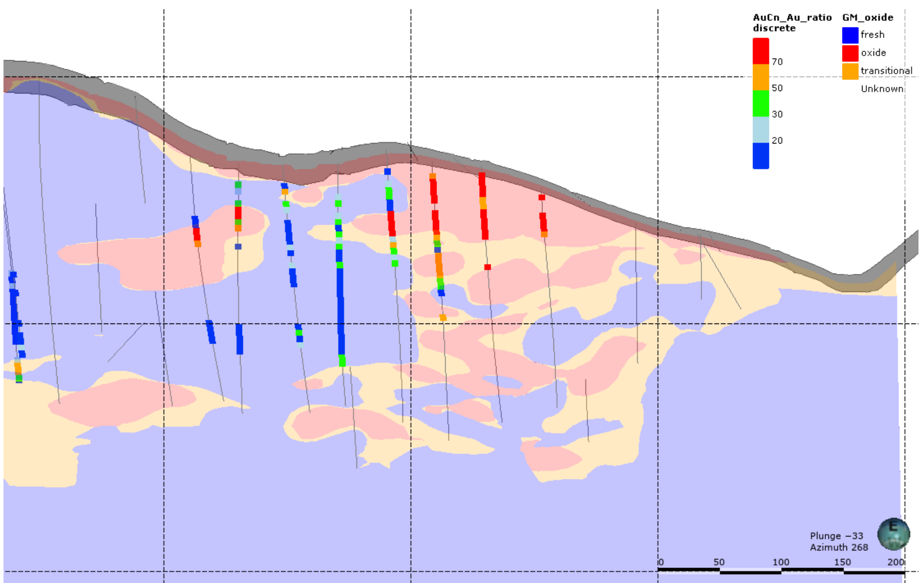
Boxplots and simple statistics were reviewed by domain for each input variable, as well as associated data used to derive AuCN/Au ratios. Data shows marked changes between each domain, with highest CNAu/Au ratios noted within the Oxide weathering domain whilst lowest values are noted within the fresh domain. Leach residues are lowest for the oxide domains whilst highest in oxide domains. The oxide percentage logged value appears to constrain well the different groups of data.

Figure 14.5. Weathering Models for Bigar Hill, showing the Oxide domain in red, transitional domain in Orange and Fresh domain in Blue



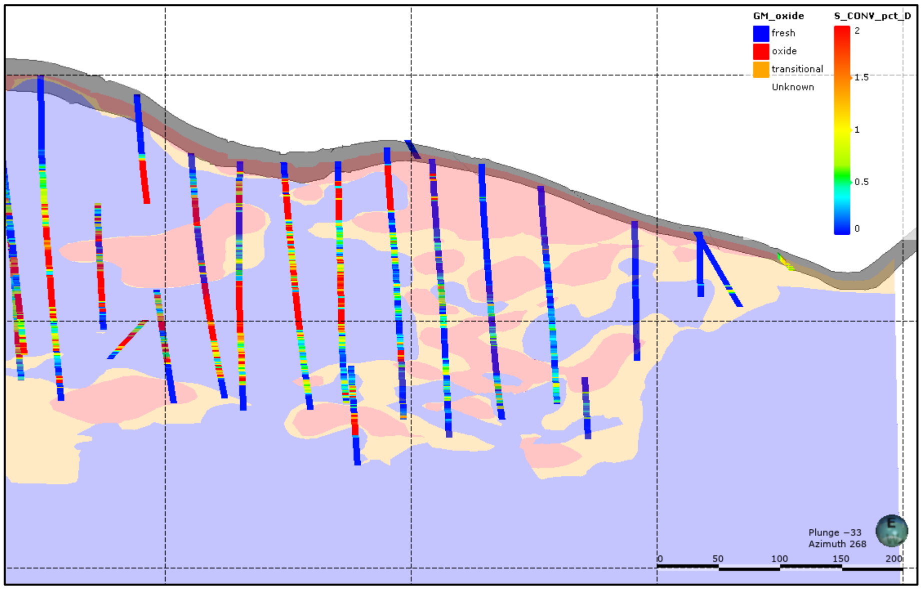
(DPM,2021) Overlaid by Logged Oxide % in drillholes.

Figure 14.6 – Weathering Models for Bigar Hill, showing the Oxide domain in Red, Transitional domain in Orange and Fresh domain in Blue



(DPM,2021) Overlaid by AuCN/AU ratio in drillholes.

Figure 14.7 – Weathering Models for Bigar Hill, showing the Oxide domain in Red, Transitional domain in Orange and Fresh domain in Blue



(DPM,2021) Overlaid by TS% in drillholes.

Figure 14.8. Boxplot and Simple Statistics for AuCN/Au for All Weathering Domains, Constrained by Mineralised Zones (DPM,2021)

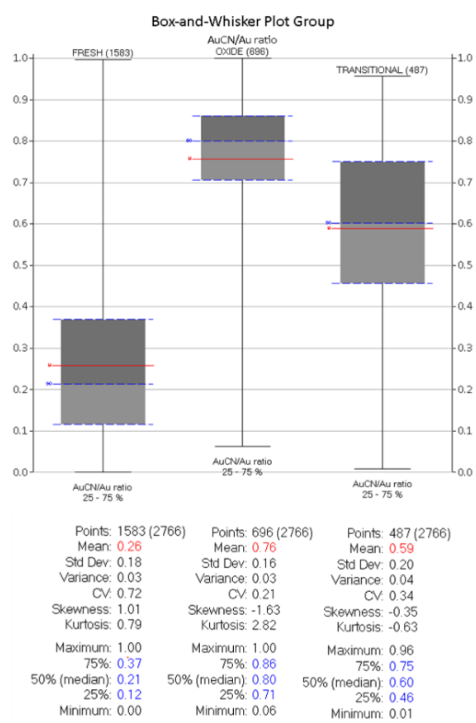


Figure 14.9. Boxplot and Simple Statistics for Logged Oxide %, for All Weathering Domains, Constrained by Mineralised Zones (DPM,2021)

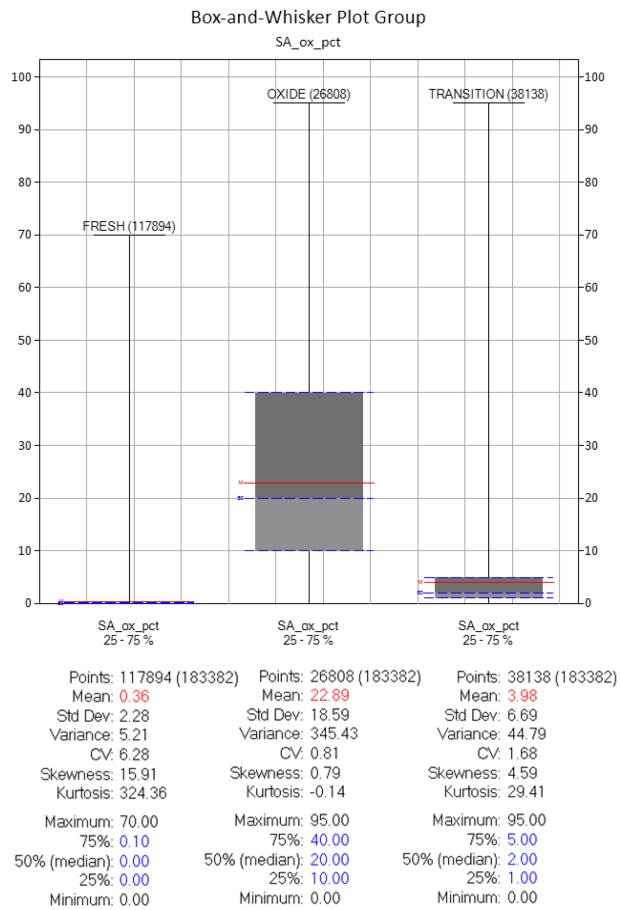
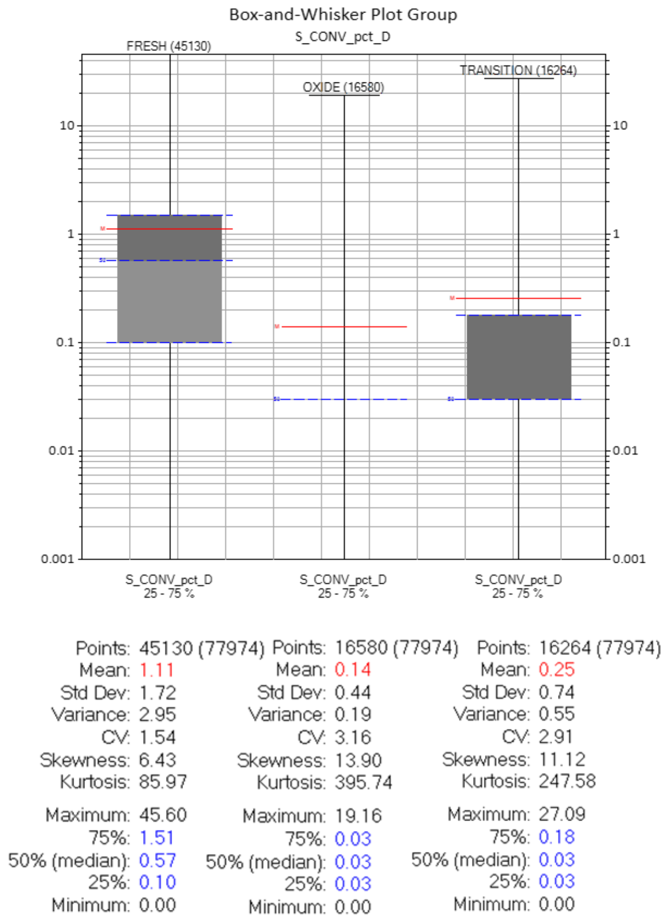


Figure 14.10 – Log Boxplot and Simple Statistics for TS%, for All Weathering Domains, Constrained by Mineralised Zones (DPM,2021



Where CN Soluble data was available, comparisons were undertaken against metallurgical testwork results. Generally, the results align with the weathering models although there are some anomalous results which will require further investigation to understand the variability of higher or lower recovery figures. The results indicate that AuCN/Au ratios are a suitable numerical measure that can be used for modelling weathering domains and that metallurgical testwork results align with the weathering domain outlines.

14.5.4 Topography

The topography of each of the three (3) deposits under consideration was supplied by DPM in DXF format and imported into Hexagon MinePlan 3D. Topographic surveys performed by DPM are discussed in greater detail in Section 8 of this Report.

14.6 Block Model Setup

A single block model was setup in Hexagon MinePlan 3D using the coordinates limits presented in Table 14.5. It was elected to proceed with a single block model for the entire Project area instead of building three different block models. A sub-blocked model was constructed and the selected size for parent blocks is 20 m x 20 m x 10 m, while sub-blocks have a size of 5 m x 5 m x 5 m. The selection of parent blocks size was guided by the average drill spacing over the Project area which is about 40 m x 40 m.

Table 14.5 – Timok Block Model Setup Parameters

Direction	Minimum (UTM)	Maximum (UTM)	Bock Size	Number of Blocks	Sub-block Size	Number of Sub-Blocks	Model Origin (UTM)
Easting (X)	569,360	571,660	20	115	5	4	569,360
Northing (Y)	4,897,660	4,900,860	20	160	5	4	4,897,660
Elevation (Z)	0	850	10	85	5	2	850
Rotation	N/A	N/A	–				

14.7 Block Model Coding

Geological, weathered and mineralised solids supplied by DPM were used to code the block model generated and the majority percent coding principle was used during the coding process. This approach means that a block is coded as falling within a solid when a least 51% of its volume falls within that solid. Areas that have not been coded with large parent blocks were coded with smaller sub-blocks, still using the majority coding principle.

Codes defined by DRA to inform the different geological, weathering and mineralised domains are presented in

Table 14.6 to Table 14.8. The same majority coding principle was applied to the drill hole database to code assays.

Table 14.6 – Codes for Geological Domains in the Block Model and Drill Hole Database

Geol. Unit	Abbrev.	Code
Overburden	OB	100
Andesite Sill	VOL	200
Hornblende Diorite Porphyry	HBP	300
Skarn	MSK	400
Marl	SMR	500
Sandstones and Conglomerates	S1	600
Conglomerates	S2	700
Basal Breccia	BBX	800
Upper (Jurassic) Cretaceous Limestone	JLS	900
Lower Cretaceous Limestone	KLS	1000
Jurassic Limestone	SLS	1100
Metamorphic Phyllite	MPH	1200

Table 14.7 – Codes for Mineralised Domains in the Block Model and Drill Hole Database

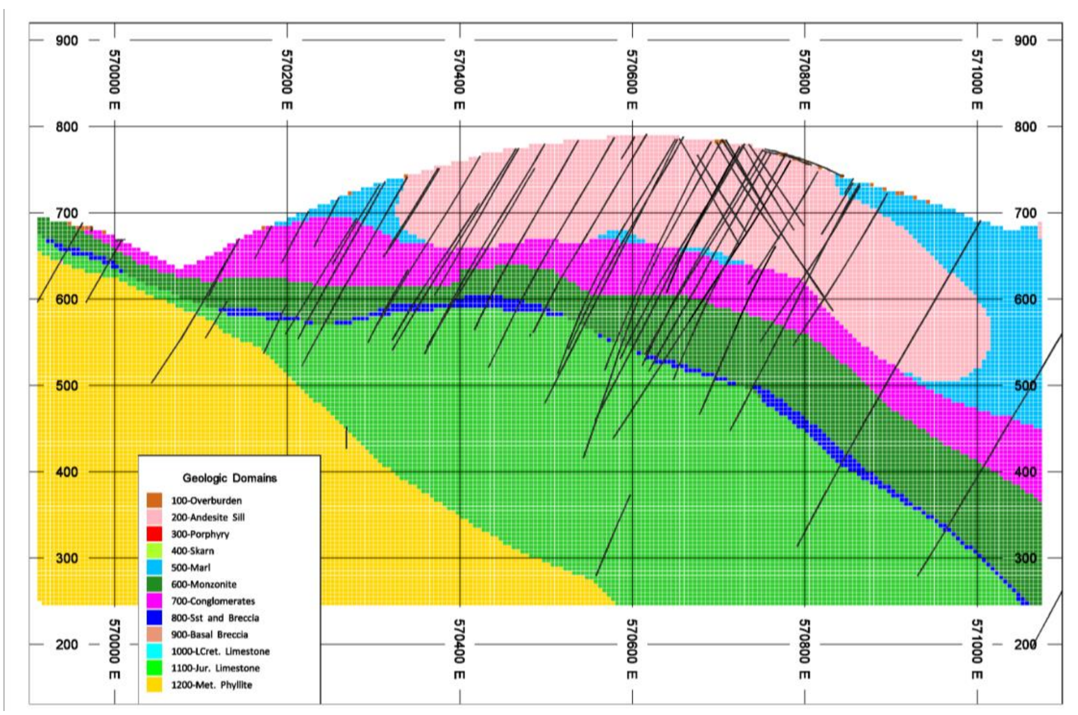
Deposit	Solid Name	MZON Code
Bigar Hill	Model10	10
	Model20	20
	Model30	30
	Model40	40
	Vein Zone SVZ2	50
	Vein Zone SVZ3	60
	Vein Zone SVZ4	70
Korkan	Model10	90
	Model20	100
	Model30	110
	Model40	120
Korkan West	Model1	130
	Model2	140
	Model3	150
	Model4	160
	Model5	170

Table 14.8 – Codes for Weathering Domains in the Block Model and Drill Hole Database

Deposit	Zone	Code
Bigar Hill	Oxide	1
	Transitional	2
	Fresh Rock	3
Korkan	Oxide	4
	Transitional	5
	Fresh Rock	6
Korkan West	Oxide	7
	Transitional	8
	Fresh Rock	9

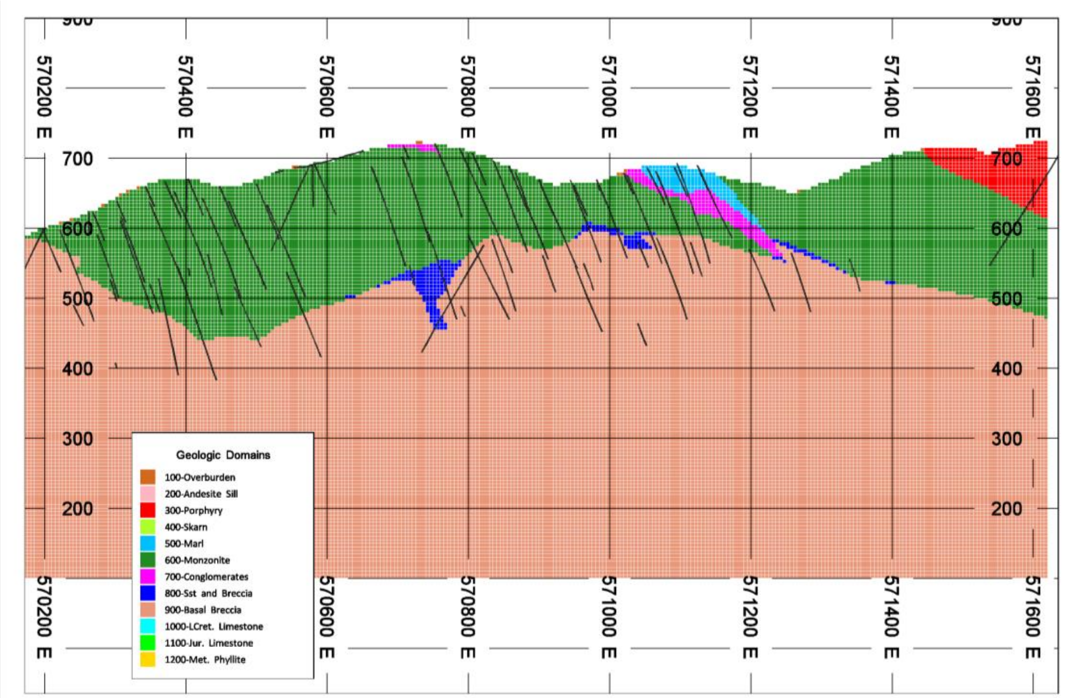
Figure 14.11 to Figure 14.13 depict the oblique 3D views of key geological units respectively at Bigar Hill, Korkan, and Korkan West while Figure 14.14 to Figure 14.16 illustrate oblique 3D views of the mineralised units coded respectively at Bigar Hill, Korkan, and Korkan West.

Figure 14.11 –View of Geological Units Coded in the BM at Bigar Hill



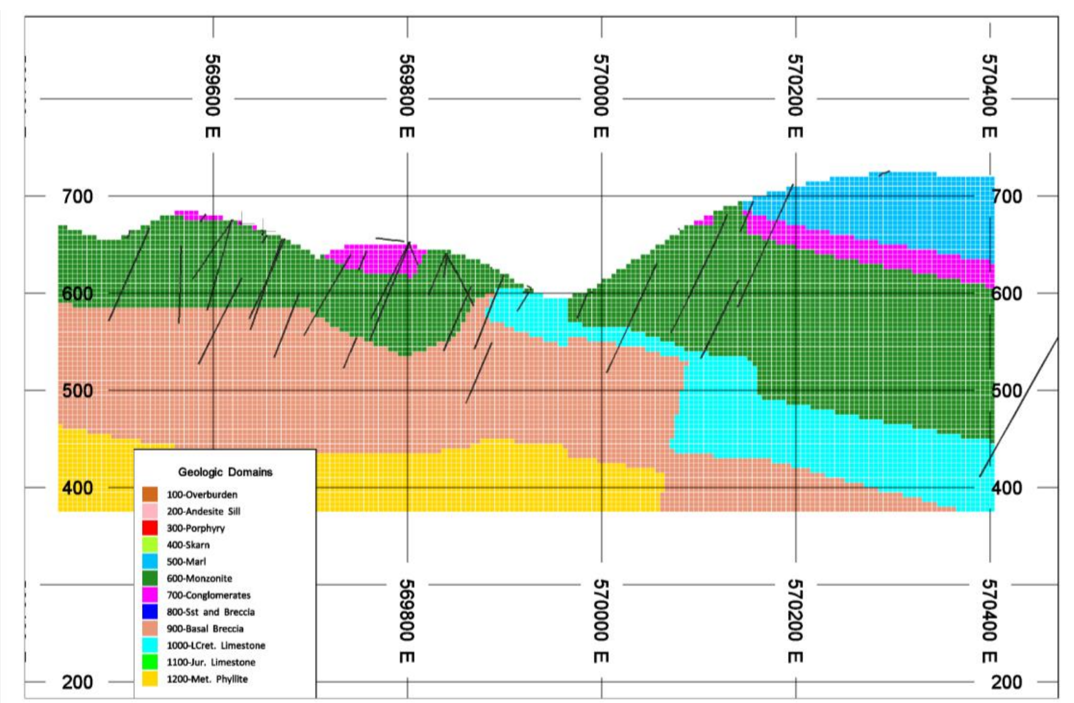
(Source DRA, 2020)

Figure 14.12 – 3D Oblique View of Geological Units Coded in the BM at Korkan



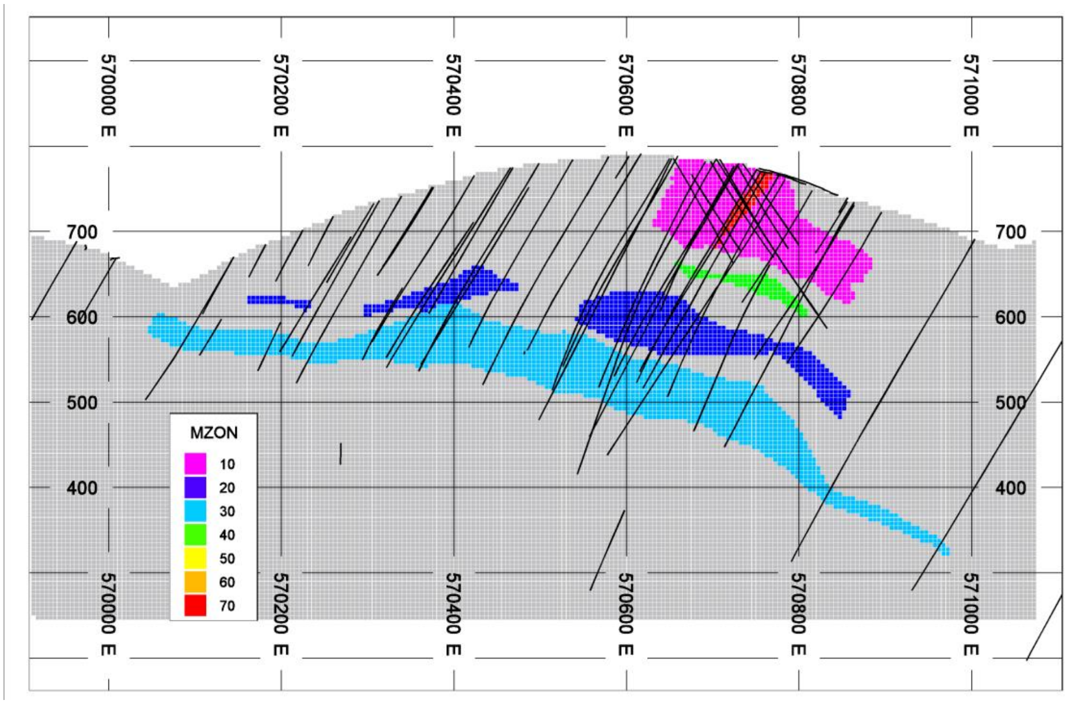
(Source DRA, 2020)

Figure 14.13 –View of Geological Units Coded in the BM at Korkan West



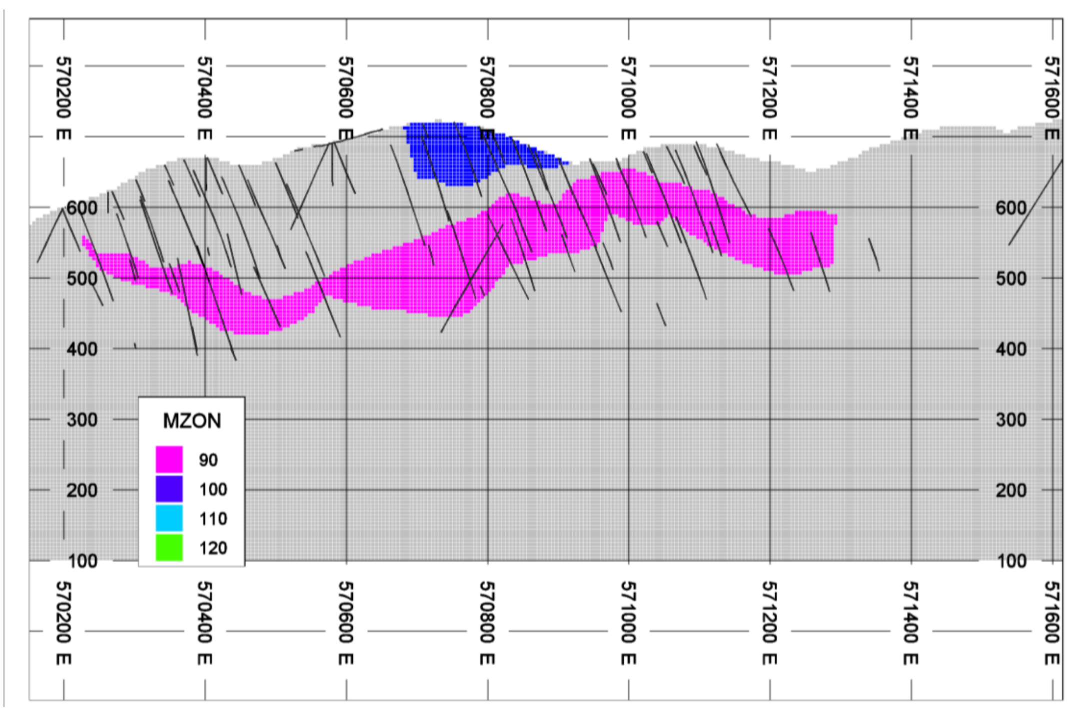
(Source DRA, 2020)

Figure 14.14 –View of Mineralised Domains Coded in the BM at Bigar Hill



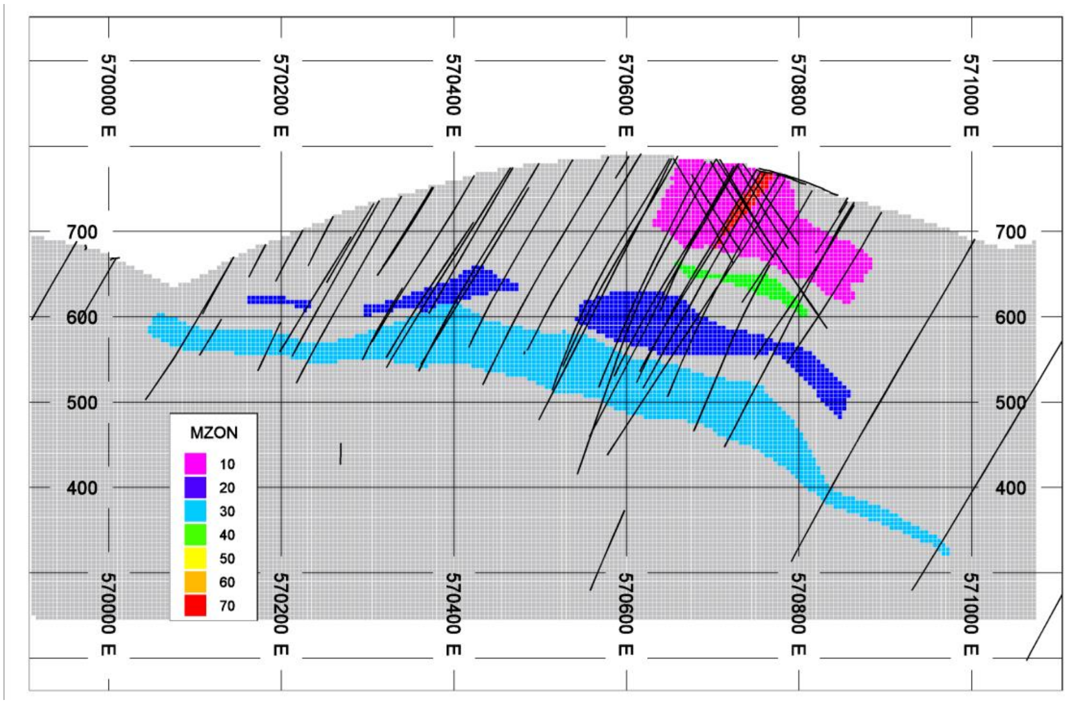
(Source DRA, 2020)

Figure 14.15 – 3D Oblique View of Mineralised Domains Coded in the BM at Korkan



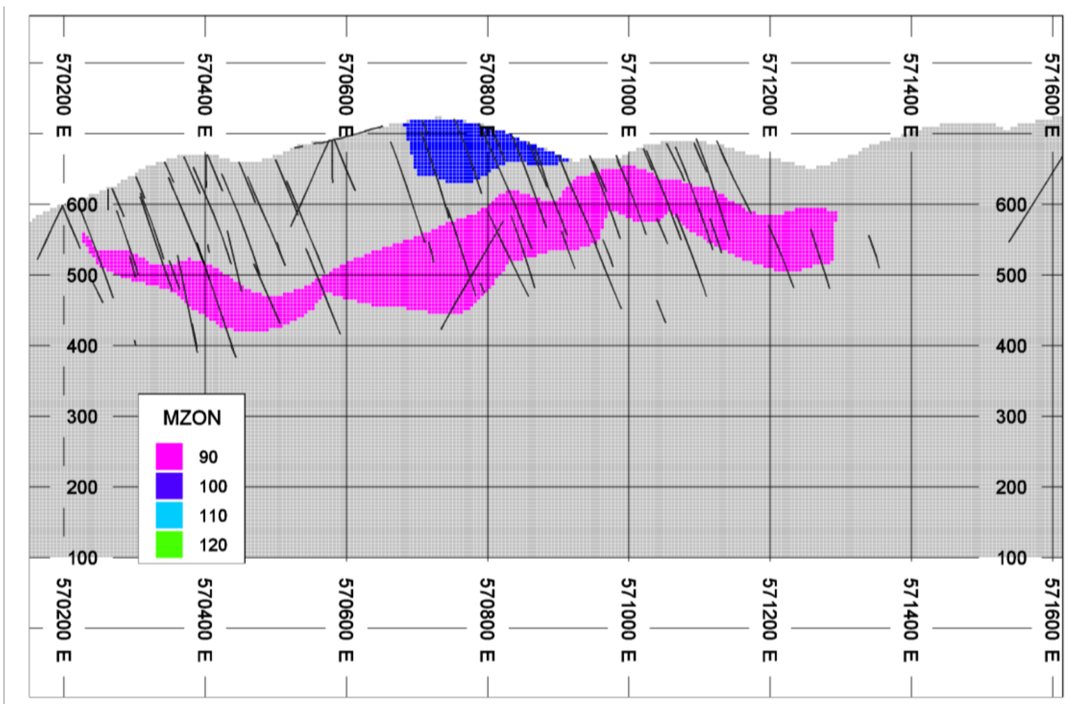
(Source DRA, 2020)

Figure 14.16 –View of Mineralised Domains Coded in the BM at Bigar Hill



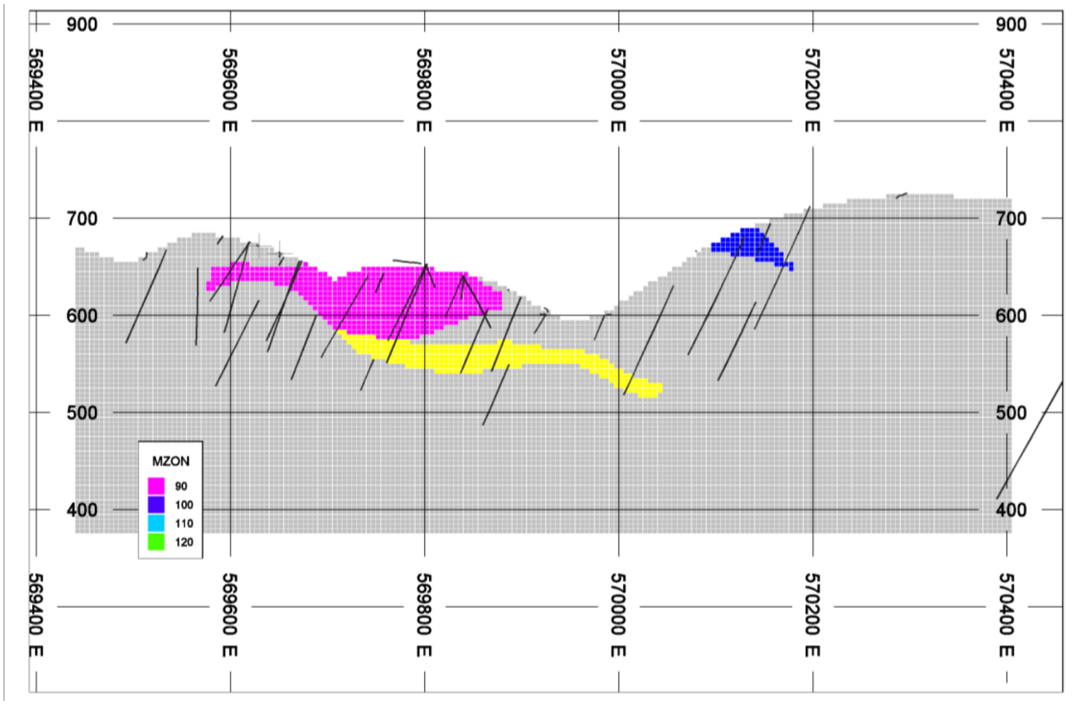
(Source DRA, 2020)

Figure 14.17 – 3D Oblique View of Mineralised Domains Coded in the BM at Korkan



(Source DRA, 2020)

Figure 14.18 – 3D Oblique View of Mineralised Domains Coded in the BM at Korkan West



(Source DRA, 2020)

14.8 Assays Statistical Analysis by Mineralised Domain

Assay raw data, coded by mineralised domain, were used to generate descriptive statistics by MZON for gold attributes that are presented in Table 14.9. MZON20 shows a high gold mean relative to other domains, with the lowest coefficient of variation. MZON10 has a high CV due to the more erratic structural controlled nature of the mineralisation in that domain. At Korkan, the most sampled mineralised domain is MZON90 (S1/JLS contact), and its coefficient of variation is also low compared to the two other mostly sampled domains (MZON100 and MZON110). MZON130 (S1/S2) is the most sampled, with the higher Au mean, at Korkan West.

Table 14.9 – Descriptive Statistics for Gold (g/t) by MZON and Deposit

Deposit	MZON	No. of Samples	Minimum	Maximum	Mean	SD	CV
Bigar Hill	10	6,917	0.001	25.87	0.14	0.95	6.92
	20	15,389	0.001	28.42	0.81	1.48	1.82
	30	17,332	0.001	46.76	0.38	1.09	2.85
	40	836	0.005	25.62	0.36	1.35	3.74
	50	60	0.005	14.12	0.51	1.91	3.74
	60	415	0.001	7.89	0.37	1.01	2.73
	70	465	0.005	30.15	0.72	2.20	3.07
Korkan	90	24,759	0.00	66.91	0.54	1.48	2.74
	100	6,191	0.001	27.42	0.21	0.69	3.27
	110	3,025	0.001	33.90	0.29	0.91	3.10
	120	909	0.005	3.78	0.19	0.33	1.81
Korkan West	130	2,017	0.001	12.13	0.74	1.09	1.48
	140	550	0.001	5.58	0.41	0.63	1.53
	150	524	0.001	1.83	0.25	0.32	1.30
	160	492	0.001	1.83	0.15	0.23	1.55
	170	506	0.001	5.12	0.22	0.44	2.05

Table 14.10 presents MZON volumes against sampling. More excessively sampled areas are correlated with domain volumes and, with the exception of Korkan West, MZON20, MZON30 and MZON90 are the major mineralised contributors to the global tonnage on the Project.

Boundary analysis, performed by CSA in 2018, demonstrated a hard boundary is still appropriate for resource estimation. DRA did a visual check using the updated information available and came to the same conclusion. It was then elected to use the same hard boundary principle for the current MRE update.

Table 14.10 – MZON Volume Against Number of Samples

MZON	Volume (Mm^3)	No. of samples
10	6.8	6,917
20	17.3	15,389
30	26.9	17,332
40	1.1	836
50	0.1	60
60	0.7	415
70	0.3	465
90	57.8	24,759
100	11.0	6,191
110	8.2	3,025
120	2.6	909
130	2.5	2,017
140	0.7	550
150	0.9	524
160	0.8	492
170	1.7	506

DRA also generated descriptive statistics by combination of MZON and weathering domains for the accessory elements having to be interpolated in this estimate run.

Table 14.11 to Table 14.13 present sulphur statistics respectively for Bigar Hill, Korkan and Korkan West. Globally, one notes a sulphur increase, as expected, from oxide domains to fresh rock domains. Korkan West possess lower grades for sulphur contained within mineralised units, whilst Bigar Hill and Korkan show quite similar statistics although sulphur at Bigar Hill is slightly higher.

Table 14.11 – Descriptive Statistics for Sulphur by MZON at Bigar Hill

Deposit (S% BH)	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Bigar Hill	10	Oxide	388	0.00	3.14	0.15	0.31	2.07
		Trans	424	0.00	3.68	0.21	0.52	2.44
		Fresh	5,366	0.00	5.61	0.30	0.83	2.74
	20	Oxide	4,201	0.00	19.16	0.20	0.65	3.23
		Trans	2,155	0.00	11.00	0.45	0.99	2.18
		Fresh	6,784	0.00	19.74	1.34	1.62	1.21
	30	Oxide	2,274	0.00	5.00	0.15	0.40	2.66
		Trans	3,835	0.00	23.74	0.23	0.70	3.06
		Fresh	9,029	0.00	16.64	1.00	1.39	1.38
	40	Oxide	–	–	–	–	–	–
		Trans	–	–	–	–	–	–
		Fresh	786	0.03	6.39	0.95	1.20	1.27
	50	Oxide	33	0.03	2.13	0.23	0.36	1.55
		Trans	33	0.03	0.83	0.09	0.15	1.69
		Fresh	87	0.03	5.54	1.67	1.69	1.01
	60	Oxide	–	–	–	–	–	–
		Trans	–	–	–	–	–	–
		Fresh	441	0.00	5.97	1.32	1.66	1.26
	70	Oxide	70	0.00	1.81	0.17	0.23	1.31
		Trans	72	0.03	4.44	0.62	1.21	1.94
		Fresh	399	0.03	5.73	1.40	1.55	1.11

Table 14.12 – Descriptive Statistics for Sulphur by MZON at Korkan

Deposit (S% KO)	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Korkan	90	Oxide	3,793	0.00	4.15	0.12	0.34	2.86
		Trans	6,150	0.00	27.09	0.19	0.68	3.57
		Fresh	13,156	0.00	45.06	1.31	2.09	1.60
	100	Oxide	1,707	0.00	6.81	0.13	0.39	2.89
		Trans	1,222	0.00	9.49	0.40	0.90	2.26
		Fresh	2,737	0.00	32.00	1.33	1.50	1.13
	110	Oxide	44	0.00	1.04	0.10	0.20	2.05
		Trans	275	0.03	8.24	0.24	0.62	2.60
		Fresh	2,706	0.00	26.95	1.25	1.98	1.58
	120	Oxide	120	0.03	0.66	0.04	0.07	1.71
		Trans	262	0.03	0.89	0.04	0.09	1.98
		Fresh	527	0.03	4.38	0.34	0.76	2.26

Table 14.13 – Descriptive Statistics for Sulphur by MZON at Korkan West

Deposit (S% KW)	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Korkan West	130	Oxide	1,638	0.00	1.79	0.08	0.15	1.93
		Trans	323	0.03	4.18	0.10	0.32	3.11
		Fresh	23	0.03	1.05	0.13	0.25	1.87
	140	Oxide	384	0.00	1.74	0.09	0.13	1.45
		Trans	100	0.03	1.32	0.06	0.15	2.53
		Fresh	32	0.03	2.04	0.18	0.36	1.95
	150	Oxide	328	0.00	1.80	0.12	0.26	2.19
		Trans	125	0.00	1.62	0.16	0.32	1.95
		Fresh	54	0.03	1.72	0.41	0.55	1.32
	160	Oxide	85	0.00	0.29	0.04	0.04	1.18
		Trans	160	0.00	2.52	0.13	0.27	2.04
		Fresh	242	0.00	2.10	0.23	0.48	2.08
	170	Oxide	236	0.00	1.71	0.07	0.21	2.94
		Trans	187	0.00	2.50	0.12	0.26	2.12
		Fresh	83	0.03	3.75	0.22	0.56	2.61

Silver statistics are presented in Table 14.14 to Table 14.16. As shown, silver appears to be elevated at Korkan and this is mainly attributed to MZON90, and at a lower level to MZON110. The minimum silver content is related to Korkan West.

Table 14.14 – Descriptive Statistics for Silver by MZON at Bigar Hill

Deposit Ag g/t BH	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Bigar Hill	10	Oxide	298	0.01	3.35	0.25	0.37	1.47
		Trans	246	0.01	22.70	0.33	1.45	4.41
		Fresh	2,414	0.01	17.10	0.39	0.57	1.45
	20	Oxide	2,505	0.01	66.50	1.47	3.48	2.37
		Trans	1,123	0.01	22.20	0.93	1.74	1.88
		Fresh	3,332	0.01	62.30	1.48	3.22	2.17
	30	Oxide	1,420	0.01	10.00	0.63	0.77	1.22
		Trans	2,251	0.01	31.40	0.59	1.16	1.97
		Fresh	4,842	0.01	53.40	0.71	2.05	2.90
	40	Oxide	–	–	–	–	–	–
		Trans	–	–	–	–	–	–
		Fresh	199	0.50	31.90	0.84	2.34	2.80
	50	Oxide	33	0.03	33.00	2.24	6.03	2.69
		Trans	9	0.03	10.00	1.43	3.28	2.29
		Fresh	47	0.03	11.70	1.00	2.49	2.48
	60	Oxide	–	–	–	–	–	–
		Trans	–	–	–	–	–	–
		Fresh	143	0.01	8.80	0.70	0.93	1.33
	70	Oxide	49	0.03	7.20	0.63	1.39	2.22
		Trans	43	0.03	10.00	0.58	1.60	2.78
		Fresh	179	0.03	10.00	0.65	1.16	1.78

Table 14.15 – Descriptive Statistics for Silver by MZON at Korkan

Deposit Ag g/t KO	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Korkan	90	Oxide	872	0.00	104.00	1.38	4.79	3.48
		Trans	890	0.01	245.60	3.11	17.87	5.74
		Fresh	3,606	0.01	509.00	9.00	35.41	3.94
	100	Oxide	80	0.03	4.70	0.97	1.09	1.13
		Trans	70	0.03	0.50	0.33	0.22	0.66
		Fresh	112	0.03	18.80	1.98	3.39	1.72
	110	Oxide	10	0.01	0.50	0.34	0.20	0.59
		Trans	241	0.03	39.00	1.30	3.57	2.74
		Fresh	1,514	0.00	251.20	2.22	11.19	5.04
	120	Oxide	27	0.07	4.39	0.60	0.86	1.44
		Trans	135	0.03	10.00	0.79	1.74	2.21
		Fresh	448	0.03	10.00	0.66	1.35	2.05

Table 14.16 – Descriptive Statistics for Silver by MZON at Korkan West

Deposit Ag g/t KW	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Korkan West	130	Oxide	1,649	0.01	10.00	1.01	1.23	1.22
		Trans	345	0.03	10.00	0.43	0.97	2.27
		Fresh	23	0.03	0.44	0.09	0.11	1.22
	140	Oxide	241	0.03	4.85	0.77	0.85	1.10
		Trans	78	0.03	2.77	0.43	0.54	1.28
		Fresh	11	0.03	0.31	0.14	0.13	0.93
	150	Oxide	143	0.01	4.74	0.68	0.78	1.14
		Trans	71	0.03	2.71	0.29	0.46	1.62
		Fresh	39	0.03	1.83	0.17	0.36	2.14
	160	Oxide	49	0.01	4.23	0.29	0.62	2.12
		Trans	117	0.01	3.59	0.14	0.37	2.55
		Fresh	215	0.01	1.08	0.09	0.13	1.43
	170	Oxide	236	0.01	10.00	0.45	1.32	2.91
		Trans	187	0.01	5.76	0.28	0.60	2.18
		Fresh	83	0.03	1.55	0.16	0.26	1.59

Iron statistics are presented from Table 14.17 to Table 14.19. The elevated iron mean is for Bigar Hill. The iron statistics do not allow drawing any conclusion on its preferential concentration in a specific oxidation domain.

Table 14.17 – Descriptive Statistics for Iron by MZON at Bigar Hill

Deposit Fe % BH	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Bigar Hill	10	Oxide	158	0.01	6.91	5.11	0.61	0.12
		Trans	160	3.65	7.18	4.94	0.57	0.12
		Fresh	943	1.56	9.53	4.89	0.53	0.11
	20	Oxide	448	0.01	8.19	2.12	1.35	0.64
		Trans	159	0.01	5.09	1.07	0.94	0.88
		Fresh	393	0.18	9.65	1.75	1.47	0.84
	30	Oxide	343	0.01	12.79	1.69	1.71	1.01
		Trans	429	0.01	8.20	0.72	0.93	1.29
		Fresh	933	0.01	9.23	0.86	1.22	1.42
	40	Oxide	–	–	–	–	–	–
		Trans	–	–	–	–	–	–
		Fresh	–	–	–	–	–	–
	50	Oxide	33	3.80	5.85	4.97	0.53	0.11
		Trans	9	4.22	6.49	5.29	0.92	0.17
		Fresh	33	4.18	5.30	4.75	0.27	0.06
	60	Oxide	–	–	–	–	–	–
		Trans	–	–	–	–	–	–
		Fresh	–	–	–	–	–	–
	70	Oxide	38	4.14	7.48	5.21	0.77	0.15
		Trans	30	3.60	5.45	4.46	0.44	0.10
		Fresh	114	3.62	5.38	4.50	0.39	0.09

Table 14.18 – Descriptive Statistics for Iron by MZON at Korkan

Deposit Fe % KO	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Korkan	90	Oxide	440	0.01	5.84	1.48	1.15	0.78
		Trans	628	0.01	15.96	0.64	0.96	1.51
		Fresh	2,224	0.01	15.00	2.02	2.04	1.01
	100	Oxide	53	0.41	13.42	4.71	3.99	0.85
		Trans	27	0.34	1.38	0.67	0.25	0.37
		Fresh	17	0.48	8.74	3.88	1.65	0.43
	110	Oxide	5	1.13	3.75	3.11	1.11	0.36
		Trans	134	0.07	7.25	0.55	0.86	1.55
		Fresh	896	0.00	13.60	1.52	1.44	0.95
	120	Oxide	27	0.15	3.66	1.70	1.29	0.76
		Trans	135	0.06	1.97	0.41	0.39	0.94
		Fresh	448	0.04	4.97	0.57	0.90	1.58

Table 14.19 – Descriptive Statistics for Iron by MZON at Korkan West

Deposit Fe % KW	MZON	Weathering Zone	No. of Samples	Minimum	Maximum	Mean	SD	CV
Korkan West	130	Oxide	1,638	0.01	15.00	1.46	1.18	0.81
		Trans	323	0.28	7.91	1.34	1.39	1.03
		Fresh	23	2.10	4.98	3.52	0.75	0.21
	140	Oxide	241	0.34	6.59	2.00	1.51	0.75
		Trans	78	0.33	2.02	0.67	0.35	0.52
		Fresh	11	0.30	0.61	0.44	0.12	0.27
	150	Oxide	143	0.01	7.42	1.40	1.34	0.96
		Trans	71	0.07	2.48	0.76	0.66	0.87
		Fresh	39	0.04	2.34	0.53	0.61	1.15
	160	Oxide	49	0.01	5.66	1.70	1.90	1.12
		Trans	117	0.01	3.00	0.36	0.39	1.07
		Fresh	215	0.01	3.12	0.30	0.32	1.08
	170	Oxide	236	0.01	4.78	1.02	0.76	0.74
		Trans	187	0.01	4.88	0.62	0.80	1.30
		Fresh	83	0.09	5.83	0.64	0.99	1.55

14.9 Bulk Density

DPM transmitted to DRA, an updated density database containing 21,988 density measurements. This source density database was coded according to the geological units, the mineralisation domains and the different weathering domains. The total number of density measurements per deposit are 8,379 measurements for Bigar Hill, 9,739 measurements for Korkan and 3,193 measurements for Korkan West. With a more detailed look at the density measurement entries within mineralised domains per deposit, the following numbers can be presented: Bigar Hill: 2,383 measurements; Korkan: 2,599 measurements and Korkan West: 607 measurements.

Density was compilated respectively by weathering domain and geological unit, and furthermore by weathering domain and mineralised unit. The results are presented in Tables 14.2 and 0.21. Additionally, bulk density was also compiled by mineralised domains versus un-mineralised domains and presented in Table 14.22.

Globally for the same geological unit, it could be noted that bulk density variation between waste and mineralised domains is quite random. It is not possible to clearly see a demarcation one of separate density populations with a mineralised zone and another with the non-mineralised zone. Furthermore, bulk density variation between the different geological units is low and within the same weathering domain, the bulk density variation from one to another mineralised unit is also low.

However, looking at statistics per weathered domain (Table 14.20) it could be noted a global slight increase of the mean density, for most of the geological units, from weathered to fresh rock domain. For these reasons, it was elected to only use the different weathering domains as hard boundaries during density interpolation in the mineralised domains. Density of waste domains was simply determined using averages based on available measurement entries.

Table 14.20 – Mean Bulk Density by Weathering Profile and Geological Unit

Deposit	Geological Unit	LCODE	All		Weathered		Partially Weathered		Fresh	
			Density g/cm³	No. of samples	Density g/cm³	No. of samples	Density g/cm³	No. of samples	Density g/cm³	No. of samples
Bigar Hill	Overburden	100	2.62	37	2.62	21	2.63	16	–	–
	Andesite Sill	200	2.66	1,983	2.62	31	2.63	113	2.67	1,839
	Marl	500	2.66	397	2.57	20	2.66	68	2.66	309
	Conglomerates (S2)	700	2.63	760	2.53	29	2.58	30	2.63	701
	Sandstones and Conglomerates (S1)	600	2.65	1,367	2.61	168	2.63	263	2.67	936
	Basal Breccia	800	2.67	308	2.63	17	2.65	37	2.68	254
	Upper Cretaceous Limestone	900	2.65	455	2.62	62	2.65	57	2.66	336
	Jurassic Limestone	1100	2.66	2,945	2.60	139	2.65	938	2.67	1,868
	Metamorphic Phyllite	1200	2.73	115	–	–	2.64	6	2.73	109
Korkan	Overburden	100	2.45	4	2.45	4				
	Andesite Sill	200	2.58	812	2.55	4	2.50	53	2.59	755
	Hornblende Diorite Porphyry	300	2.66	474	2.57	3	2.65	5	2.66	466
	Marl	500	2.66	399	2.56	17	2.63	94	2.68	288
	Conglomerates (S2)	700	2.62	402	2.57	40	2.56	52	2.63	310
	Sandstones and Conglomerates (S1)	600	2.64	2,968	2.61	302	2.64	587	2.65	2,079
	Basal Breccia	800	2.69	84	2.56	3	2.71	7	2.69	74
	Jurassic Limestone	900	2.68	4,526	2.65	155	2.66	1,030	2.68	3,341
	Metamorphic Phyllite	1200	2.74	42	–	–	–	–	2.74	42
Korkan West	Overburden	100	–	–	–	–	–	–	–	–
	Marl	500	2.62	96	2.53	3	2.59	19	2.63	74
	Conglomerates (S2)	700	2.58	274	2.54	56	2.58	51	2.59	167
	Sandstones and Conglomerates (S1)	600	2.60	1,676	2.58	743	2.61	534	2.64	399
	Lower Cretaceous Limestone	1000	2.65	187	2.60	7	2.64	65	2.66	115
	Jurassic Limestone	900	2.65	987	2.63	114	2.64	254	2.66	619
	Metamorphic Phyllite	1200	2.69	12	–	–	2.69	8	2.70	4

Table14.21 – Mean Bulk Density by Weathering Profile and Mineralised Unit

Deposit	MCODE	All		Weathered		Partially Weathered		Fresh	
		Density g/cm³	No. of Samples	Density g/cm³	No. of Samples	Density g/cm³	No. of Samples	Density g/cm³	No. of Samples
Bigar Hill	10	2.66	411	2.62	14	2.66	46	2.66	351
	20	2.61	588	2.59	132	2.63	125	2.62	331
	30	2.66	1,280	2.62	112	2.65	388	2.68	780
	40	2.69	41	–	–	–	–	2.69	41
	50	2.67	13	2.63	3	2.68	1	2.68	9
	60	2.67	12	–	–	–	–	2.67	12
	70	2.65	38	–	–	2.60	8	2.67	30
Korkan	90	2.67	1,833	2.63	243	2.66	433	2.68	1,156
	100	2.63	335	2.59	48	2.62	117	2.65	170
	110	2.68	263	–	–	2.68	42	2.68	221
	120	2.67	168	2.66	4	2.64	39	2.67	125
Korkan West	130	2.58	331	2.57	261	2.60	70	–	–
	140	2.59	44	2.56	21	2.61	19	2.65	4
	150	2.65	29	2.66	9	2.65	10	2.64	10
	160	2.64	89	2.67	10	2.65	23	2.64	56
	170	2.63	114	2.59	44	2.65	48	2.66	22

Table 14.22 – Mean Bulk Density by Lithology and Mineralised Unit Versus Waste

Deposit	Geological Unit		All		Mineralised		Waste	
	Unit Name	LCODE	Density g/cm³	No. of Samples	Density g/cm³	No. of Samples	Density g/cm³	No. of Samples
Bigar Hill	Overburden	100	2.62	37	2.66	6	2.62	31
	Andesite Sill	200	2.66	1,983	2.66	502	2.66	1,481
	Marl	500	2.66	3.97	2.52	8	2.66	389
	Conglomerates (S2)	700	2.63	760	2.55	151	2.65	609
	Sandstones and Conglomerates (S1)	600	2.65	1,367	2.64	600	2.66	767
	Basal Breccia	800	2.67	308	2.67	166	2.68	142
	Jurassic Limestone	900	2.65	455	2.66	179	2.64	276
	Metamorphic Phyllite	1200	2.73	115	–	–	2.73	115
Korkan	Overburden	100	2.45	4	2.67	3	1.80	1
	Andesite Sill	200	2.58	812	2.69	3	2.58	809
	Hornblende Diorite Porphyry	300	2.66	474	2.66	6	2.66	468
	Marl	500	2.66	399	2.70	36	2.66	363
	Conglomerates (S2)	700	2.62	402	2.60	87	2.67	315
	Sandstones and Conglomerates (S1)	600	2.64	2,968	2.65	834	2.64	2,134
	Basal Breccia	800	2.69	84	2.69	79	2.69	5
	Jurassic Limestone	900	2.68	4,526	2.67	1,885	2.67	4,251
	Metamorphic Phyllite	1200	2.74	42	–	–	2.74	42
Korkan West	Overburden	100	–	–	–	–	–	–
	Marl	500	2.62	96	–	–	2.62	96
	Conglomerates (S2)	700	2.58	274	2.52	43	2.59	231
	Sandstones and Conglomerates (S1)	600	2.60	1,676	2.59	392	2.61	1,284
	Lower Cretaceous Limestone	1000	2.64	148	2.64	17	2.64	131
	Jurassic Limestone	900	2.65	987	2.65	155	2.65	832
	Metamorphic Phyllite	1200	2.69	12	–	–	2.69	12

14.10 Grade Capping

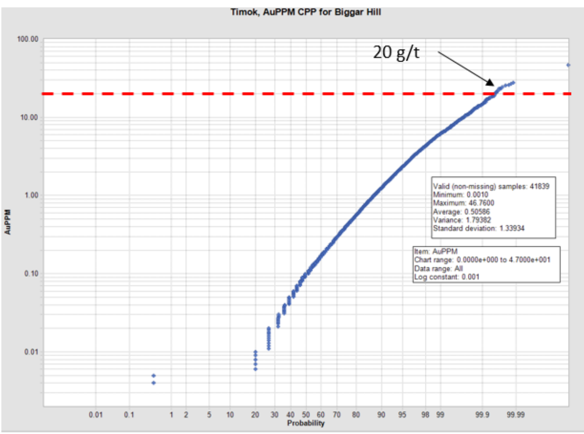
Grade capping is an approach traditionally used in mineral resource estimation to reduce the impacts of extreme high-grade values (outliers) on the estimate outcome, which are not representative of the sample population as a whole. In such cases where individual samples would unduly influence the values of surrounding model cells, without the support of other high-grade samples, capping is applied.

DRA generated a Cumulative Probability Plot (CPP) for each prospect and analyzed it to determine population breaks and ultimately choose the suitable value for grade capping. Outlier values, identified on the CPP curves, were identified and checked for their spatial location for each deposit. Figure 14.19 to Figure 14.21 show the CPP plots along with the associated histograms respectively for Bigar Hill, Korkan, and Korkan West.

For Bigar Hill, a gold high value break at 20 g/t on the CCP was considered as the limit for grade top cut value and all gold values higher than 20 g/t in the database for this deposit were reduced to this limit prior to sample compositing. For Korkan, an outlier grade capping value of 25 g/t Au was chosen. At Korkan West, the analysis of the CPP determined a statistical break at a value of 7 g/t which represents the base of the population of extreme values.

A total of 39 gold values were capped and most of them are related to MZON30 at Bigar Hill, MZON90 at Korkan and MZON130 at Korkan West. A statistical table was generated to compare means of un-cut and top-cut populations by mineralised unit and is presented in Table 14.23. Globally, it is noted that the impact of the grade capping process on gold mean values is low.

Figure 14.19 – Gold Cumulative Probability Plot and Histogram for Bigar Hill



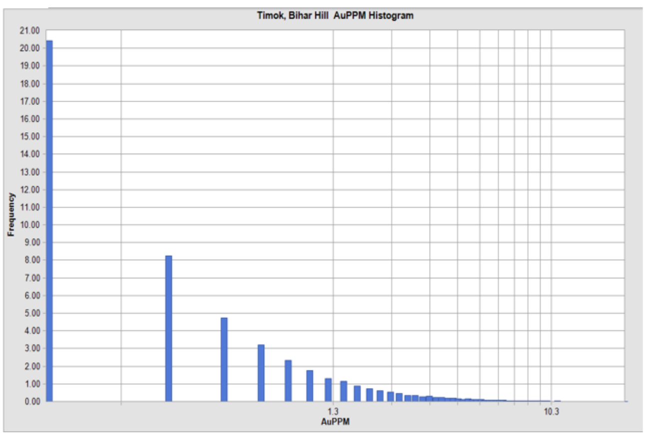


Figure 14.20 – Gold Cumulative Probability Plot and Histogram for Korkan

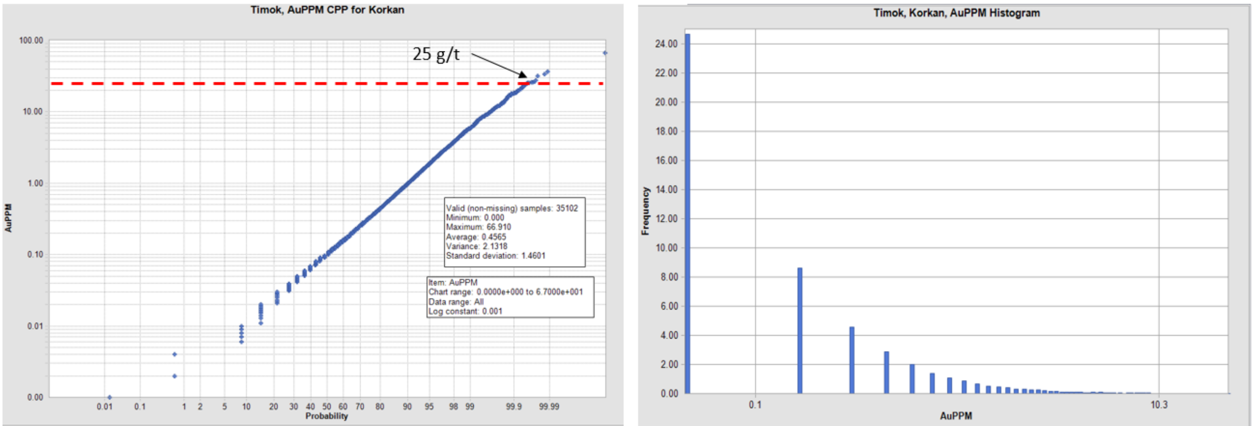


Figure 14.21 – Gold Cumulative Probability Plot and Histogram for Korkan West

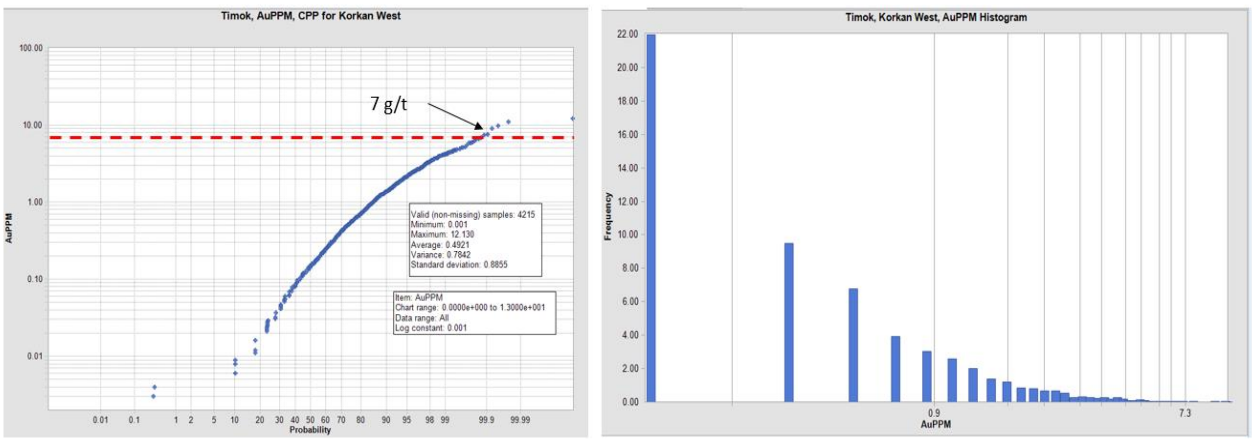


Table 14.23 – Mean Comparison Between Un-Cut and Top-Cut Populations by MZON

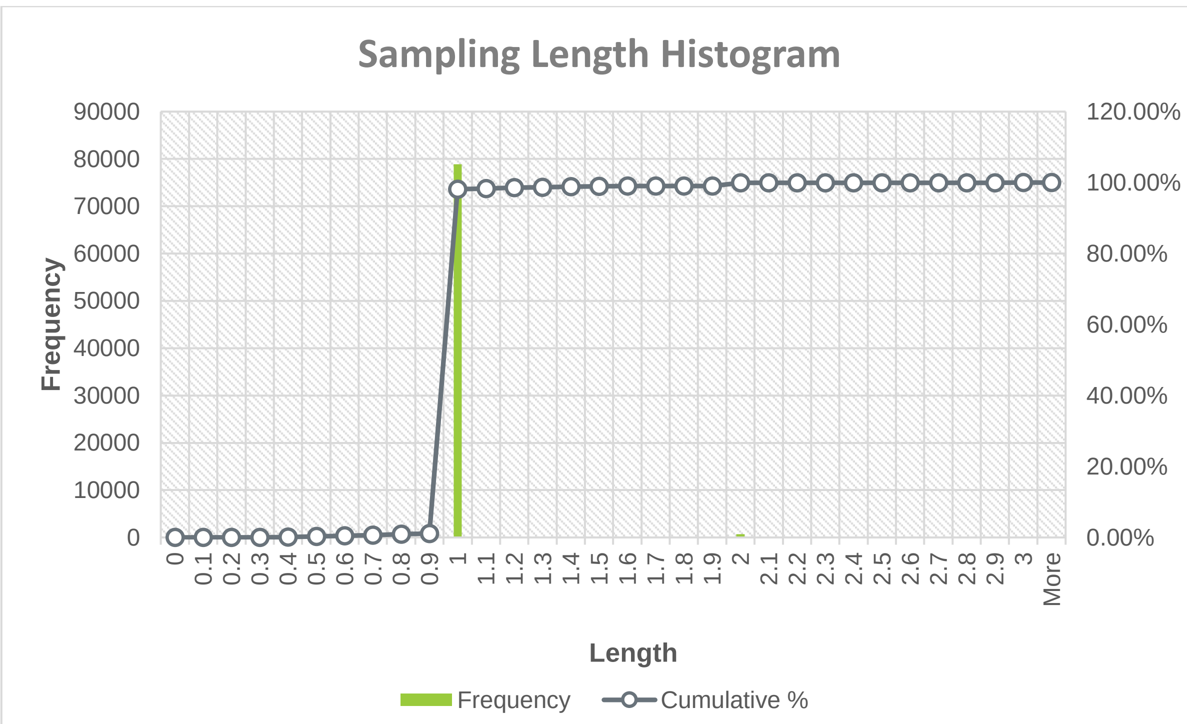
Deposit	MZON	Capped Values	Top-Cut Mean	Un-Cut Mean
Bigar Hill	10	3	0.14	0.14
	20	4	0.81	0.81
	30	9	0.38	0.38
	40	1	0.35	0.36
	70	2	0.68	0.72
Korkan	90	11	0.53	0.54
	100	1	0.21	0.21
	110	2	0.29	0.29
Korkan West	130	6	0.73	0.74
Global		**39**	**0.46**	**0.47**

14.11 Compositing

Compositing is a regularisation process with the objective giving identical weighting to all samples prior to the MRE. This process is most relevant where different sampling lengths are present. The choice of regularisation length is usually based on statistical analysis of the sampling lengths. A sampling histogram, based on assay data within mineralised shells, was generated and presented in Figure 14.22. Tables shown below are for all prospects and domains combined.

The normal practice is to composite at the statistical mode of the sampling since it will avoid or reduce introducing a bias related to samples over splitting. A compositing process should mostly consist of an aggregation process rather than a splitting process. A splitting process will introduce a bias since the same analytical results will just be repeated but at different lengths.

Figure 14.22 – Sampling Length Histogram



The histogram clearly shows 1 m as being the statistical mode of the sampling length population with about 97% of the mineralised shells sampled at this length. This is the length elected for compositing which was completed downhole using a fixed length method. With this approach, composite length is reset each time that estimation domain changes. All residual composites, less 0.3 m, were removed from the estimation process to avoid bias that may be introduced due to short composites lengths.

A superimposition of assay and composite histograms is presented in Figure 14.23 while Table 14.24 shows a comparison of descriptive statistics between assays and 1 m composites data. Composites count is about 750 entries higher than the assays count and this is explained by the fact that globally, the splitting process during compositing takes precedence over the aggregation process. However, this does not lead to any bias on the global mean.

Figure 14.23 – Histograms of Assays and 1 m Composites

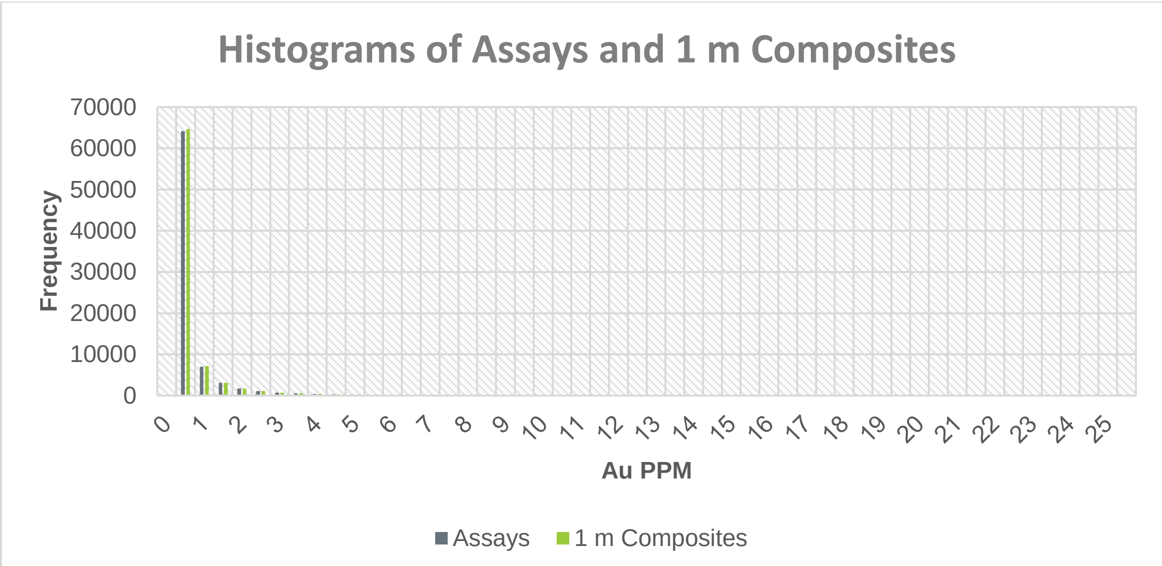


Table 14.24 – Comparative Descriptive Statistics Between Assays and 1 m Composites for All Prospects Combined

Description	Assays	1 m Composites
Mean	0.48	0.48
Median	0.10	0.10
Mode	0.01	0.01
St. Dev.	1.28	1.28
COV	2.68	2.67
Minimum	0.00	0.00
Maximum	25.00	25.00
Count	**80,386**	**81,162**

Following statistical and visual reviews of grade distribution and continuity, selected mineralised geological units were combined within each separate MZON, resulting in the creation of estimation zones that were used to control compositing, variography analysis and block model coding for

Mineral Resource interpolation. Table 14.25 presents the estimation zones defined for the three (3) deposits.

Table 14.25 – Defined Estimation Zones for the Three (3) Deposits

Deposit	MINZON	ESTZON	Geological Unit (GEOL)
Bigar Hill	10	500	Overburden, SMR, SLS, VOL
	20	600	BBX, S1, S2, VOL
		700	OB, SMR, S2
	30	800	BBX, MPH
		900	OB, S1, S2
		1000	SLS, SMR, VOL
	40	1100	S1, S2, SMR
		1200	VOL
	50	1300	OB, VOL
	60	1400	SLS, VOL
	70	1500	OB, VOL
	80	Back diluted to MZON10 because of it very small size to be coded properly in the Block Model	
Korkan	90	1700	JLS, MPH
		1800	HBP, S1, S2, OB, BBX, SMR, VOL
	100	1900	BBX, HBP, S1, S2, SMR, OB
	110	2000	JLS
		2100	BBX, HBP, S1, S2, SMR, VOL, OB
	120	2200	OB, BBX, JLS, S1
Korkan West	130	2300	JLS, S1
	140	2400	S1, S2, JLS
	150	2500	JLS, KL:S, S1
	160	2600	JLS, KLS
	170	2700	JLS, KLS, S1

14.12 Variogram Analysis

Variograms were generated to analyze the spatial continuity of grades for each ESTZON and ultimately guide in the definition of parameters to lead the mineral resource interpolation. The selected interpolation approach, Ordinary Kriging, is a linear geostatistical estimator which requires variogram parameters as inputs to guide the weighting to be applied to each sample depending on its spatial location in relation with the position to the block being estimated.

Variograms have been modelled by estimation zone using the 1 m composite raw data using Minesight Data Analyst (MSDA). For each estimation zone, a range of variograms, combining different horizontal directions and dip directions, were generated. More specifically, the software was set to generate variograms starting from 0° to 360° in a horizontal orientation with a 15° incremental step and starting from 0° to -90° in the vertical with a dip 10° incremental step. The resultant variograms were analyzed and the best axes of continuity, in strike, dip and plunge, were identified and selected.

The nugget effect was modelled from the downhole variograms with a lag set equal to the composite length of 1 m. The generated variograms were globally from poor to well-structured with moderate to long ranges and moderate to high nugget effect. Table 14.26 presents a synthesis of variogram parameters by estimation domain and by deposit.

Table 14.26 – Variogram Parameters by ESTZON and Deposit

Deposit	ESTZON	Azimuth (°)	Dip (°)	Range Major Axis	Range Semi-Major Axis	Range Minor Axis	Nugget	Sill
Bigar Hill	500	160	0	67	63	15	0.752	1.44
	600	220	0	95	88	15	0.350	0.89
	700	360	0	93	51	15	1.410	4.39
	800	260	-10	99	63	15	0.030	0.93
	900	280	0	134	48	15	0.210	0.71
	1000	360	0	73	60	15	0.660	2.17
	1100	180	-10	100	50	15	0.550	1.39
	1200	180	-10	100	50	15	0.550	1.39
	1300	360	0	73	73	15	0.500	2.4
	1400	280	-10	91	44	15	0.350	0.46
	1500	100	-50	84	71	15	1.450	0.91
Korkan	1700	80	0	57	54	15	0.510	1.19
	1800	120	-20	84	40	15	1.080	1.91
	1900	100	-60	58	41	15	0.140	0.41
	2000	80	0	70	66	15	0.070	0.83
	2100	160	-30	188	81	15	0.008	0.18
	2200	360	0	40	40	15	0.040	0.06
Korkan West	2300	20	-40	104	73	15	0.400	1.3
	2400	140	-20	120	74	15	0.100	0.29
	2500	280	-20	90	84	15	0.020	0.12
	2600	360	-10	95	94	15	0.010	0.03
	2700	100	0	72	70	15	0.030	0.15

Figure 14.24 to Figure 14.26 present modelled variogram examples for each of the three (3) deposits.

Figure 14.24 – Example Variograms of Gold Grades for Bigar Hill (ESTZON600)

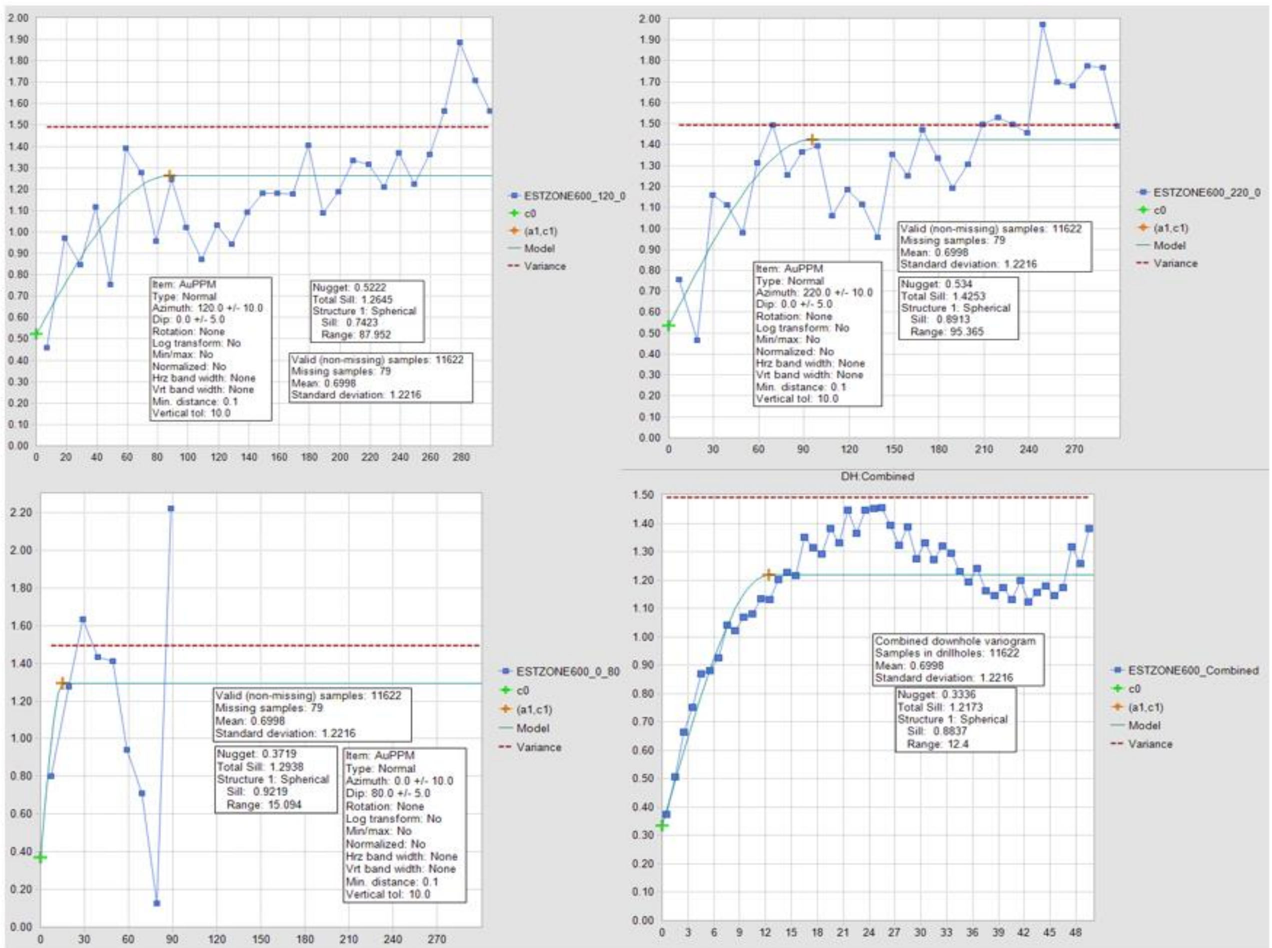


Figure 14.25 – Example Variograms of Gold Grades for Korkan (ESTZON1800)

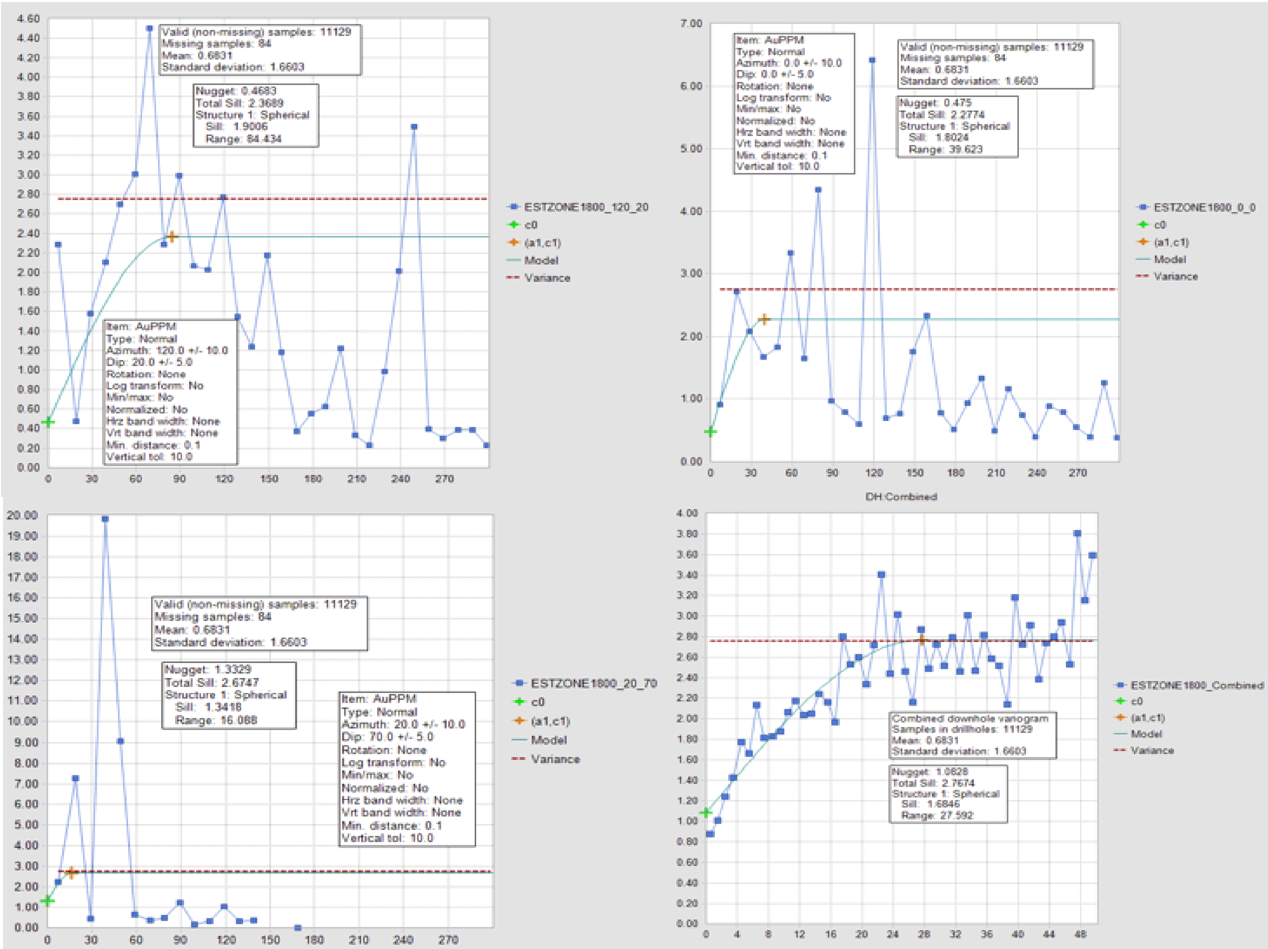
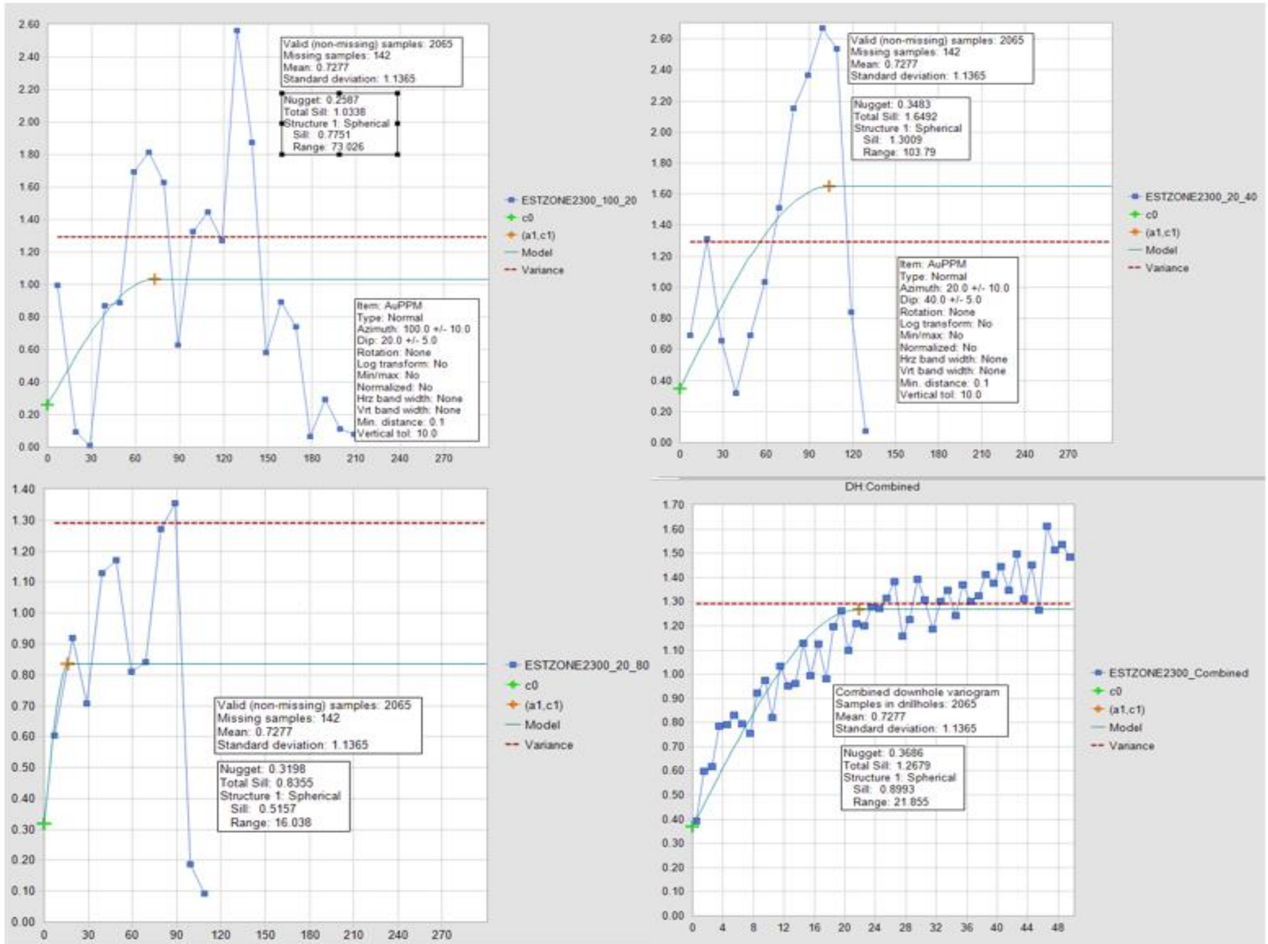


Figure 14.26 – Example Variograms of Gold Grades for Korkan West (ESTZON2300)



14.13 Kriging Neighborhood Analysis

OK is the estimation approach selected to interpolate the Mineral Resources of the Project, a Kriging Neighborhood Analysis (KNA) was required in order to define the optimal parent block size and to help in the definition of some theoretical optimal search parameters required as input prior to resource interpolation. Since the previous MRE, infill and extensional drilling on a 40 m × 40 m spacing has been undertaken on the flanks of the prospect areas. Given that limited, sub 40 m × 40 m spacing exists within the current dataset, DRA reviewed the KNA analysis performed in 2018 by CSA and elected to use the block sizes as determined during this study.

DRA is of the opinion that it is not necessary to re-conduct a new KNA since no major changes have been introduced since the 2018 estimate. For the definition estimation parameters, DRA has elected to combine findings of CSA 2018 KNA analysis with the judgment and experience of the QP responsible for the MRE.

14.14 Mineral Resource Estimation Procedure

The choice of the estimation approach was guided by the following considerations:

- Drill spacing is at an average of 40 m × 40 m × 10 m over the three deposits. This spacing does not allow direct grade estimation at the SMU scale, since estimation at block sizes beyond the resolution of the available data would result in an overly smoothed estimate. does not
- The necessity to accurately honor the volumes of different geological domains, characterized both by variable widths and geometries. Some domains can be particularly narrow (the vein zones hosted within the volcanics for instance);
- The need to reproduce the grade distribution at the envisaged level of selectivity for mining (SMU).

Taking all of the above criteria into consideration, it was elected to proceed with a recoverable resource estimation workflow to accurately estimate at the SMU scale from a larger panel scale estimate. The selected SMU size (5 × 5 × 5 m) is based upon the envisaged mining method and equipment size.

As an initial check, a visual review of the drill hole data and panel block data was undertaken to ensure that grade distributions were suitable for UC. This review identified that the grade values within the panel scale block model adhered to a diffusive pattern, exhibiting gradational grade transitions.

14.14.1 Panel Scale Gold Interpolation using ordinary Kriging

Mineral Resources have been interpolated by OK into a sub-block model defined in Section 14.6. The block model defined in Minesight, is a sub-block model with parent blocks of 20 m × 20 m × 10 m and sub-block size of 5 m × 5 m × 5 m. Sub-block were assigned grades based on the parent (panel scale) block size grade estimate.

This model has been used to interpolate Au values using search parameters defined in Table 14.27. Three (3) successive interpolation passes were used, and the search ellipse was relaxed from one pass to the subsequent pass in a way such that all blocks within the mineralised units are estimated at the end of the interpolation process.

For all passes, the maximum number of composites allowed for a single hole was set to three (3). The discretisation size used was 4 × 4 × 2 m. An octant search method was applied for composites data selection and a maximum of 4 composites was allowed per octant. Each estimation zone was split into different sub-domains, depending on its 3D morphology. This is to ensure that the search ellipse was locally oriented adequately in terms of strike, dip and plunge.

Before application of a change of support, the panel scale model was validated by visual review, comparisons of global statistics against declustered sample values and swathe plot analysis. The validation results indicated satisfactory estimation of Au grades within the model.

Table 14.27 – Interpolation Parameters

Deposit	ESTZON	Pass 1			Pass 2			Pass 3		
		Ranges	Composites		Range	Composites		Range	Composites	
			Min.	Max.		Min.	Max.		Min.	Max.
Bigar Hill	500	67 x 63 x 15	9	18	Range 1 x 2	9	18	Range 1 x 4	9	18
	600	95 x 88 x 15	9	18		9	18		9	18
	700	93 x 51 x 15	9	18		9	18		9	18
	800	99 x 63 x 15	9	18		9	18		9	18
	900	134 x 48 x 15	9	18		9	18		9	18
	1000	73 x 60 x 15	9	18		9	18		9	18
	1100	100 x 50 x 15	9	18		9	18		9	18
	1200	100 x 50 x 15	9	18		9	18		9	18
	1300	73 x 73 x 15	9	18		9	18		9	18
	1400	91 x 44 x 15	9	18		9	18		9	18
	1500	84 x 71 x 15	9	18		9	18		9	18
Korkan	1700	57 x 54 x 15	9	18	Range 1 x 2	9	18	Range 1 x 4	9	18
	1800	84 x 40 x 15	9	18		9	18		9	18
	1900	58 x 41 x 15	9	18		9	18		9	18
	2000	70 x 66 x 15	9	18		9	18		9	18
	2100	188 x 81 x 15	9	18		9	18		9	18
	2200	40 x 40 x 15	9	18		9	18		9	18
Korkan West	2300	104 x 73 x 15	9	18	Range 1 x 2	9	18	Range 1 x 4	9	18
	2400	120 x 74x 15	9	18		9	18		9	18
	2500	90 x 84x 15	9	18		9	18		9	18
	2600	95 x 94x 15	9	18		9	18		9	18
	2700	72 x 70x 15	9	18		9	18		9	18

14.14.2 Uniform Conditioning (UC) and Change Of Support

After Au values were interpolated at the panel scale block size, a change of support was applied to the estimate to emulate the grade and tonnage relationship at the SMU scale. For this purpose, it was elected to use Uniform Conditioning (UC) as a methodology with a change of support. The process was completed within the Datamine mine planning package by Datamine office SA.

UC uses a method of change of support to indirectly model a distribution of small block grades instead of directly modelling small blocks themselves. It estimates a recoverable resource (tonnes and grade) above multiple cut-off grades in a panel. The following sections document the steps taken during the UC phase of the estimate.

14.14.2.1 Declustering and Normal Scores Transformation of Samples

Sample data was transformed to normal score equivalent values. It is imperative that the normal score transformation correctly describes the distribution of grade values, as the Gaussian Anamorphosis is modelled on transformed grade values. As well, samples must be appropriately declustered so that the derived model is correct.

To select the optimum de-clustering size per ESTZON, the mean grade versus declustered grid size was plotted based on successively increased increments of the grid size. The grid size configuration with the lowest mean value was chosen as the optimum declustering size.

Figure 14.27 shows an example of the determination of the optimum grid size ESTON600 declustering. Table 14.28 shows the declustering sizes for all ESTZON.

Figure 14.27 – Definition of the Optimum Declustering Size (70×70×70) for ESTZON600

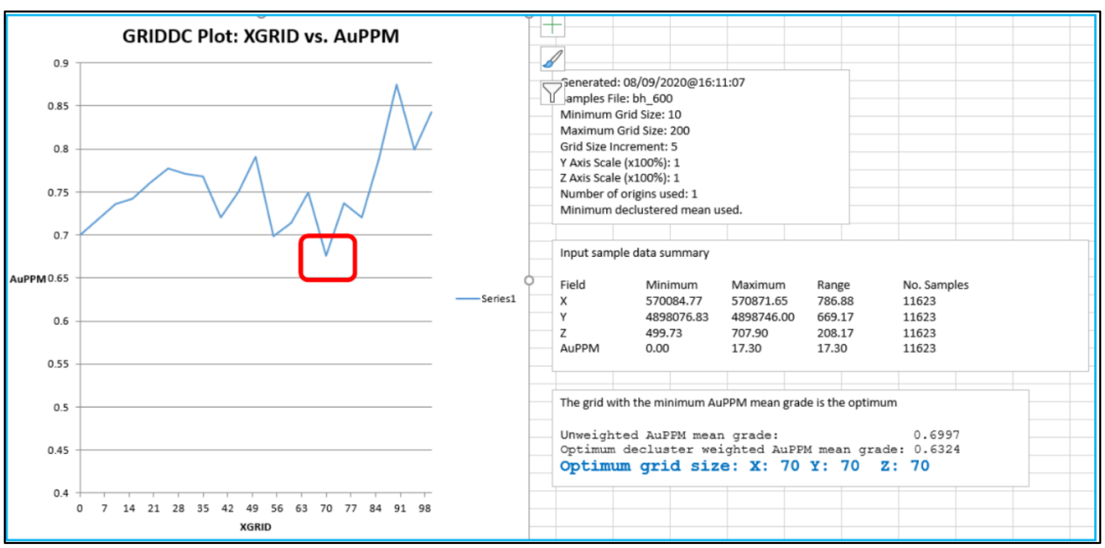


Table 14.28 – Declustering Sizes per ESTZON

Decluster	ESTZON	Decluster	ESTZON
60×60×60	ESTZON500	130×130×130	ESTZON1700
70×70×70	ESTZON600	70×70×70	ESTZON1800
70×70×70	ESTZON700	70×70×70	ESTZON1900
50×50×50	ESTZON800	60×60×60	ESTZON2000
95×95×95	ESTZON900	50×50×50	ESTZON2100
95×95×95	ESTZON1000	110×110×110	ESTZON2200
70×70×70	ESTZON1100	50×50×50	ESTZON2300
90×90×90	ESTZON1200	50×50×50	ESTZON2400
10×10×10	ESTZON1300	90×90×90	ESTZON2500
90×90×90	ESTZON1400	50×50×50	ESTZON2600
90×90×90	ESTZON1500	150×150×150	ESTZON2700

14.14.2.2 Uniform Conditioning and Change of Support

The discrete Gaussian model was used for calculating the change of support. The variogram models as presented in Table 14.26, along with SMU block size and number of discretization points, were used to calculate the SMU Change of Support coefficients (r). The r coefficient allows for the calculation of the distribution of grades at an SMU support, from a conditional panel grade.

Based upon review of cumulative probability plots for Au and to ensure reliable estimates at the grades of interest, the cut-offs as shown in Table 14.29 were used. As the panel estimation was done in original grade units, each estimate had to be transformed to Gaussian units using a panel anamorphosis. Each cut-off grade was also transformed to Gaussian units. The cut-offs grades were transformed using the SMU anamorphosis.

The grade distribution of the SMU's within a panel were calculated using panel kriged values for conditioning. The recoverable resources are defined by the proportion and quantity of metal above each cut-off grade.

Panel size of the parent block model was 20 × 20 × 10 m. The selected SMU (5 × 5 × 5 m) size is much smaller than the panels and is based upon the mining method and equipment size. Panel search volumes were determined using the variogram ranges and were kept same as the panel search parameters for panel model. Discretization points used for panel scale block estimate is 4 × 4 × 2 m and it was elected to keep same for the UC.

Table 14.29 – Cut-Offs Used during the UC

Cut-Offs (Au ppm)		
0.00	0.55	2.00
0.10	0.60	3.00
0.15	0.65	4.00
0.20	0.70	5.00
0.25	0.75	7.00
0.30	0.80	10.00
0.35	0.85	12.00
0.40	0.90	15.00
0.45	0.95	18.00
0.50	1.00	20.00

14.14.3 LOCALISED UNIFORM CONDITIONING (LUC)

UC is a non-linear change of support model that indirectly models a distribution of small block grades instead of directly modelling the small blocks themselves. The model estimates a recoverable resource (tonnes and grade) above multiple cut-offs grade within a panel. Localised Uniform Conditioning (LUC) is an enhancement to UC which doesn't change the results of UC, but rather, presents the recoverable resource into a more practical format for mine planning with a grade attributed to each block of the SMU size.

A grade model, of the same block size as the SMU, is estimated using OK. The purpose of this estimation is to identify the spatial location of high- and low-grade SMU block. Using the number of SMU in the panel, a set of average UC grades for each SMU in the panel is created that honor the proportion above cut-off grade curve. The mean of the set of UC grades is equal to the panel grade.

A second set of search parameters was required for estimating the model used for the localization of the UC estimate. These search parameters should use considerably less samples and smaller ranges in order to describe the local grade variability which will be used for the localization process, so the actual grade values are only used for the localization ranking process. A search volume of 100 × 100 × 15 m was used during the localization process with a minimum of 1 sample to conduct an estimate.

The set of UC grades are ranked from lowest to highest grade value. The set of UC grades are then mapped onto the grade model SMU, and the local UC grades are applied to the model which means that the SMU with the highest OK grade will be given the highest OK grade and the SMU with the lowest OK grade will be given the lowest OK grade. In conclusion, the result is a direct model of SMU blocks that respect the UC distribution for the panel and attempts to respect the estimated location of high and low grades within the panel.

14.15 Estimation of Accessory Elements

Accessory elements, S, Ag and Fe, were estimated directly into the block model after LUC postprocessing. Since S, Ag and Fe sampling grids are dissimilar, it was necessary to implement an individual interpolation runs for each of these elements. Sulphur is the most populated in the database followed by silver, and Iron is the less populated. Since the majority of these elements were analysed at a nominal sampling length of 1m, it was decided to convert assaying values of each element into 1m composites and proceed directly with their interpolation.

All the accessory elements were interpolated using inverse square weighted distance. A single search ellipse with a radius of 800 m was used for each element and weathering domains were setup as controlling boundaries for each interpolation. This means only composites coded in an oxidation domain are used to interpolate blocks located within this domain.

This was done to minimise smearing of grades between weathering domains. All weathering domains have hard boundaries, especially for sulphur. The minimum and maximum numbers of composites to code a block were respectively set to six (6) and fifteen (15) while the maximum number of composites allowed for a single hole was set equal to three (3).

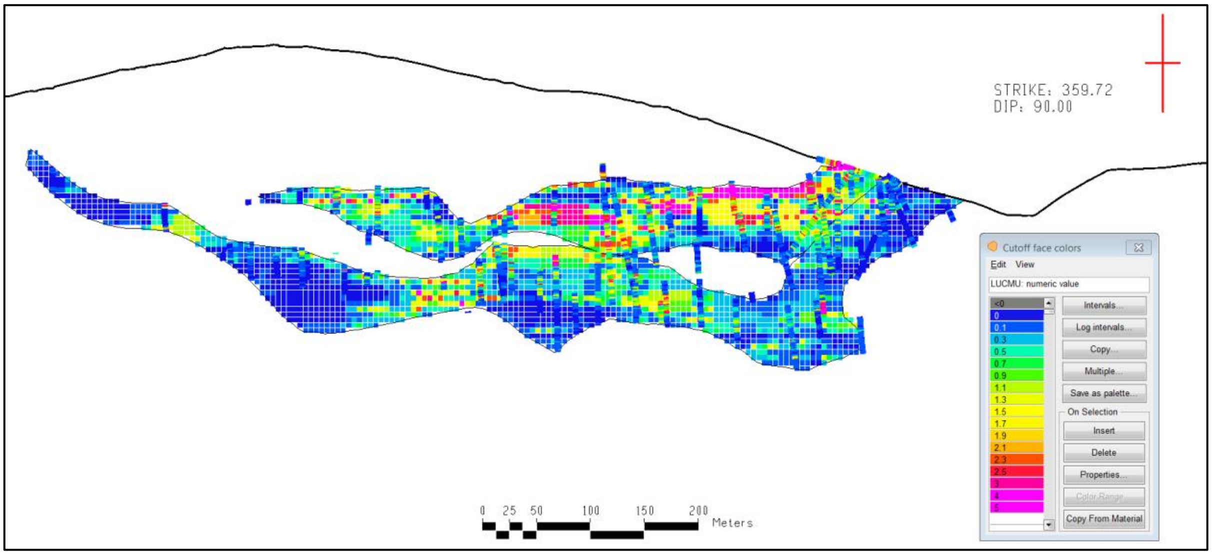
14.16 Bulk Density Estimation

Bulk density was interpolated into the final LUC block model. Bulk density raw data was converted into composites and used as the source for interpolation. Data selected for density interpolation was constrained within weathering domains to ensure no mixing of assay values occurred between domains. An inverse distance weighting squared approach was used to inform blocks and a single spherical ellipse with a radius of 600 m was used. The minimum and maximum numbers of composites to code a block were respectively set to three (3) and fifteen (15) while the maximum number of composites allowed for a single hole was set equal to three (3). Validation of the Mineral Resource Estimates

14.16.1 Visual Validation

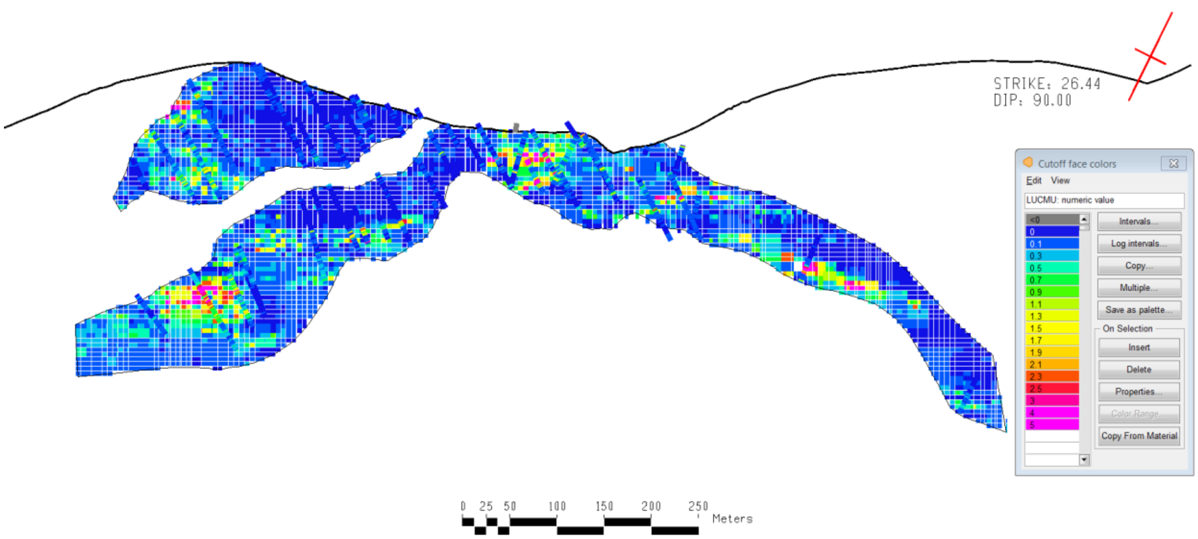
The first step in estimate validation of the LUC model is a visual comparison of estimated blocks grades with the composites grade values. This was done both in 3D and in 2D, on a sectional basis. Globally, block estimates compared well with composites input data, both in 2D and 3D. Figure 14.28 to Figure 14.30 depict examples of cross sections for Bigar Hill, Korkan, and Korkan West.

Figure 14.28 – Bigar Hill - Example Cross Section Showing Drill Hole Composites Grades vs. Block Grades (DRA, 2020)



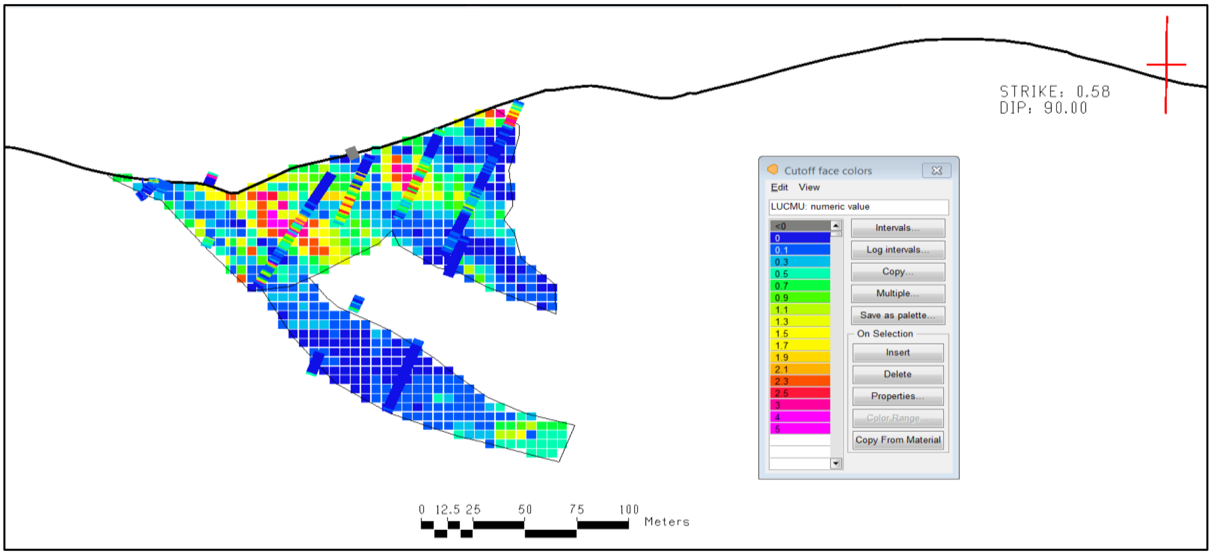
(Source: DRA Global 2020)

Figure 14.29 – Korkan - Example of Cross Section Showing Drill Hole Composites Grades vs. Block Grades (DRA, 2020)



(Source: DRA Global 2020)

Figure 14.30 – Korkan West - Example of Cross Section Showing Drill Hole Composites Grades vs. Block Grades



(Source: DRA Global 2020)

14.16.2 STATISTICAL VALIDATION

Results have also been validated statistically by generating descriptive statistics of de-clustered composites and naïve values and comparing these data with block model statistics for each deposit. The de-clustering procedure assigns equal weighting of samples based on the spatial distribution, which is necessary given the varying drilling densities at the Project. A grid-weighted de-clustering approach was chosen, using a cell size of 20 m × 20m × 10 m. Composites de-clustering was performed in MSDA.

The global statistics of Au g/t (Indicated Mineral Resources only) are summarised in Table 14.30. The mean grades in the estimated block model were compared to the mean of the composite set declustered and un-declustered.

Globally, the models validate well as block model mean and de-clustered composites mean are close. Table 14.30 presents the comparative statistics generated for the estimate validation purpose.

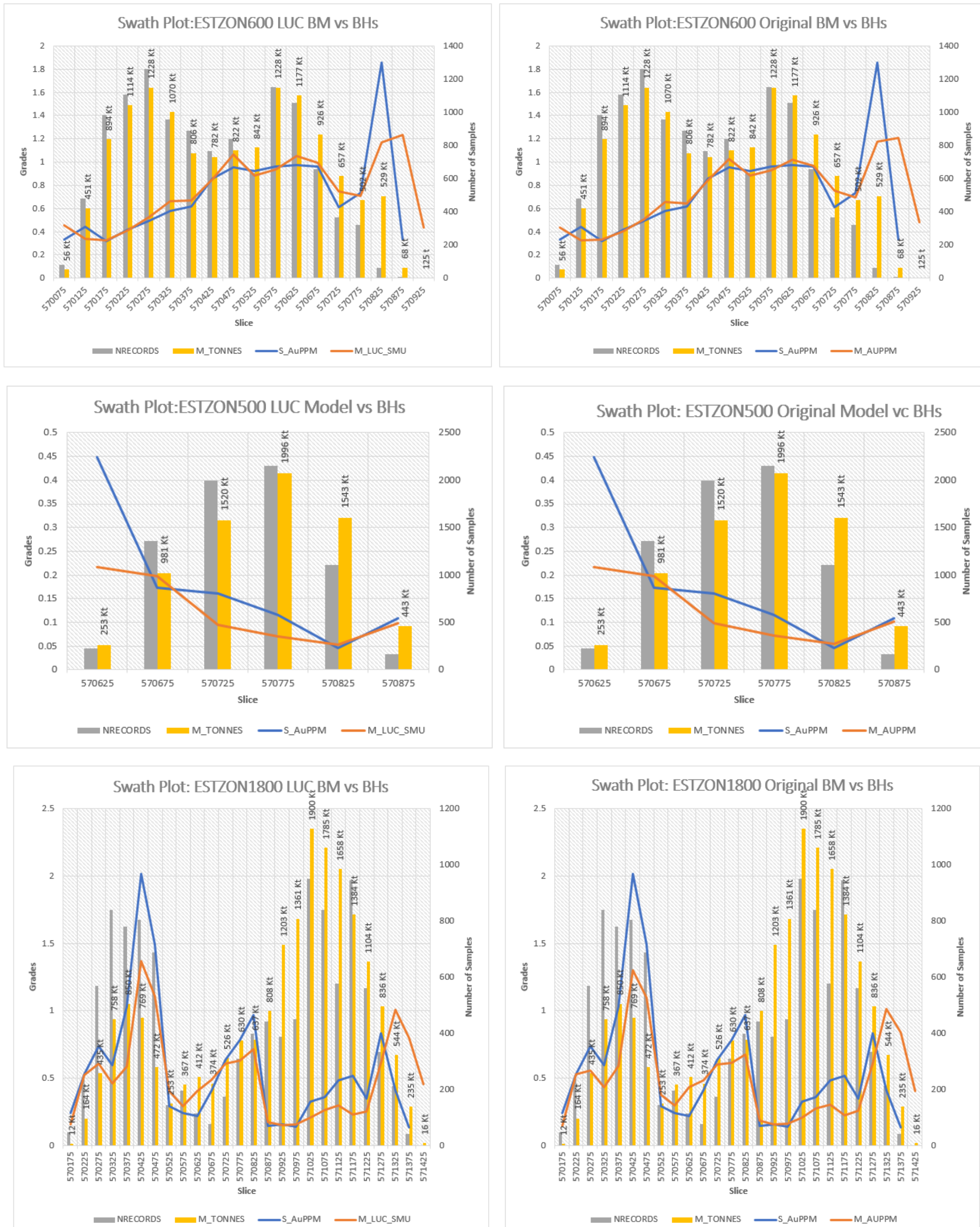
Table 14.30 – Statistical Validation by Comparison of Composites and Blocks Grade Means

Deposit	Description	Variable	Count	Minimum	Maximum	Mean	SD	CV
Bigar Hill	Composites Native	AUC	38,699	0.00	20.00	0.50	1.27	2.56
	Composites Declustered	AUC	38,699	0.00	9.26	0.48	0.70	1.47
	Model	AU	383,466	0.00	18.89	0.48	0.92	1.91
Korkan	Composites Native	AUC	33,306	0.00	25.00	0.43	1.30	3.02
	Composites Declustered	AUC	33,306	0.00	13.44	0.40	0.73	1.81
	Model	AU	373,153	0.00	22.71	0.38	0.79	2.10
Korkan West	Composites Native	AUC	4,215	0.00	7.00	0.49	0.85	1.74
	Composites Declustered	AUC	4,215	0.00	7.00	0.42	0.66	1.55
	Model	AU	46,833	0.00	5.70	0.41	0.59	1.44

14.16.3 SWATH PLOTS

Figure 14.31 shows examples of Swath Plots of Drill Holes versus the Original Panel estimate and the LUC estimate. The swathe plots indicate that the LUC model is sufficiently honouring the local grade fluctuations.

Figure 14.31 – Swath Plots of Drill Holes vs Original Panel Model

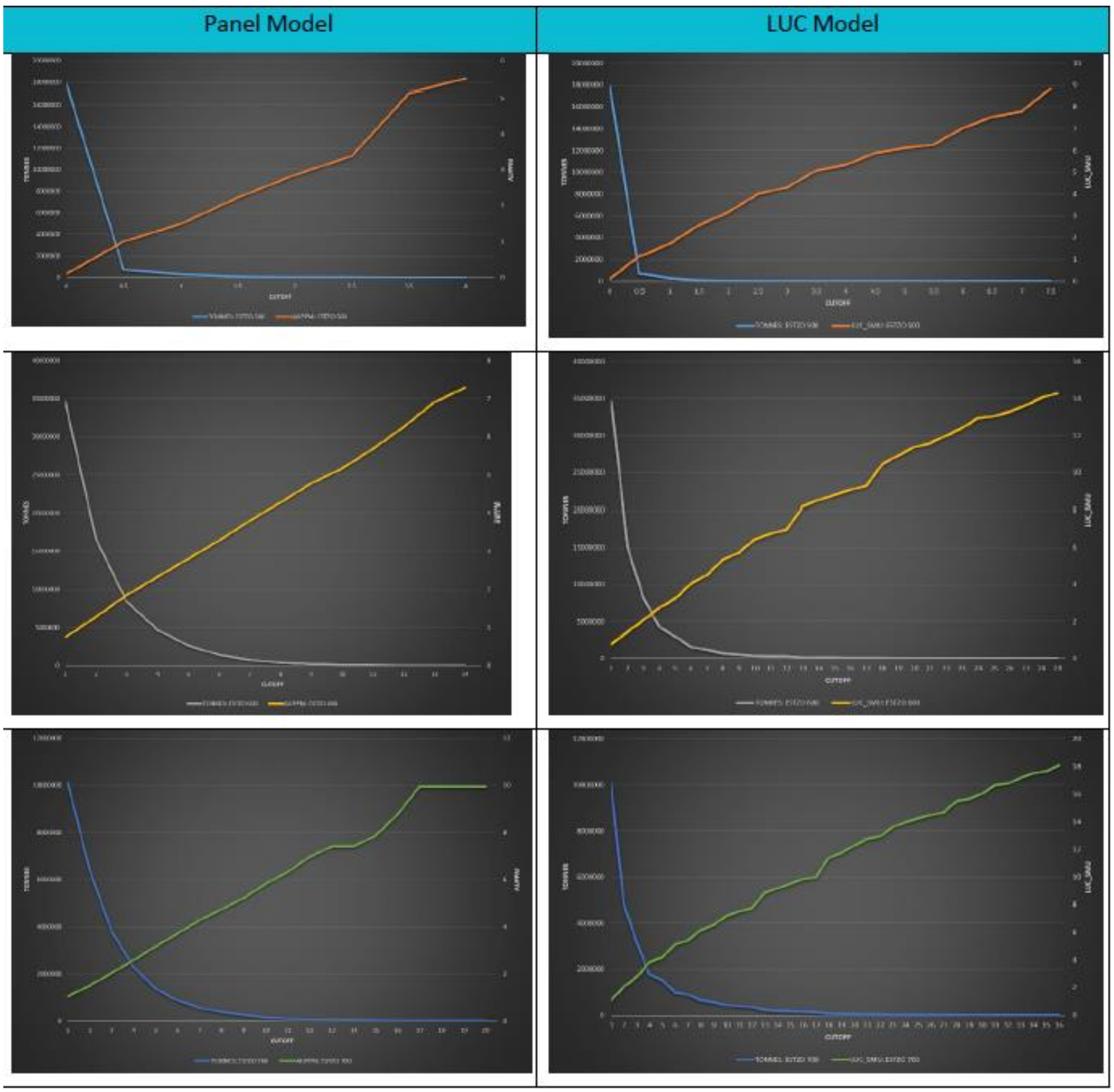


(Source: Datamine 2020)

14.16.4 GRADE TONNAGE CURVES

Figure 14.32 presents grade tonnage curves comparing the panel and LUC estimates. The total tonnage of the original (Panel) model and LUC model was identical for all domains. As one would expect, the LUC grade tonnage curves indicate more material at higher cut-offs due to the smaller block size (SMU) resulting in less smoothing of the estimate when compared to the large panels of the original block model.

Figure 14.32 – Example of Grade Tonnage Curves for the Panel Model and LUC



(Source: Datamine 2020)

14.16.5 HERCO Analysis

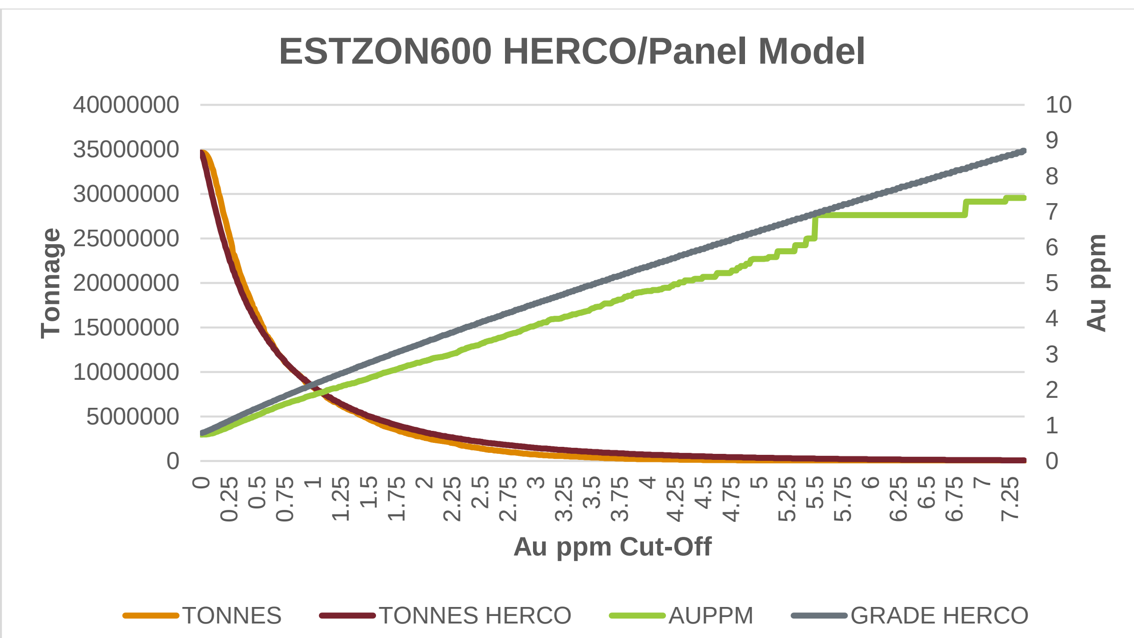
To further verify the global performance of the estimates, a series of comparative grade-tonnage curves were prepared. Using discrete Gaussian variogram models and Hermite Polynomials, a theoretical grade-tonnage curve for a specified block sizes can be obtained. These grade-tonnage curves are treated as the "ideal" GT curves that is attainable and also known as a HERCO diagram. The HERCO analysis was conducted by comparing the following:

- The 20×20×10 m original block model with the 20×20×10 m HERCO;
- The 5×5×5 m LUC block model with the 5×5×5 m HERCO.

Figure 14.33 shows an example of GT curves comparing HERCO tonnes to panel Model tonnes and panel Au grades with theoretical HERCO grades for 20×20×10 m blocks. Whilst Figure 14.34 shows an example of GT curves comparing HERCO tonnes to LUC Model tonnes and LUC Au grades with theoretical HERCO grades for 5×5×5 m blocks. Overall tonnages are observed to be similar but as expected, panel model grades using larger blocks are more smoothed compared to the HERCO theoretical curve. This is thought to be attributed to the estimate parameters, i.e., use of octants, and maximum sample numbers, applied during estimation of panel grades.

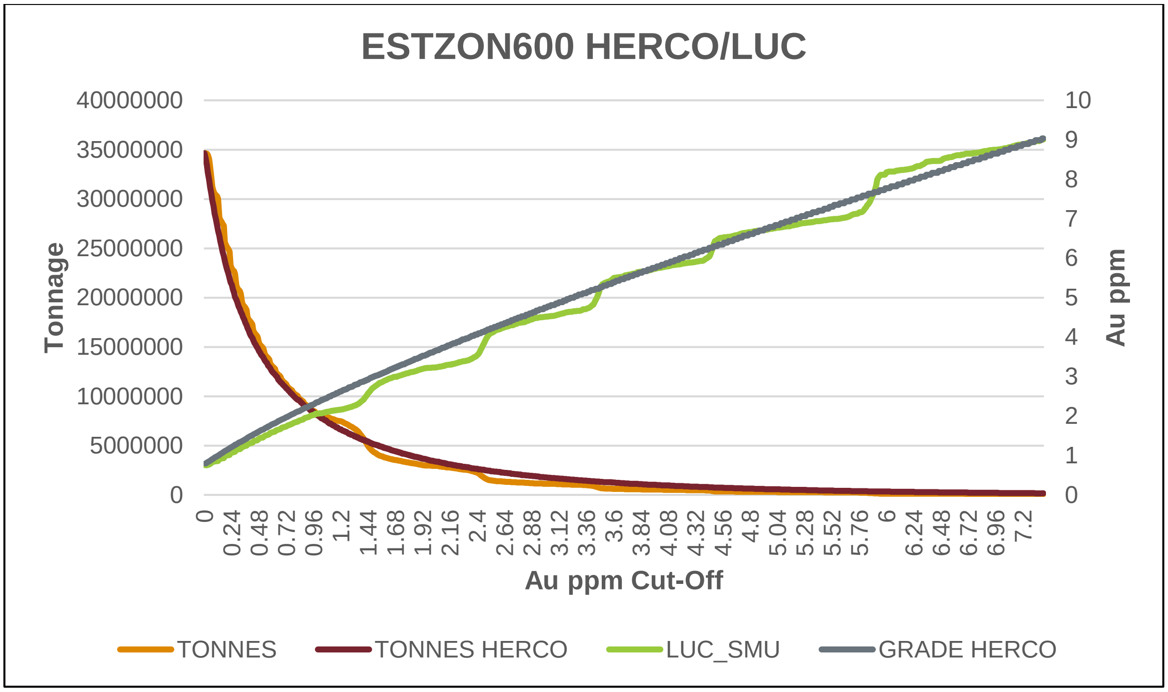
This smoothing effect is consistently lower when comparing the LUC grades to the HERCO theoretical grades for smaller blocks. When comparing HERECO and LUC estimates, the resultant grade tonnage curves are consistently closely aligned, particularly at the lower cut-offs, which are of most interest to the Project.

Figure 14.33 –GT Curves Comparing HERCO output and the Panel Block Model



(Source: Datamine 2020)

Figure 14.34 – GT Curves Comparing HERCO Output and the LUC Block Model



(Source: Datamine 2020)

14.17 Mineral Resource Classification

Mineral Resource estimates for the Project have been classified in accordance with CIM Definition Standards on Mineral Resources and Mineral Reserves (May 10, 2014).

The Mineral Resources classification is based on confidence in the continuity of geology and grades which in most cases, is related to the drilling density. Areas that are more densely drilled are usually better known and understood than areas with sparser drilling that are considered to have a lower confidence level. However, in certain cases, even tight drilling patterns may not allow for certainty of grade continuity. This can particularly be the case for structurally complex deposits or for deposits showing high variability on grades and/or high nugget effect.

The QP has considered the following factors for the Mineral Resources classification of the Project:

- The quality and reliability of the source data supplied by DPM. The QP undertook a data verification exercise whereby PDF laboratory certificates were compared with database outputs for about 8% of the drill holes;
- The confidence in the geological interpretations and the interpretation of mineralised units and oxide units. DRA has reviewed the different wireframes modelled by DPM and was confident that they reflect the reality of the geology, mineralisation and oxidation domains;

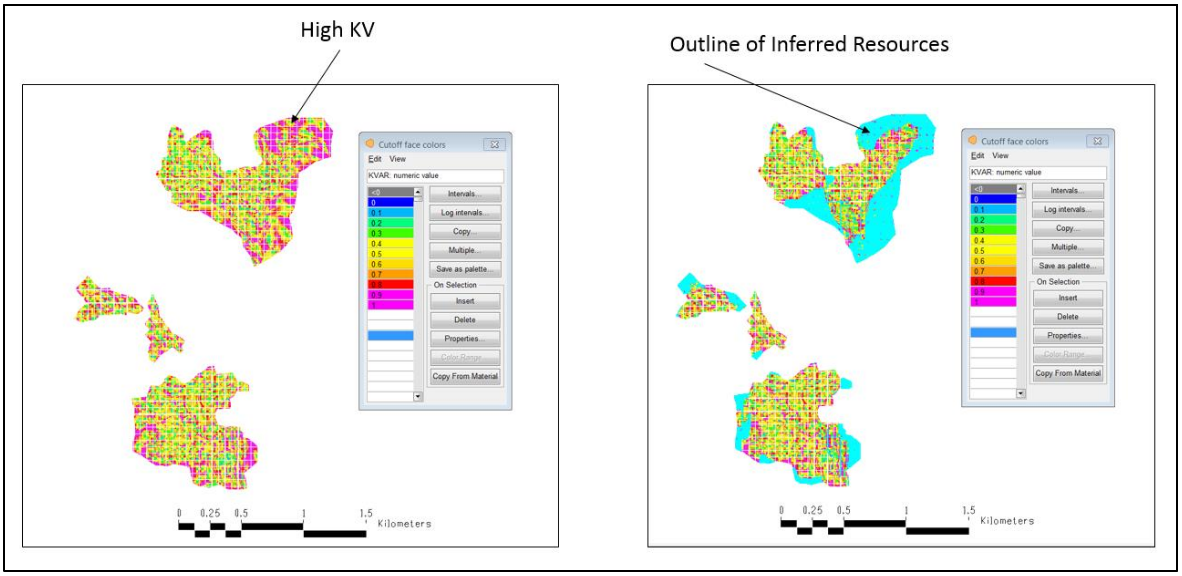
- The observed grade continuity noted during 2D and 3D visualisations and through geostatistical analyses;
- The implementation of rigorous internal QA/QC procedures and review of their results;
- The average drill spacing of a least 40 m × 40 m over the three deposits; and
- The fact that the current MRE is an update from a last estimate performed in 2018 by CSA which the QP has reviewed and deemed to have met accepted industry standard.

The Mineral Resource classification, undertaken by the QP, was conducted by reviewing the 2018 classification outlines plotted against the new estimated block model and additional drillhole information collected since the previous estimate. New classification polygons were then drawn on a 2D basis to delineate outlines of Indicated Mineral Resources for each deposit. These polygons were then joined to each other and built into a resultant 3D wireframe, which constrained the Indicated category blocks for each deposit. Blocks falling outside the constructed Indicated category shells were considered with lower confidence and classified as Inferred Mineral Resources Table 14.31 summarises criteria considered by the QP when classifying the Mineral Resources of the Project.

Table 14.31 – Criteria Classification

Resource Category	Criteria for Classification
Indicated Mineral Resource	Areas of high density of drilling information
	Areas with at least a drill spacing of 40 m × 40 m
	Areas of good grade continuity (Stable Variograms)
	Areas coded during the first interpolation pass
	Areas with low to moderate Kriging Variance
Inferred Mineral Resource	Blocks within mineralised shells but not coded as Indicated
	Correspond to peripheral areas with sparse drilling and higher KV (KV ≥ 0.8) - See Figure 14.35 – Left - Image showing blocks with high Kriging Variance (KV ≥ 0.8) on the Peripheries of the MRE

Figure 14.35 – Left - Image showing blocks with high Kriging Variance (KV ≥ 0.8) on the Peripheries of the MRE



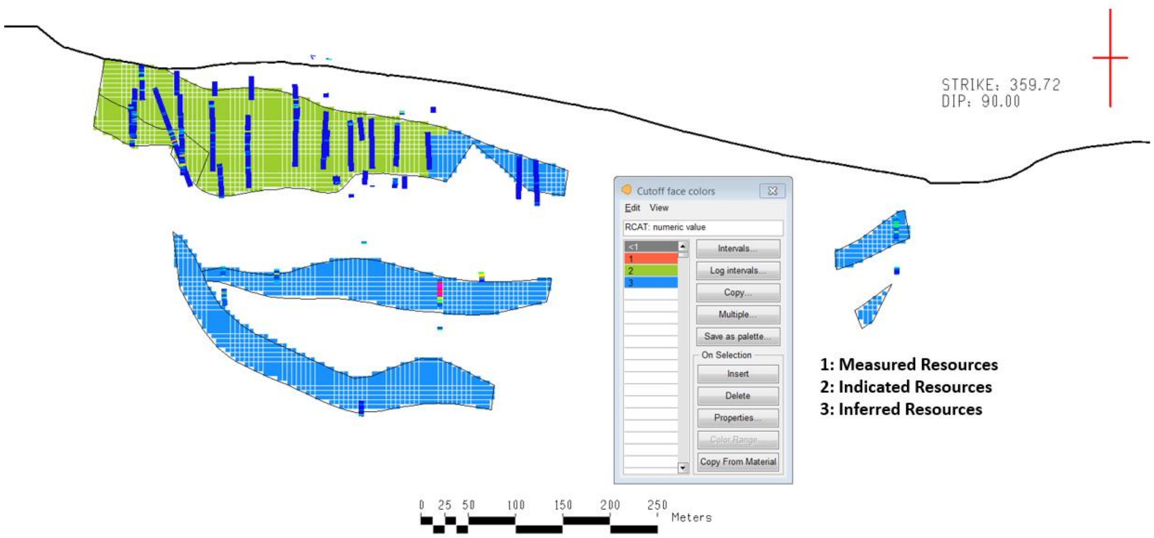
(DRA, 2020) Right - Resultant outlines of blocks classified as Inferred Mineral Resources.

The main objective of drilling campaigns undertaken since the last MRE (CSA, 2018) was to convert Inferred Mineral Resources into Indicated Mineral Resources. Infill and extensional drilling in these areas encountered weaker mineralisation than expected which in turn truncated Mineral Resource domains compared to the previous models. The consequences of this is a volumetric reduction (in the range of -3% to -6%) between the 2018 and 2020 mineralised shells.

Figure 14.36 to Figure 14.38 show examples of sections with classified blocks along with composite data respectively for Bigar Hill, Korkan, and Korkan West.

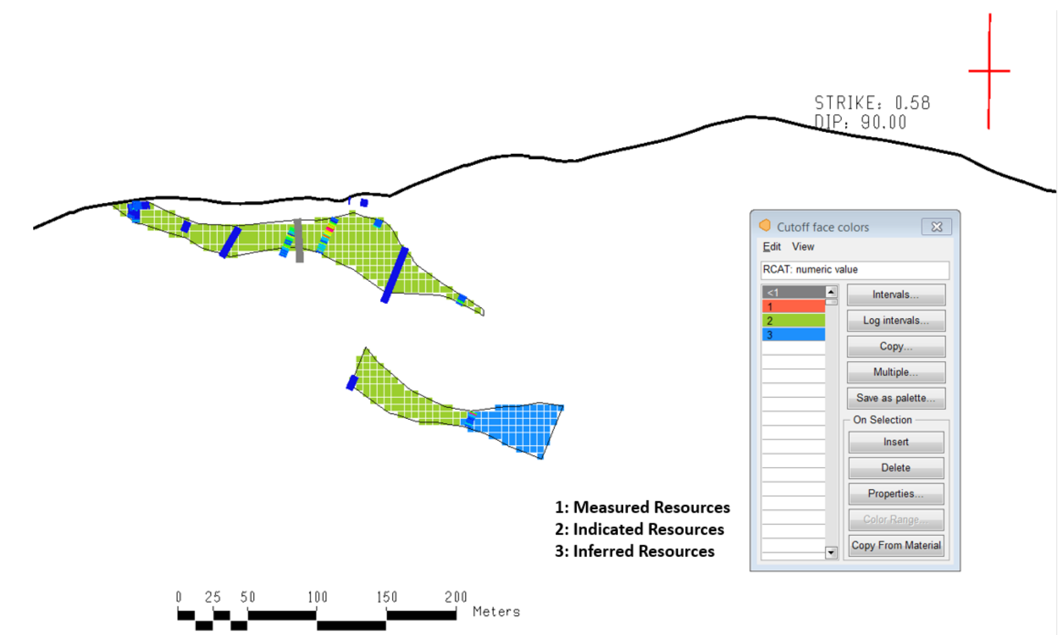
No blocks were classified within the Measured Category in any of the three (3) deposits.

Figure 14.36 – Bigar Hill – Example of Cross Section Showing Classified Blocks



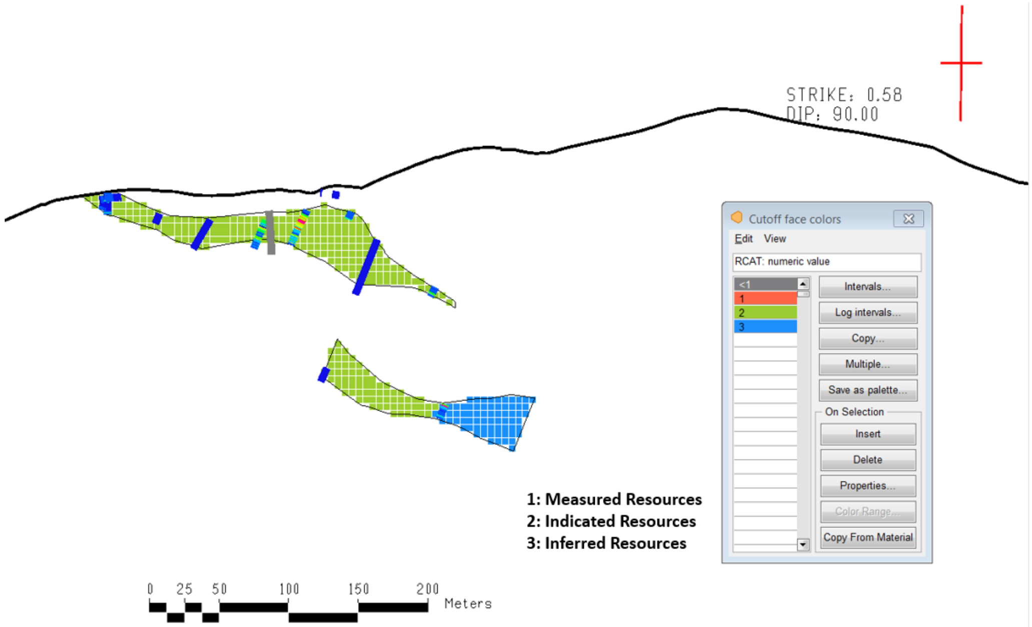
(Source: DRA Global 2020)

Figure 14.37 – Korkan West – Example of Cross Section Showing Classified Blocks



(Source: DRA Global 2020)

Figure 14.38 – Korkan West – Example of Cross Section Showing Classified Blocks (



(Source: DRA Global 2020)

14.18 Kraku Pester Mineral Resource Estimates

14.18.1 Introduction

CSA Global has previously reported an MRE update for the Kraku Pester deposit dated 7 November 2018 (CSA Global, 2018). The 2018 MRE remains current for Kraku Pester and a summary of all relevant methodology, parameters and key assumptions regarding the preparation of the updated MRE is reported below. Readers are referred to the 2018 Technical Report (CSA Global, 2018) for full details of the MRE. Since the 2018 MRE, no further exploration works have been undertaken at the Kraku Pester prospect.

14.18.2 2018 Mineral Resource Update

The MRE was based on interpretations using integrated geological and grade information recorded from RC and diamond core logging and assaying. DPM geologists conducted the geological interpretation and modelling work using the Leapfrog software package. CSA Global reviewed these models and found them suitable for use in the MRE. The estimation work was completed by CSA Global using the Datamine Studio and Isatis software packages. The date of receipt of final data for the Kraku Pester deposit was 15 May 2018, which is considered the effective date of the MRE.

The deposit has been evaluated regarding the UTM grid (Zone 34 North in WGS 84 datum), and all directional references in the MRE portions of this report are according to this grid.

Solid wireframes were created to represent the geological units at Kraku Pester which includes models of Overburden, Andesite Sill, Hornblende Diorite Porphyry, Skarn, Marl, Monzonite, Jurassic Limestone and Metamorphic Phyllite.

Suites of interpreted fault structures were defined as wireframe planes, and solid wireframes were created to represent sulphide (fresh – no oxidation), partial oxidation and complete oxidation.

Unlike the Bigar Hill and Korkan deposits, gold mineralisation at Kraku Pester is observed to dominantly locate within the sediments and monzonite units. Elevated gold values are concentrated along the contact between the sediments and the monzonite. The upper and lower boundaries of the mineralisation are in the sediments and the monzonite respectively. Both the upper and lower boundaries are variable and gradational in character, and there is a high proportion of very low grades within the broad zone of interpreted mineralised intercepts.

In view of both the gradational character of the mineralisation boundaries and particular requirements of the selected grade estimation method, it was considered more appropriate to apply looser constraints for the interpretation of mineralisation, compared to Bigar Hill and Korkan, but still within an indicative cut-off grade of 0.1 g/t Au, applied to 1 m composites.

A single mineralised shell volume was defined for Kraku Pester mineralisation, straddling the sediments/monzonite interface, striking roughly north-south with a near surface dip to the west of approximately 20°, steepening down dip to 50°. Mineralisation extends about 600 m east-west and 705 m north-south, and from near-surface to a depth of 450 m.

The stratigraphic, structural and mineralisation surfaces and solids were used as constraints in the construction of a cell model, based on parent cell XYZ dimensions of 20 m x 20 m x 10 m. To better represent the geometries of the mineralisation, cells were permitted to reduce to 5 m, 5 m, and 5 m in the X, Y and Z dimensions respectively. Models were coded to reflect the stratigraphic units and individual mineralised zones, as well as to distinguish between weathered and un-weathered material. Triangulated surfaces of topography were used to constrain the upper bounds of the models.

Drill-hole samples were coded by stratigraphic unit, mineralisation zone and weathering in a manner consistent with the cell model. The high coefficients of variation (CV) values within the individual mineralisation zones indicate highly skewed distributions with large grade ranges, or more than one population within a mineralised shell. These distribution characteristics are consistent with expectations given the loosely-domained mineralised shells based on a notional 0.1 g/t gold cut-off.

The mean gold grade for the single identified zone at Kraku Pester is low (0.31 g/t), reflecting the high proportion of very low grades captured within the mineralised shell.

Following statistical and visual review of grade distribution and continuity, selected mineralised geological units were combined into estimation domains. Samples were composited to 1 m lengths within these domains, which is the most common sample interval length.

Higher grades in the various domains represent relatively small proportions of each complete domain grade distributions and tend to be spatially discontinuous on a local scale, within more continuous trends of elevated grades, at larger scales.

Log probability plots and the spatial distribution of higher grades for each estimation domain were examined for high-grade outlier values. Top cutting was applied to various domains to reduce local high-grade bias due to very high-grade samples.

Statistical observations, along with visualisation of mineralisation characteristics, were used to guide the selection of grade estimation technique. OK was considered for the mineralised domains. However, within broad mineralisation zones (defined at approximately 0.1 g/t gold and within geological boundaries), the grade architecture at Kraku Pester is gradational rather than mosaic, i.e. there is a transition between high grades and low grades, rather than extremely sharp contacts, where Multiple Indicator Kriging may be suitable. As such, estimation of Mineral Resources with the potential for economic extraction based on a SMU of 5 m x 5 m x 5 m was completed using UC.

The UC estimate was further post-processed to produce single cell grades for each SMU, based on LUC where the grade tonnage of the panel gets reconstituted in SMU sized blocks resulting in a block model with single grades. The location of the high and low grades in each panel is an estimate based on the spatial distribution of high- and low-grade samples within the panel, but exact locations of the SMUs remain unknown.

Experimental variograms were generated and modelled based on 1 m gold composites within the defined estimation domains. Traditional semi-variograms were used as the spatial model for this study, with variography completed using Supervisor software.

The UC method required the estimation, by OK, of gold into 20 m × 20 m × 10 m panels, and gold into SMU-sized cells. Search ellipse orientations were consistent with local stratigraphic trends.

Post-processing of the panel estimates was applied to account for change of support, and the kriged SMU estimates were used to guide the distribution of panel estimates into an SMU-sized cell.

A review of the database of bulk density determinations at Kraku Pester showed that the variation in densities between lithological units and mineralised zones is low. In view of the large number of density values available, bulk density estimates were interpolated into the cell models by inverse distance squared weighting, subject to stratigraphic search constraints.

Procedures for classifying the reported "Mineral Resource", "Inferred Mineral Resource" and "Indicated Mineral Resource" were undertaken under the guidelines adopted by CIM, as the CIM Definition Standards on Mineral Resources and Mineral Reserves.

- Mineral Resources estimated have been classified with consideration of the following criteria:
  - Quality and reliability of raw data (sampling, assaying, surveying).
  - Confidence in the geological interpretation.
  - Number, spacing, and orientation of drill-hole intercepts through mineralised zones.
  - Knowledge of grade continuities gained from observations and geostatistical analyses.
  - The likelihood of material meeting economic mining constraints over a range of reasonable future scenarios, and expectations of relatively low selectivity of mining.
  - The Kraku Pester deposit has been classified as containing dominantly Indicated Mineral Resources with subsidiary Inferred Mineral Resources.

CSA Global completed the following validation checks on the MRE:

- Swath plots depicting model tonnes, input de-clustered composite gold grade, output block model gold grade and drill metres per slice for each domain of each deposit for the purposes of comparing input and output grades and trends.
- On-screen visual comparisons of the block model grades (via LUC) for all domains.
- Statistical comparison between the input composite grades and output model grades globally and for all domains.
- Results of the validations of the MRE supports the use of the resource model to underpin mine planning work, once constrained by a pit using appropriate parameters.

The Mineral Resources are constrained within pit shells based on the parameters presented in Table 14.32.

Oxide and transitional mineralised material from the Project will be treated using conventional heap leaching technology. Additionally, the sulphide mineralised material will be processed by flotation to produce a saleable gold-bearing concentrate.

Table 14.32 – Parameters Used for Kraku Pester Pit Optimisation

Units				Kraku Pester
Costs	Mining Cost	Waste	$/t mined	2.45
		Feed (oxide and transitional)	$/t feed	2.45
		Feed (sulphide)	$/t feed	3.15
		Incremental cost per 10 m bench	$/t mined	0.045 from 480 RL
		Rehabilitation	$/t mined	0.09
		Feed haulage from Kraku Pester	$/t feed	3.5
	Processing and Administration	Feed (oxide and transitional)	$/t feed	6.22
		Feed (sulphide)	$/t feed	12.81
	Off-Site Costs	Feed (oxide and transitional)	$/tr oz	5
		Total concentrate and smelter cost (sulphide)	$/tr oz	200
		Royalty	%	5
Parameters	Mining Parameters	Mining recovery	%	95.00
		Dilution	%	0.00
	Au Processing Recovery	Feed (oxide)	%	72.8
		Feed (transitional)	%	69.3
		Feed (sulphide)	%	50
	Overall Slope Angle	Oxide zone	°	45
		Transitional and sulphide	°	52.5
Revenue		Price of gold	$/oz t	1,250 (RF=1). Pit shell at 1,400
		Payable for oxide and transitional	%	99
		Payable for sulphide	%	100.00
Analysis		Discount rate	%	7.50
		Grams in a troy ounce	g/oz t	31.1035
		Processing rate	Mtpa	2.0

The price adopted for this study is $1,250/oz of gold as a pit shell with revenue factor=1; however, the pit shell generated at $1,400/oz has been selected as a constraining shell for reporting Mineral Resources.

Pit optimisations run in Whittle software resulted in varying cut-off grades, dependent on oxidation state, per deposit (Table 14.33).

Table 14.33 – Mineral Resource Reporting Cut-Off Grades for Kraku Pester

Deposit	Cut-off grade (Au g/t)					
	Cut-off for oxide in Whittle	Cut-off for oxide rounded	Cut-off for transitional in Whittle	Cut-off for transitional rounded	Cut-off for sulphide in Whittle	Cut-off for sulphide rounded
Kraku Pester	0.351	0.35	0.369	0.40	1.065	1.05

Mineral Resources are reported constrained within conceptual pit optimisation shell for each deposit, for the purposes of demonstrating "reasonable prospects of eventual economic extraction", required for Mineral Resource disclosure. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The constraining open-pit shell has been determined via consideration of various cut-off grades for material types that were calculated based upon, among other things, the material type, haulage distance and recoveries derived from metallurgical testwork. The Mineral Resource statement for each deposit is presented in Table 14.34.

Table 14.34 – Mineral Resource Estimate for Kraku Pester, as at 15 May 2018

Deposit		Indicated Mineral Resource			Inferred Mineral Resource		
		Tonnage	Au		Tonnage	Au	
		(Mt)	(g/t)	koz	(Mt)	(g/t)	koz
Kraku Pester	Oxide	0.7	0.95	22	0.1	1.3	5
	Transitional	0.1	0.95	4	0.0	1.2	0
	Sulphide	1.5	2.01	95	0.0	1.8	0
GRAND TOTAL		**2.3**	**1.61**	**122**	**0.1**	**1.3**	**6**

Notes:

- The effective date of the MREs is 15 May 2018.
- Mineral Resources are reported in accordance with CIM guidelines.
- A cut-off of 0.35 g/t Au for the oxide material, 0.40 g/t Au for the transitional material, and 1.05 g/t Au for the sulphide material is applied at Kraku Pester.
- Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.
- Due to rounding, some columns or rows may not compute exactly as shown.
- The Mineral Resources are stated as in-situ dry tonnes. All figures are in metric tonnes.
- The models are reported above surfaces based on conceptual $1,400 gold price pit shells to support assumptions relating to reasonable prospects of eventual economic extraction.
- 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.

14.19 Mineral Resource Statement

Under CIM Definition Standards, Mineral Resources are that part of a mineral inventory that possess *"reasonable prospects of eventual economic extraction"*. This requirement implies that the quantity and grade of estimate meet certain technical and economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade.

In order to demonstrate reasonable prospects of eventual economic extraction, a pit optimisation study using the Lerchs-Grossmann Algorithm was undertaken within GEOVIA Whittle® software. The operating parameters for the optimisations and cut-off grade estimates were based on reasonable technical (maximum pit slopes, gold recovery) and economical (gold price, unit mining and processing costs) parameters. The pit optimisation study considered both Indicated and Inferred blocks of the Mineral Resource inventory.

Reader should note that the overall slope angle parameters used for the three (3) deposits vary by lithology and type of oxidation. The overall slope angles are detailed in Section 16.3 – Geotechnical of the Report.

The Mineral Resources are constrained within pit shells based on the parameters presented in Table 14.35 and a summary of the MRE is provided in Table 14.36. Risks as related to Mineral Resource estimates are discussed in Section 25.2.

Table 14.35 – Parameters Used for Mineral Resource Constraining Pit Shell Optimisation

Description / Parameter	Unit	Bigar Hill	Korkan	Korkan West
Gold Price	$/oz	1,400		
Overall Slope angle - Oxide	degree	Refer to Section 16.3		
Overall Slope angle - Transitional and Sulphide	degree	Refer to Section 16.3		
Mining Recovery	%	95	95	95
Mining Dilution	%	0	0	0
Mining Cost - Waste	$/t mined	1.71	1.84	1.96
Mining Cost - Oxide and Transitional	$/t mineralisation	2.07	2.46	2.15
Mining Cost - Sulphide	$/t mineralisation	2.16	2.45	1.67
Gold Process Recovery – Oxide	%	85.03	85.03	85.03
Gold Process Recovery – Transitional	%	74.52	74.52	74.52
Gold Process Recovery – Sulphide	%	89.90	89.90	89.90
Process Cost - Oxide and Transitional	$/t processed	5.24	5.24	5.24
Process Cost - Sulphide	$/t processed	19.10	19.10	19.10
G&A Cost - Oxide and Transitional	$/t processed	1.60	1.60	1.60
G&A Cost - Sulphide	$/t processed	2.67	2.67	2.67
Metal Payable - Oxide, Transitional and Sulphide	%	99		

Description / Parameter	Unit	Bigar Hill	Korkan	Korkan West
Royalty	%	5		
Rehabilitation Cost	$/t waste	0.0207		
Process Rate - Heap Leach (Oxide and Transitional)	Mtpa	2.5		
Process Rate - Sulphide Concentrator	Mtpa	1.5		
Grams in a Troy Ounce	g/oz	31.1035		
Discount Rate	%	10		

Footnote:
Indicated Gold recoveries are based on PEA (CSA,2018) test work results as initial input to the PFS and does not reflect actual test results as described in Section 13.

Table 14.36 – Mineral Resource Estimate – May 29, 2020

Deposit	Ore Type	Indicated Mineral Resource						Inferred Mineral Resource					
		Tonne (Mt)	Au		Accessory Elements			Tonne (Mt)	Au		Accessory Elements		
			(g/t)	Koz	S (%)	Fe (%)	Ag (g/t)		(g/t)	Koz	S (%)	Fe (%)	Ag (g/t)
Bigar Hill	Oxide	5.2	0.79	132	0.17	2.44	1.0	0.0	0.69	0	0.25	3.62	0.46
	Transitional	6.2	0.94	186	0.32	1.45	1.0	0.0	0.94	1	0.29	3.89	0.62
	Sulphide	10.9	1.63	572	1.39	2.46	1.5	0.6	1.79	36	0.60	4.20	0.93
	Sub-total	***22.3***	***1.24***	***890***	***0.81***	***2.17***	***1.3***	***0.7***	***1.73***	***37***	***0.58***	***4.17***	***0.91***
Korkan	Oxide	2.2	0.70	49	0.15	1.61	1.2	0.0	0.42	0	0.18	2.31	5.45
	Transitional	2.0	0.78	50	0.26	0.67	1.1	0.1	0.54	1	0.67	1.07	0.33
	Sulphide	3.4	1.89	206	1.81	1.89	2.5	0.0	1.17	0	2.24	2.22	7.64
	Sub-total	***7.6***	***1.25***	***305***	***0.92***	***1.49***	***1.7***	***0.1***	***0.56***	***2***	***0.65***	***1.58***	***2.68***
Korkan West	Oxide	0.0	0.74	1	0.05	1.51	0.8						
	Transitional	0.0	0.64	0	0.08	1.11	0.3	0.0	0.15	0	0.15	0.48	0.15
	Sulphide	0.0	1.12	0	0.45	0.61	0.2						
	Sub-total	***0.1***	***0.71***	***2***	***0.06***	***1.37***	***0.6***	***0.0***	***0.15***	***0***	***0.15***	***0.48***	***0.15***
Sub total - Oxide		*7.4*	*0.76*	*182*	*0.16*	*2.19*	*1.0*	*0.0*	*0.49*	*1*	*0.20*	*2.63*	*4.23*
Sub total - Transitional		*8.2*	*0.90*	*237*	*0.30*	*1.25*	*1.0*	*0.1*	*0.65*	*2*	*0.48*	*2.01*	*0.41*
Sub total - Sulphide		*14.3*	*1.69*	*778*	*1.49*	*2.32*	*1.7*	*0.6*	*1.79*	*36*	*0.62*	*4.17*	*1.02*
TOTAL		**30.0**	**1.24**	**1,197**	**0.83**	**2.00**	**1.4**	**0.8**	**1.57**	**39**	**0.58**	**3.81**	**1.12**

Notes
1. The effective date of the MRE is May 29, 2020
2. Mineral Resources are reported in accordance with CIM Definition Standards
3. A cut-off of 0.19 g/t Au for the Oxide material, 0.216 g/t Au for the Transitional material, and 0.571 g/t Au for the Sulphide material is applied at Bigar Hill, Korkan, and Korkan West.
4. Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.
5. Due to rounding, some columns or rows may not compute exactly as shown.
6. The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.
7. The Mineral Resources are stated as in in situ gold ounces.
8. The models are reported above surfaces based on conceptual $1,400 gold price pit shells to support assumptions relating to reasonable prospects of eventual economic extraction.
9. 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.
10. Mineral Resources are reported exclusive of Mineral Reserves.

14.20 Consolidated Mineral Resource Statement

The overall consolidated Mineral Resource statement for the Project is shown in Table 14.37. Risks as related to Mineral Resource estimates are discussed in Section 25.2.

Table 14.37 – Consolidated Mineral Resource Estimate – May 29, 2020

Deposit	Ore Type	Indicated Mineral Resource			Inferred Mineral Resource		
		Tonne (Mt)	Au		Tonne (Mt)	Au	
			(g/t)	k oz		(g/t)	k oz
Bigar Hill	Oxide	5.2	0.79	132	0.0	0.69	0
	Transitional	6.2	0.94	186	0.0	0.94	1
	Sulphide	10.9	1.63	572	0.6	1.79	36
	Sub-total	*22.3*	*1.24*	*890*	*0.7*	*1.73*	*37*
Korkan	Oxide	2.2	0.70	49	0.0	0.42	0
	Transitional	2.0	0.78	50	0.1	0.54	1
	Sulphide	3.4	1.89	206	0.0	1.17	0
	Sub-total	*7.6*	*1.25*	*305*	*0.1*	*0.56*	*2*
Korkan West	Oxide	0.0	0.74	1			
	Transitional	0.0	0.64	0	0.0	0.15	0
	Sulphide	0.0	1.12	0			
	Sub-total	*0.1*	*0.71*	*2*	*0.0*	*0.15*	*0*
Kraku Pester	Oxide	0.7	0.95	22	0.1	1.3	5
	Transitional	0.1	0.95	4	0.0	1.2	0
	Sulphide	1.5	2.01	95	0.0	1.8	0
	Sub-total	*2.3*	*1.61*	*122*	*0.1*	*1.3*	*6*
	Total Oxide	*8.2*	*0.78*	*205*	*0.2*	*1.1*	*5*
	Total Transitional	*8.3*	*0.90*	*241*	*0.1*	*0.7*	*3*
	Total Sulphide	*15.8*	*1.72*	*873*	*0.6*	*1.8*	*37*
	Grand Total	**32.3**	**1.27**	**1,319**	**0.9**	**1.5**	**45**

Footnotes:

1. The effective date of the MRE for Bigar Hill, Korkan and Korkan West is May 29, 2020
2. The effective date of the MRE for the Kraku Pester Estimate is May 15, 2018
3. Mineral Resources are reported in accordance with CIM Definition Standards
4. A cut-off of 0.19 g/t Au for the Oxide material, 0.216 g/t Au for the Transitional material, and 0.571 g/t Au for the Sulphide material is applied at Bigar Hill, Korkan and Korkan West.
5. A cut-off of 0.35 g/t Au for the Oxide material, 0.40 g/t Au for the Transitional material, and 1.05 g/t Au for the Sulphide material is applied at Kraku Pester.
6. Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.
7. Due to rounding, some columns or rows may not compute exactly as shown.
8. The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.
9. The Mineral Resources are stated as in-situ gold ounces.
10. The models are reported above surfaces based on conceptual $1,400 gold price pit shells to support assumptions relating to reasonable prospects of eventual economic extraction.
11. 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.
12. Mineral Resources are reported exclusive of Mineral Reserves

14.21 Previous Mineral Resource Estimates

MRE for the Bigar Hill, Korkan, and Korkan West deposits were previously completed in May 2018, (CSA Global, 2018). The key changes between the 2018 MRE for the three (3) deposits (CSA, 2018) and this current updated MRE are:

- Additional drilling since 2018 update the MRE;
- Updated interpretations by DPM of the different mineralised and weathering domains based on the new information added;
- Mineral Resources are now reported exclusive of Mineral Reserves.

The 2018 MRE remains current for Kraku Pester.

15 MINERAL RESERVES ESTIMATE

15.1 Introduction

This section covers the conversion of the Mineral Resources to Mineral Reserves of the Project.

15.2 Mineral Resource Model Import and Preparation

The PFS was based on Mineral Resources for estimated by DRA with an effective date of May 29, 2020 which is detailed in Section 14 of this Report.

The HxGN Mine Plan Mineral Resource 3D block model was imported into Surpac Software where the geotechnical parameters described below and in Section 16 were integrated into the block model to ensure proper slopes were used in the economic pit limit analysis. In addition, incremental vertical mining costs were also added to the block model.

The Surpac 3D block model was then exported in a format that is directly read by Whittle software for pit optimisation.

15.3 Pit Optimisation

The open pit optimisation was conducted on the deposits to determine the economic pit limits. The optimisation was carried out using initial cost, sales price, and pit and plant operating parameters. The pit optimisation was re-evaluated after a preliminary mine plan was completed, and the cost, sales price, and pit and plant operating parameters were better defined. The pit optimisation was done using GEOVIA Whittle^{TM}. The optimiser operates on a net value calculation for all the blocks in the model (i.e., revenue from sales of gold bars less operating cost). The formula is presented below:

$$\boldsymbol{Gold\ Oz\ Produced} = \frac{Mineralized\ Tonnage \times Recovery \times Au\ Content}{31.1035}$$

$$\boldsymbol{Revenue} = \text{Gold oz produced} \times \text{Sales Price} \times \text{Payable Gold}$$

$$\boldsymbol{Net\ Value} = \text{Revenue} - (Mining\ Cost + Processing\ Cost + G\&A\ Cost + Rehab\ Cost + Royalty)$$

Only Measured and Indicated Mineral Resources from the 3D block model have been considered in the optimization and mine plan. Table 15.1 presents the pit optimisation parameters. The parameters were developed assuming a standard open pit truck and shovel operation, a heap leach process, and a production rate of 2.5 Mtpa of ore. Process recoveries and all production costs were defined or developed by DRA based on metallurgical testing and detail cost calculations. Only oxide and transitional mineral resources were considered for Mineral Reserves in the PFS based on results of process trade-off studies for the heap leach as described is Section 17 of the report. No Mineral Reserve estimates were performed for the Kraku Pester prospect since this prospect is comprised of predominantly primary sulphide mineralisation. The Mineral Reserves are reported at a gold price of $1,250 per oz as per DPM current Mineral Reserve reporting guidelines.

15.3.1 Pit Optimisation Parameters

Table 15.1 – Mineral Reserves Pit Optimisation Parameters

Description / Parameter	Unit	Bigar Hill	Korkan	Korkan West
Gold Price	$US/oz	1,250		
Overall Slope angles - Oxide	degree	Variable per pit and material type. Refer to Section 16.3 for details		
Mining Recovery	%	95	95	95
Mining Dilution	%	0	0	0
Mining Cost - Waste	$US/t mined	1.71	1.84	1.96
Mining Cost - Oxide and Transitional	$US/t ore	2.07	2.46	2.15
Incremental mining cost per 5m pit depth	$US/t (ore & waste)	0.008	0.008	0.008
Gold Process Recovery - Oxide	%	85.03	85.03	85.03
Gold Process Recovery - Transitional	%	74.52	74.52	74.52
Process Cost - Oxide and Transitional	$US/t processed	5.24	5.24	5.24
G&A Cost - Oxide and Transitional	$US/t processed	1.60	1.60	1.60
Metal Payable - Oxide and Transitional	%	99		
Royalty	%	5		
Rehabilitation Cost	$/t waste	0.0207		
Process Rate - Heap Leach (Oxide and Transitional)	Mtpa	2.5		
Grams in a Troy Ounce	g/oz	31.1035		
Discount Rate	%	10		

15.3.2 Gold Price

The selected gold price of $1,250/oz is based on DPM's current price for Mineral Reserve estimation for its operations and other mining projects. This value is considered conservative considering that gold is currently trading at $1,825/oz (as of February 28, 2021) and average three-year price (2018-2020) is close to $1,500/oz.

15.3.3 Overall Slope Angles

The slope angles used in the Whittle optimisations followed the recommended slope angles as defined by SRK and reviewed by DRA for each of the different pits and material types. Details are given in Section 16.3 of the report.

15.3.4 Mine Dilution and Mining Recovery

The Timok deposit is a massive oxide and transitional deposit and its economical ore zones are defined by gradual grade change rather than clear ore/waste geological boundary and therefor a recovery factor of 95% was applied to tonnage mined with no dilution.

15.3.5 Mining Costs

Mining Costs were derived from first principles calculation based on the mine plan developed in the PEA. DRA applied equipment productivities, unit operating costs, and manpower requirements to arrive at individual operating costs for each pit and material type. Incremental mining cost per bench were also estimated.

15.3.6 Gold Process Recovery

DRA process engineers developed gold process recovery factors that were used for the economic pit limit determination. The different recovery factors for each material type are based on extensive testing as detailed in Section 13 of the Report.

15.3.7 Process Cost

Process unit costs were estimated by DRA process engineers based on the process flowsheet for heap leach operation and production of gold doré bars and a production rate of 2.5 Mtpa of ore as presented in Section 17. Process costs were estimated for the different ore types.

15.3.8 G&A Costs

G&A costs were estimated by DRA based on an annual production of 2.5 Mt.

15.3.9 Payable Metal

The payable gold on the doré bars, after refinery charges is estimated at 99% of gold value.

15.3.10 CUT-OFF GRADE

The cut-off grade (COG) has been calculated according to the following formula:

$$COG = Gold\ grade\ \times \frac{Mining\ cost + Process\ Cost + G\&A}{Sales\ price \times Mill\ recovery}$$

$$Marginal\ COG = Gold\ grade\ \times \frac{Process\ Cost + G\&A}{Sales\ price \times Mill\ recovery}$$

Using the economic parameters presented in Table 15.1, a COG for each pit, and for each type of material (oxide or transitional), was calculated. Table 15.2 presents the COG result in each case.

Table 15.2 – COG Results

Parameters	Unit	Bigar Hill		Korkan		Korkan West	
Ore Type		Oxide	Transitional	Oxide	Transitional	Oxide	Transitional
Cut-off Grade	Au (g/t)	0.25	0.28	0.26	0.29	0.25	0.28
Marginal Cut-off	Au (g/t)	0.21	0.24	0.21	0.24	0.21	0.24

The COG is used to determine whether the material being mined will generate a profit after paying for the mining, processing and administrative costs and is the basis for defining the economic pit. The Marginal COG is used to identify after a material is mined if it has enough value to go to the plant and make a profit or if not, will go to the waste stockpile. Material that is mined below the Marginal COG is always sent to the waste stockpile. Material between the COG and Marginal COG can either be sent directly to the process plant or sent to a lower grade stockpile for future processing or blending.

15.3.11 PIT OPTIMISATION RESULTS

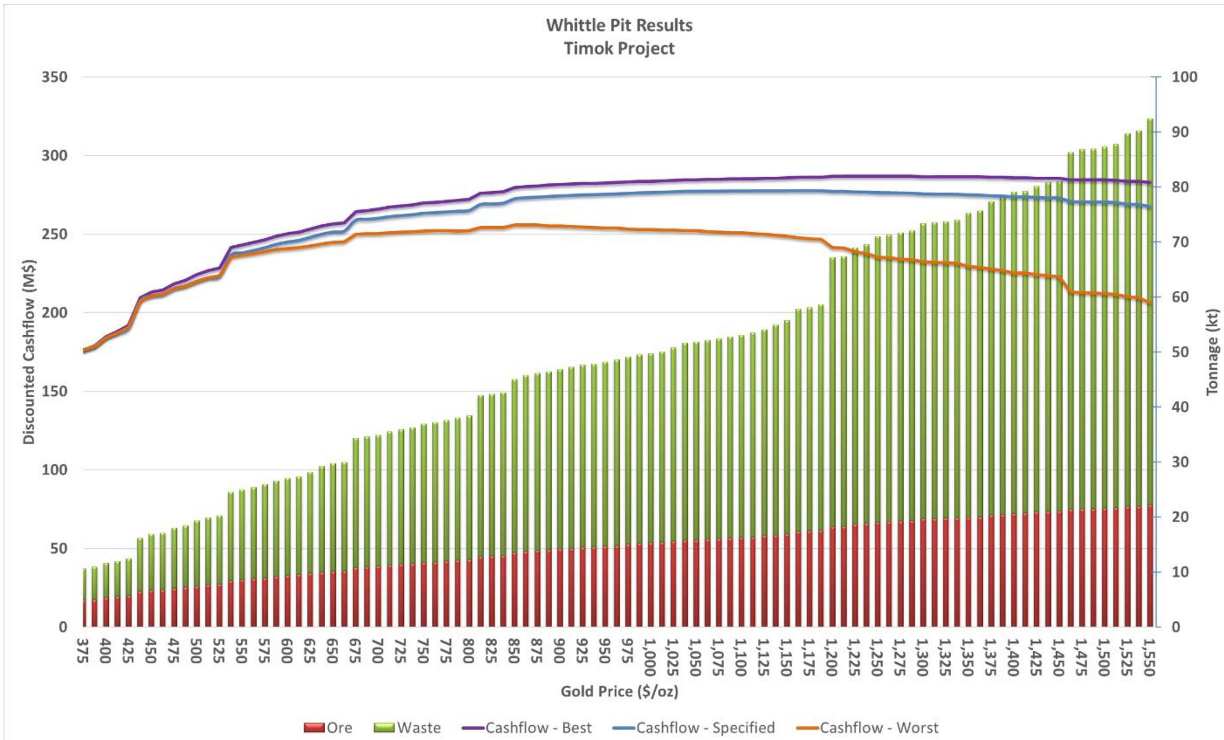
Optimal open pit mining limits were established using GEOVIA Whittle™, which uses the Lerchs-Grossman algorithm for pit optimization and the parameters indicated in Table 15.1. The NPV sensitivity analysis was used as the main criterion to select the optimal pit.

When varying gold sales price from $375/oz to $1,550/oz (Figure 15.1), the NPV increases gradually until the price reaches $1,125/oz. From this point on, the NPV decreases slightly as the costs associated with waste production exceed profits generated from gold sales. Pit limits were optimised for a base case price of $1,250 /oz. This optimisation takes in to account a heap leach production throughput of 2.5 Mtpa.

The pit optimisation has been performed for the Bigar Hill, Korkan, and Korkan West using the parameters as defined in Table 15.1.

An optimised pit shell containing 17.6 Mt of ore at an average Au grade of 1.16 g/t was selected (revenue factor 0.95 or $1,188/oz) which provides a mine life of approximately 7 years.

Figure 15.1 – Whittle Pit Optimisation Results



15.4 Pit Design

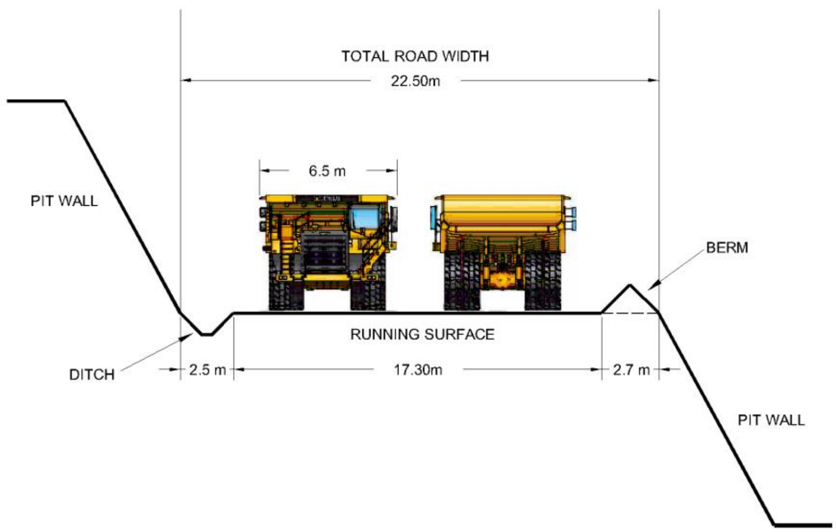
The next step in the Mineral Reserve estimation process is to design an operational pit that will form the basis of the production plan. This pit design uses the selected economic pit shells as a guideline and includes smoothing the pit walls, adding ramps to access the pit bottom and ensuring that the pit can be mined using the initially selected equipment. The pit designs were designed in Datamine Studio software based on the 3D Mineral Resource Block model provided by DRA as well as the Whittle pit shells as limits. The following section provides the parameters that were used for the open pit design and presents the results.

15.4.1 Haul Road Design

The ramps and haul roads were designed with an overall width of 22.5 m. For double lane traffic, industry practice indicates the running surface width to be a minimum of 2.5 times the width of the largest truck. The overall width of a 90-ton haulage truck is 6.5 m which results in a running surface width of approximately 16.2 m. The allowance for berms and ditches increases the overall haul road width to 22.5 m.

A maximum ramp grade of 8 to 10% was considered for the PFS. Figure 15.2 presents a typical section of the in-pit haul road/ramp design.

Figure 15.2 – Haul Road Design



15.4.2 Pit Slopes

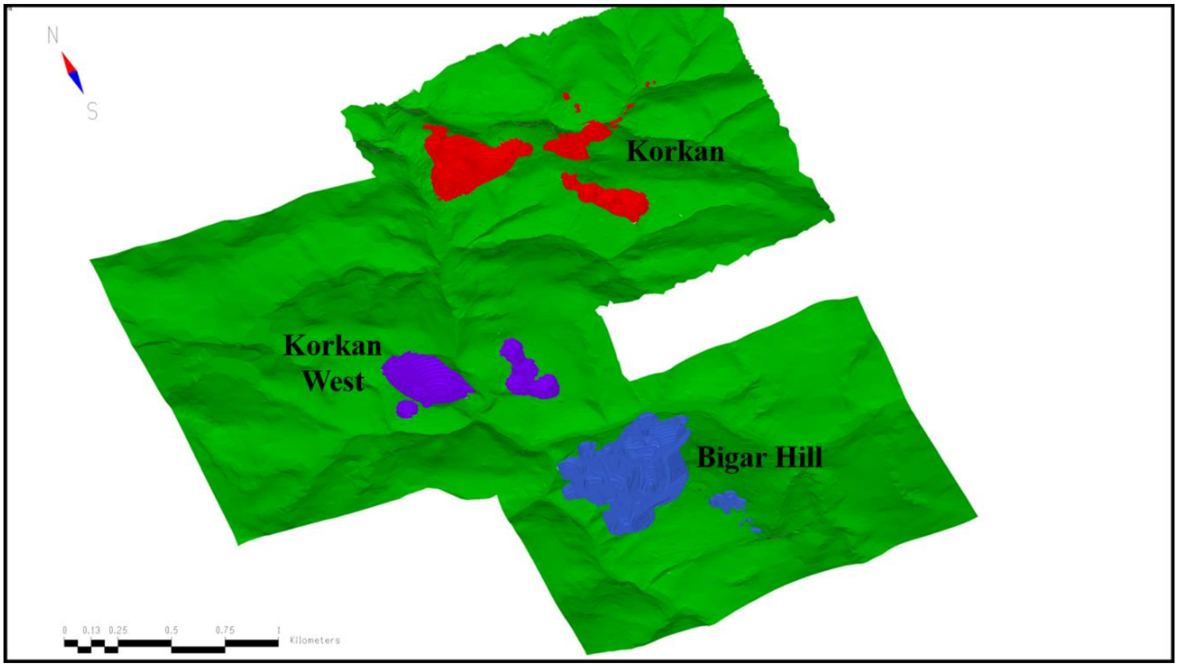
The pit designs followed the recommended geotechnical slopes for Bigar Hill, Korkan, and Korkan West deposits as described in Section 16.3 of the Report.

15.4.3 Open Pit Design Results

Figure 15.3 depicts the open pit design for the three (3) Timok Pits.

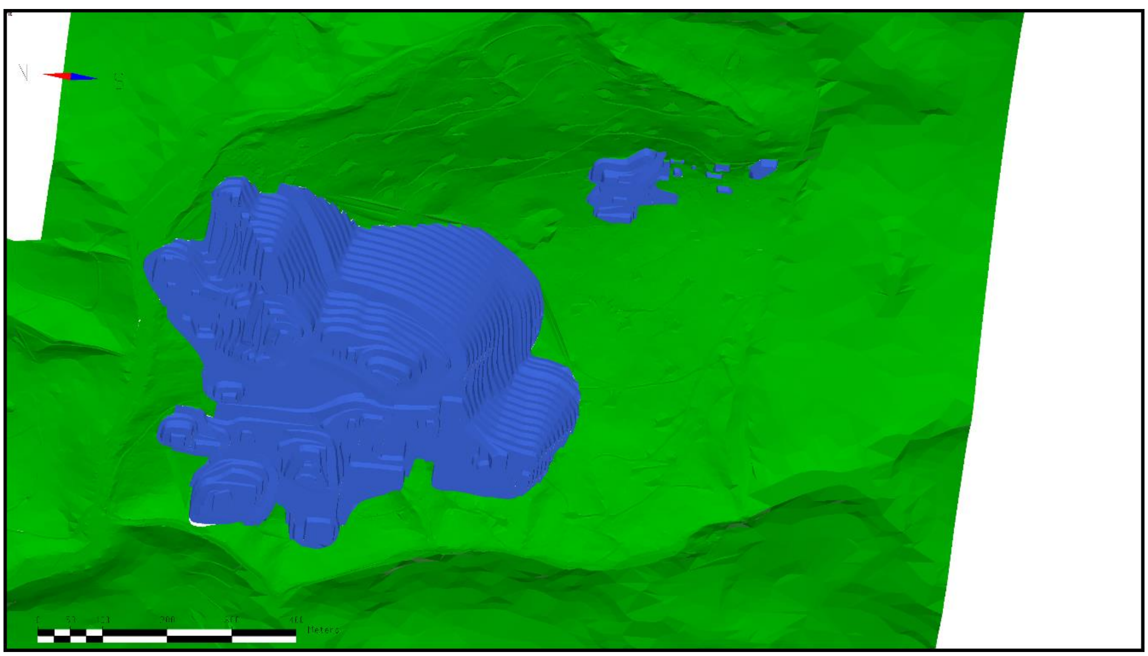
The pits designed for the Project are generally comprised of a larger main pit and satellite pits for each of the three (3) deposits. The pit designs for each deposit presented in Figures 15.4 to 15.6 were used for the estimation of Mineral Reserves. The final pit designs delineated 67.6 Mt of combined ore and waste compared to 58.6 Mt from the Whittle Pit shells. The difference is an increase in 15% in tonnage, 9% more in ore, and 17% more in waste due to following the geotechnical pit slope parameters, including the catch benches and to the inclusion of ramps and practical mining areas.

Figure 15.3 – Timok Pits



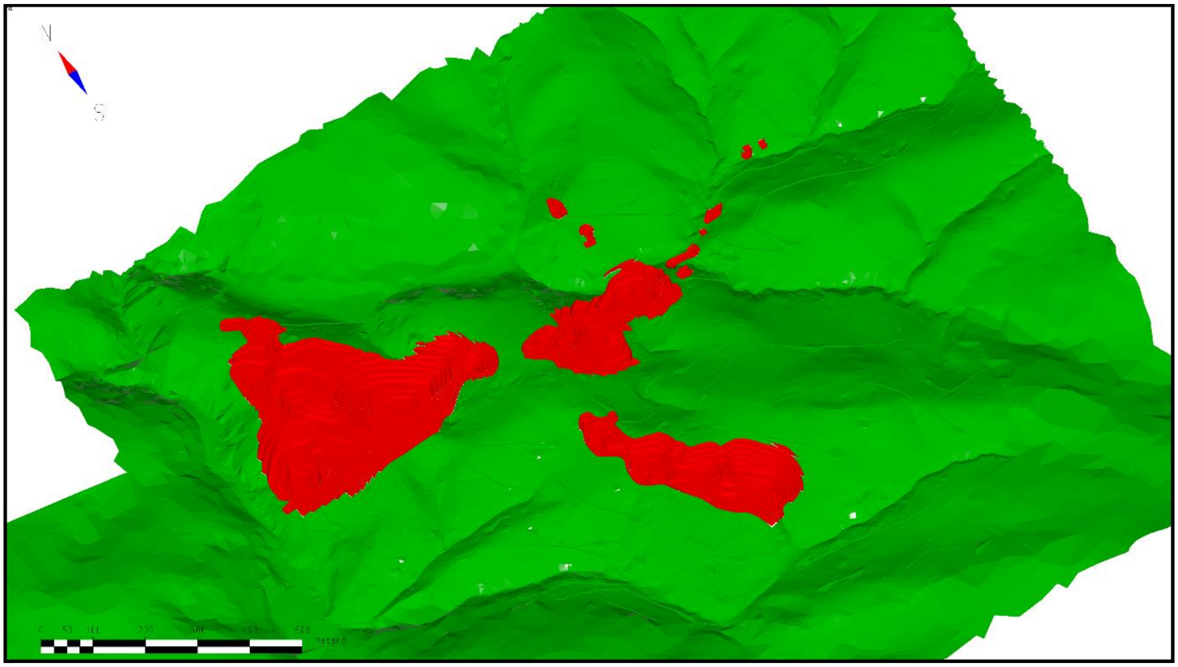
Source: DRA, 2020

Figure 15.4 – Bigar Hill Pits



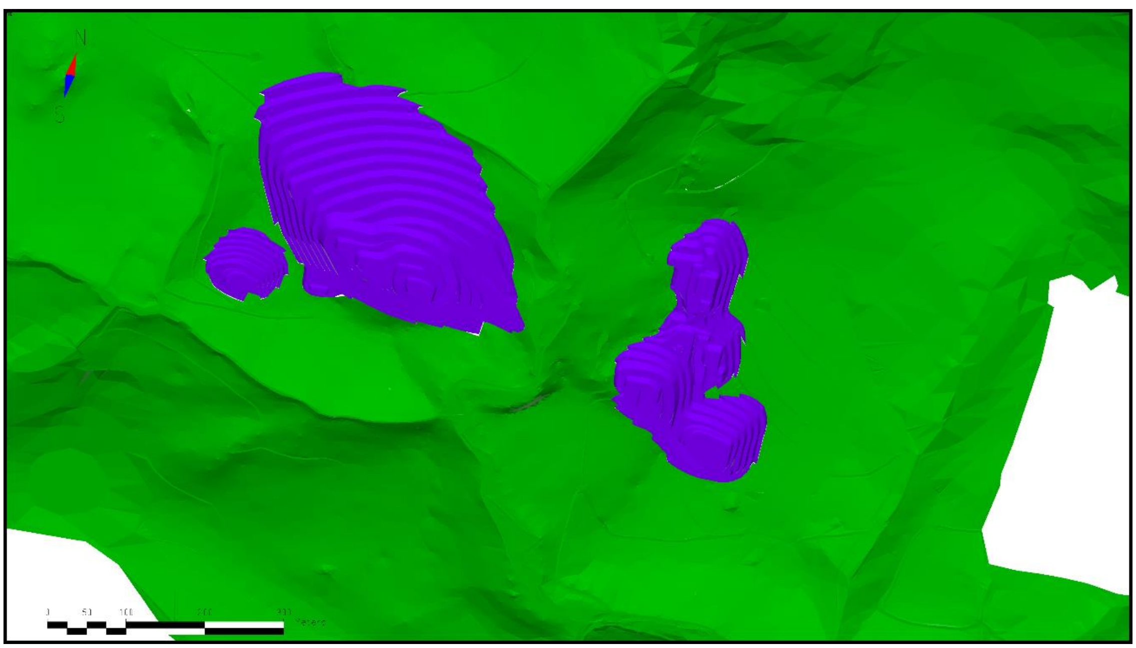
Source: DRA, 2020

Figure15.5 – Korkan Pits



Source: DRA, 2020

Figure 15.6 – Korkan West Pits



Source: DRA, 2020

15.5 Mineral Reserves

The Mineral Reserves for the Project are estimated at 19.2 Mt of Probable Mineral Reserves at a grade of 1.07 g/t Au based on the marginal cut-off grades of 0.21 g/t for oxide ore and 0.24 g/t for transitional ore. In order to access these reserves, 48.3 Mt of waste rock will need to be removed. This results in a stripping ratio of 2.52 to 1 (waste/ore). Table 15.3 presents the open pit Mineral Reserves for the Project. No Mineral Reserves have been estimated for the Kraku Pester prospect since this prospect is comprised predominantly of primary sulphide mineralisation.

Table 15.3 – Mineral Reserves Estimate by Pit – Effective May 29, 2020

Deposit	Ore Type	Probable Reserves Tonnes (Mt)	Au Grade (g/t)	Mined Au ounces (k oz.)	Strip Ratio (Waste/Ore)
Bigar Hill	Oxide	8.8	1.19	334	
	Transitional	1.9	1.09	67	
	Sub total	**10.7**	**1.17**	**401**	**2.85**
Korkan	Oxide	3.4	0.90	97	
	Transitional	1.2	1.02	39	
	Sub total	**4.6**	**0.93**	**137**	**2.69**
Korkan West	Oxide	3.7	0.99	118	
	Transitional	0.3	0.74	6	
	Sub total	**4.0**	**0.97**	**124**	**1.42**
Total	Oxide	15.8	1.08	549	
	Transitional	3.4	1.04	110	
	Total	**19.2**	**1.07**	**662**	**2.52**

Footnotes:
1. The effective date of the M The effective date of the Mineral Reserve Estimate. is May 29, 2020.
2. Mineral Reserves are reported in accordance with CIM guidelines.
3. A marginal cut-off of 0.21 g/t Au for the Oxide material, and 0.24 g/t for the Transitional material is applied at all deposits.
4. Mineral Reserves were estimated at a gold price of $1,250 per oz and include modifying factors related to mining cost, and dilution and recovery, process recoveries and costs, G&A, royalties and rehabilitation costs.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows may not compute exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher. All figures are in metric tonnes.

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource, included diluting materials. A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applied to a Probable Mineral Reserve is lower than that applied to a Proven Mineral Reserve. A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors. Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

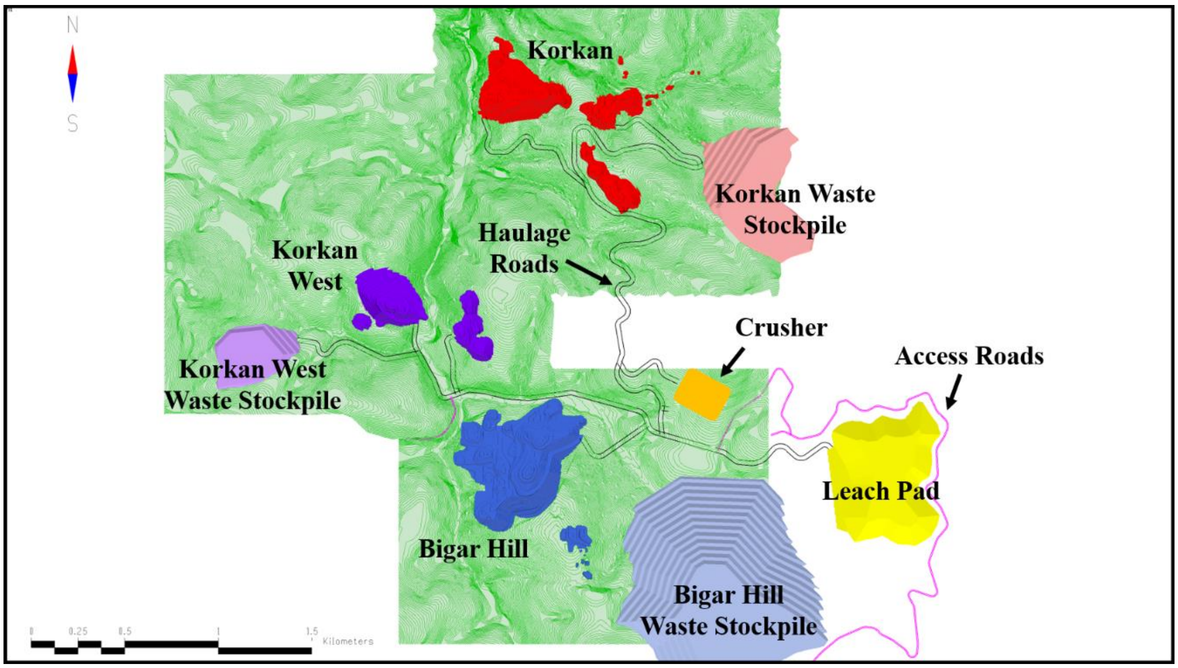
16 MINING METHOD

16.1 Mining Operation

The mining method selected for the Project is a conventional open pit operation with rigid body mining trucks, hydraulic excavators, and wheel loaders.

The Project consists of three (3) separate mining areas: the Bigar Hill pits to the South, the Korkan West pits to the West, and the Korkan pits to the North. The ore will be transported by truck from the pits to the crusher located east of Bigar Hill. Waste material will be deposited in waste piles close to each pit. Mineralised sulphide material above the cut-off grade will be also stored in separate stockpiles in the waste stockpile areas for potential future reclaim and treatment. Figure 16.1 shows the location of the different pits, stockpiles, crusher/process plant and heap leach pad.

Figure 16.1 – Timok Project Layout



A trade-off study was completed for three (3) operating scenarios: 60 t trucks and related fleet, 90 t trucks and related fleet, and contractor mining using 36 t trucks and related fleet. Based on the results of the trade-off study, the case with the 60 t trucks and related fleet was chosen.

The ore and waste rock will be loaded into haul trucks with excavators and/or wheel loader. The ore is a combination of oxide material and transitional material with different process recoveries. There is no need for specific blending of the oxide and transitional minerals before being sent to the crusher and leach pad and therefore, no need for an ore stockpile near the crusher. Marginal ore (above the marginal cut-off grade but below the mining cut-off grade) will be mined and stockpiled near each pit for transport and crushing at the end of the mine life.

The mine will operate year-round, seven (7) days per week, twenty-four (24) hours per day (2 – 12-hour shifts). Two (2) weeks of adverse weather conditions per year are considered in the mine plan.

16.2 Mine Design

Mineral Reserves were estimated for the Bigar Hill, Korkan, and Korkan West pits based on the economic and pit design parameters detailed in Section 15. The total tonnage to be mined from these pits is estimated at 67.6 Mt, ore and waste combined. This material will be mined over a 7-year period.

16.2.1 Pit Designs

The final pits designed for the Bigar Hill, Korkan, and Korkan West pits follow the recommended geotechnical parameters and domains defined in Section 16.3 and the material types in the DRA mineral resource models. Detailed tonnages by ore type and by pit is presented in Table 16.1. Figures 16.2 to 16.4 present the final pit designs for each deposit.

Table 16.1 – Timok Reserve by Pit

Pit	Total Ore		Waste	Strip Ratio
	kt	g/t	kt	w/o
Bigar Hill	10,683	1.17	30,426	2.85
Korkan	4,567	0.93	12,307	2.69
Korkan West	3,964	0.97	5,612	1.42
Total	**19,214**	**1.07**	**48,345**	**2.52**

Figure 16.2 – Bigar Hill Pits

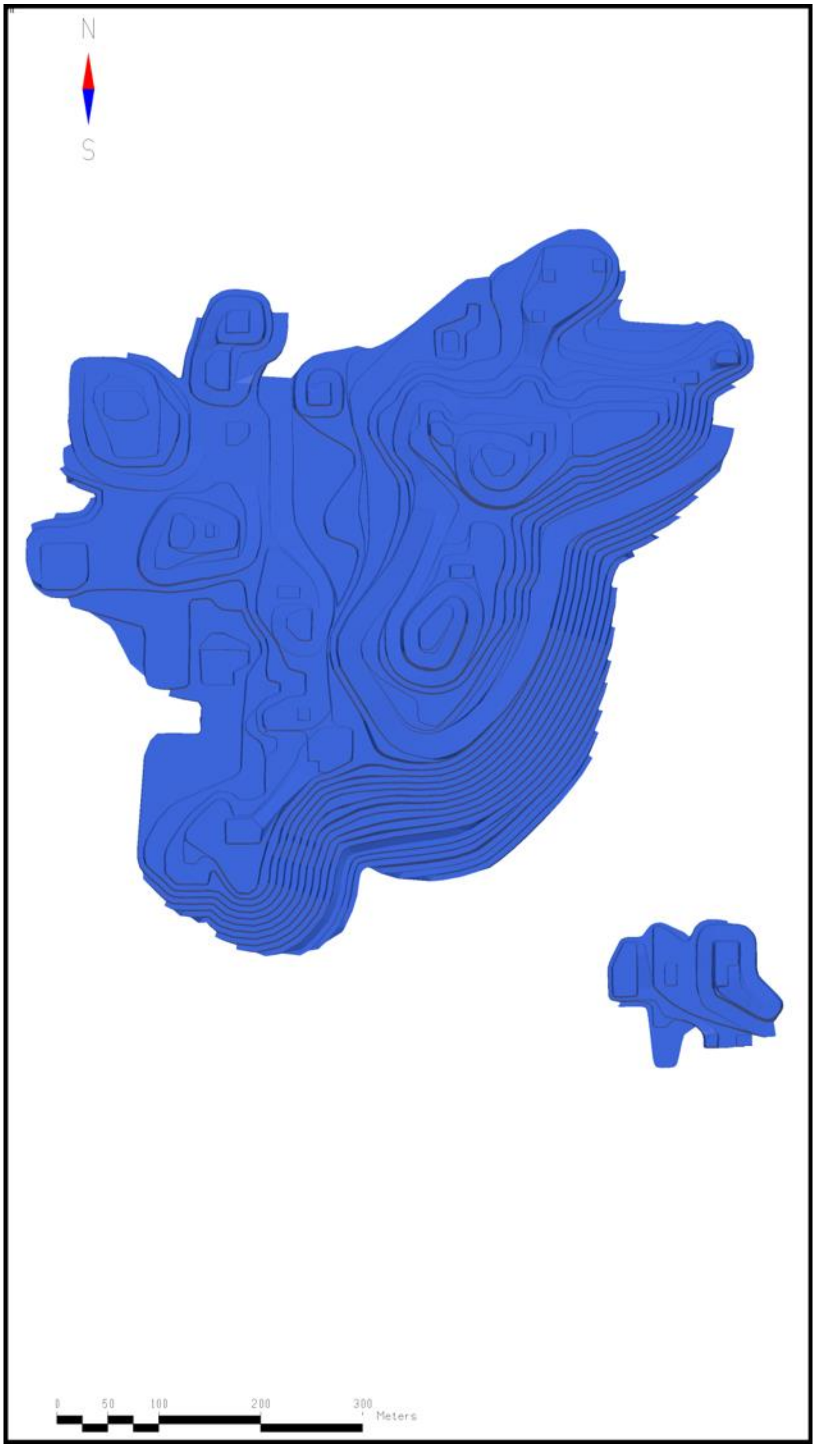


Figure 16.3 – Korkan Pits

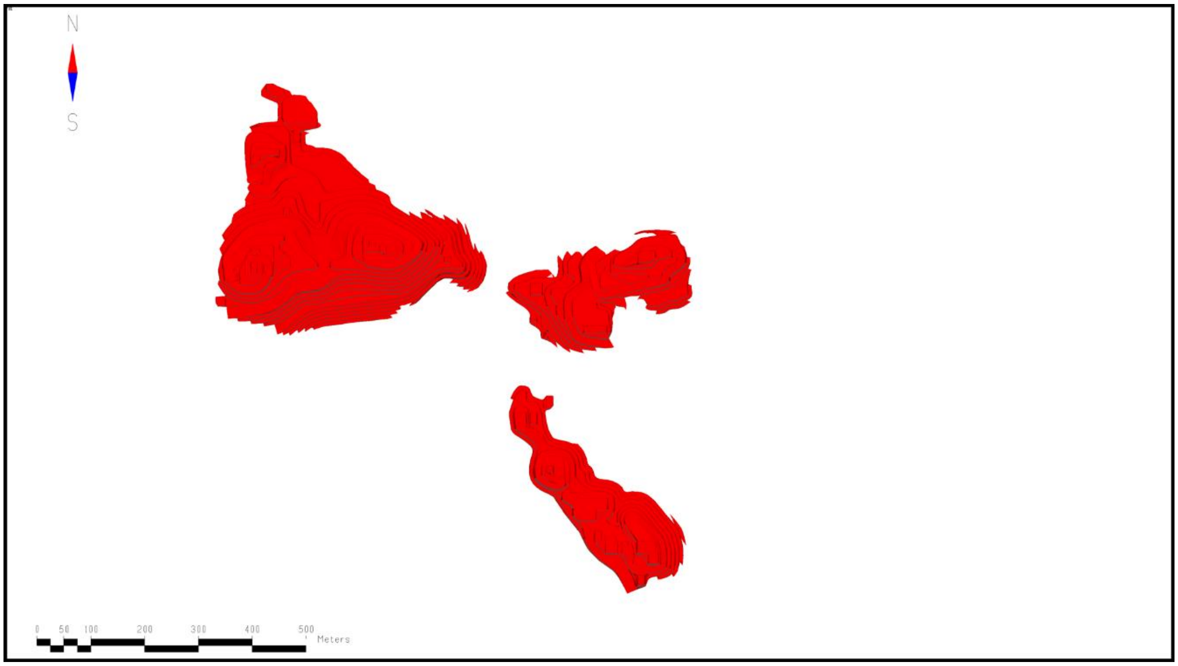
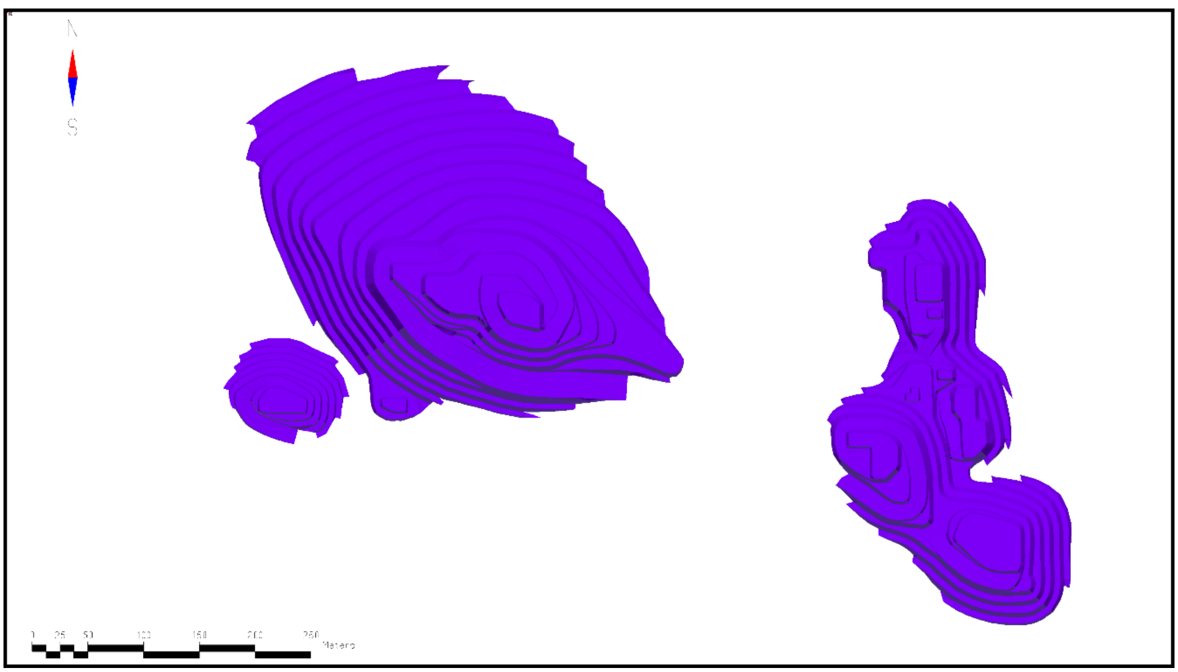


Figure 16.4 – Korkan West Pits



16.2.2 WASTE STOCKPILE DESIGNS

Waste material mined from each of the Timok deposits will be stored in piles located close to each pit to limit haulage costs. In addition to pure waste rock and mineralised material below the breakeven COG, the stockpile areas will also store the sulphide material above the breakeven cut-off grade of 0.57 g/t Au identified in the MRE in a separate pile for potential future reclamation and treatment.

The stockpiles were designed with the following parameters:

- 10 m high lifts;
- 33.6° lift face angle;
- 8.3 m berms every lift;
- 24° overall slope;
- 33% swell factor (from bank in-situ $m^3$to in-place m^3).

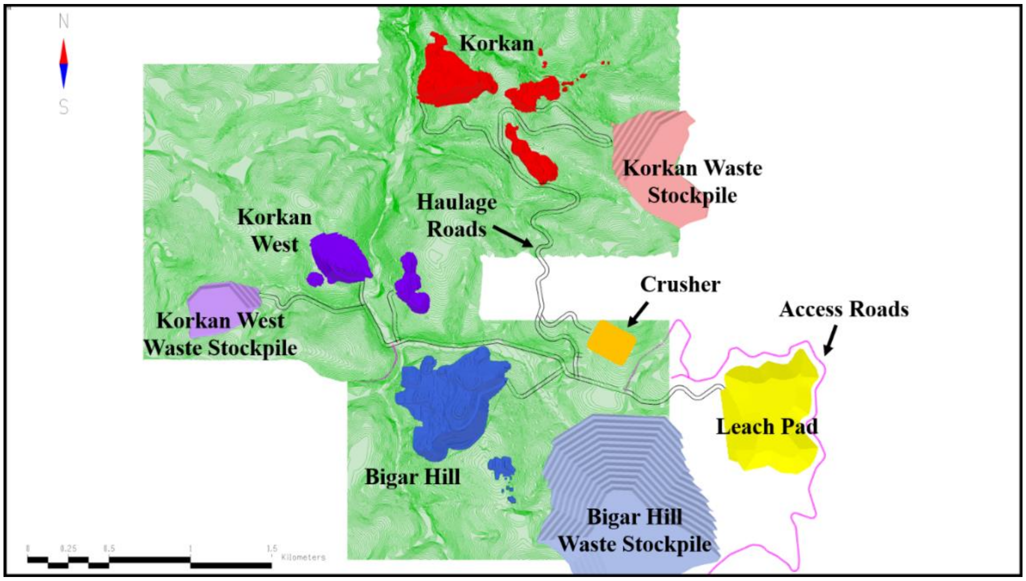
The total volume of for each stockpile is given in Table 16.2 and locations relative to each deposit is shown in

Figure 16.5.

Table 16.2 – Waste Stockpiles

Description	Tonnage (kt)	Volume (m^3)
Bigar Hill	30,426	15,213,000
Korkan	12,307	6,153,000
Korkan West	5,612	2,806,000

Figure 16.5 – Waste Stockpile Locations



16.3 Geotechnical

This section is reproduced and/or summarised from the SRK consulting report titled "*Timok Pit Geotechnical Pre-Feasibility Study" prepared for* DPM and *issued in July 2020.*

16.3.1 Pit Slope Angles

16.3.1.1 Overview

SRK has provided open pit slope design parameters for the three (3) proposed open pits of the Project: Bigar Hill, Korkan , Korkan West . The design parameters are based on geotechnical site investigations, available local and regional geological data, and well-established geotechnical design methods used to estimate the Project design pit slope angles. SRK conducted the work in four phases over the period of January 2019 to July 2020.

A total of fourteen (14) HQ-sized triple-tube oriented geotechnical drill holes for a total of 2,204.2 m were drilled, logged and sampled to collect geotechnical information from the wall rock of the planned ultimate pits. Geotechnical data was also collected from an additional sixteen (16) resource holes (1,569.8 m) to provide additional coverage and information. SRK also conducted core photo logging of selected resource drill holes in areas where insufficient geotechnical data had been collected.

SRK has identified geotechnical rock mass units associated with the primary rock and alteration types, based on the results of the site investigation by SRK, the University of Belgrade Faculty of Mining and Geology and geological interpretations by DPM. Major geological structures (faults and foliation) have been included in the geotechnical slope stability analyses for each pit. Slope stability analyses were conducted using industry standard limit-equilibrium software, finite element analysis software, and in-house proprietary SRK tools.

Where possible, televiewer (optical and acoustic), caliper and full waveform sonic downhole surveys were conducted by Terratec Geophysical Services. A total of fourteen (14) drill holes were partially surveyed. A point load testing program was set up as well as a geomechanical laboratory testing program commissioned by the University of Belgrade. Most of the testing was conducted by the mining faculty of the University of Bor. Approximately 10% of the total testing was conducted by the Mining Institute Ltd. Belgrade to accommodate some type of testing not feasible at University of Bor.

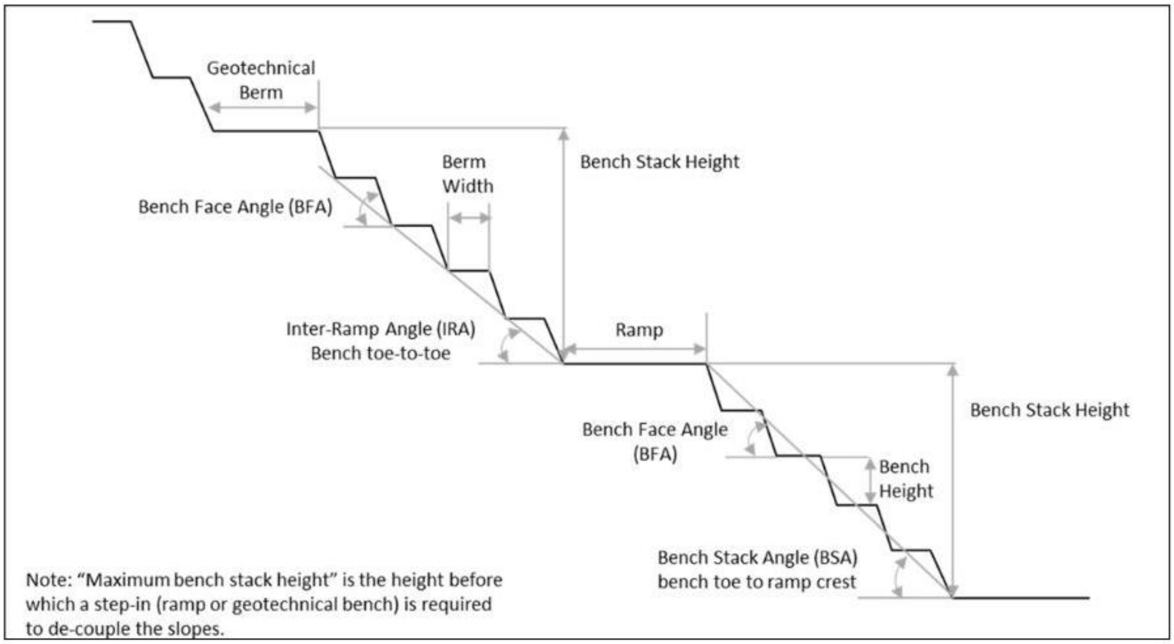
A field program to characterize the hydrogeological conditions was conducted as part of the investigations by the University of Belgrade (Faculty of Mining and Geology, Department of Hydrogeology). The program included in-situ-hydraulic conductivity testing, and the installation of standpipe and vibrating wire piezometers (VWPs). Details from the report relevant to this study were incorporated into the SRK Geotechnical report. Full details of the field program are provided in the report by the University of Belgrade (University of Belgrade, 2019a).

SRK completed hydrogeological studies for each of the proposed pits, and numerical simulations of pit dewatering/depressurization have been carried out. SRK interpreted hydrostratigraphic units, estimated hydraulic conductivity and storage parameter values, and formulated a conceptual hydrogeologic model for the Project area. The conceptual model was used as the basis for developing a numerical hydrogeologic model. The calibrated numerical model was used to evaluate the effort required to depressurize the open pit slopes to satisfy geotechnical constraints identified in the open pit slope designs.

SRK reviewed the proposed pit areas and surrounding terrain for potential geohazards, including the identification of karts areas, utilising field mapping, aerial photographs, and satellite imagery. SRK and Bor University personnel completed ground-truthing of potential geohazards; the preliminary design of mitigation structures were completed by those responsible for the various Project facilities at risk from the identified geohazards.

SRK used observations on site, data, and statistical analyses to define geotechnical domains and select representative geomechanical properties. Domains were grouped by lithology and modelled in Leapfrog using the lithology 3D models. For Korkan and Korkan West the units were further defined using the oxidation model. For Korkan, definition was also done using a numeric model of RQD

Figure16.6 – Typical Open-Pit Wall Design



Source: SRK 2020, page 66

16.3.1.2 Bigar Hill Pit Design

A multi-component site investigation program was completed to provide data for the Bigar Hill pit design work. Approximately 1,510 m of geotechnical drilling was completed, distributed over nine (9) core holes. SRK geotechnically logged the holes.

A laboratory testing program was completed, consisting of the following tests:

- Uniaxial compressive strength (56 tests);
- Triaxial (42 tests);
- Brazilian tensile strength (49 tests);
- Static Modulus (84 tests);
- Direct shear testing (9 tests);
- Wave velocity (49 tests).

An appropriate quantity of quality data was collected to characterize the geological units of the study area and support PFS-level slope designs.

The proposed Bigar Hill pit has been divided into five (5) geotechnical domains based on rock types and the different structural geology fabrics in the area. Discontinuity sets and geotechnical units for each domain are identified for use in the slope designs. Design sectors are based on the anticipated main orientations of the proposed pit walls, as determined from previous pit optimization studies.

Recommended inter-ramp slope angles vary from 36° to 58° based on wall orientation, overall wall height, geotechnical domain, and controls on slope stability. Inter-ramp slope heights are limited to 100 m, after which a geotechnical berm (or ramp) with a minimum width of 15 m is required. The inter-ramp height limits and geotechnical berms provide flexibility in the mine plan to mitigate potential slope instability; access for slope monitoring installations; and working space for in-pit wells, drains, and other water management infrastructure. All final pit slopes are assumed to be excavated using controlled blasting. Depressurization of the pit slopes is required and should be achievable with a combination of vertical wells and horizontal drains.

The Bigar Hill open pit slope designs are outlined in Table 16.3 and Figure 16.7.

Table 16.3 – Bigar Hill Design Recommendations

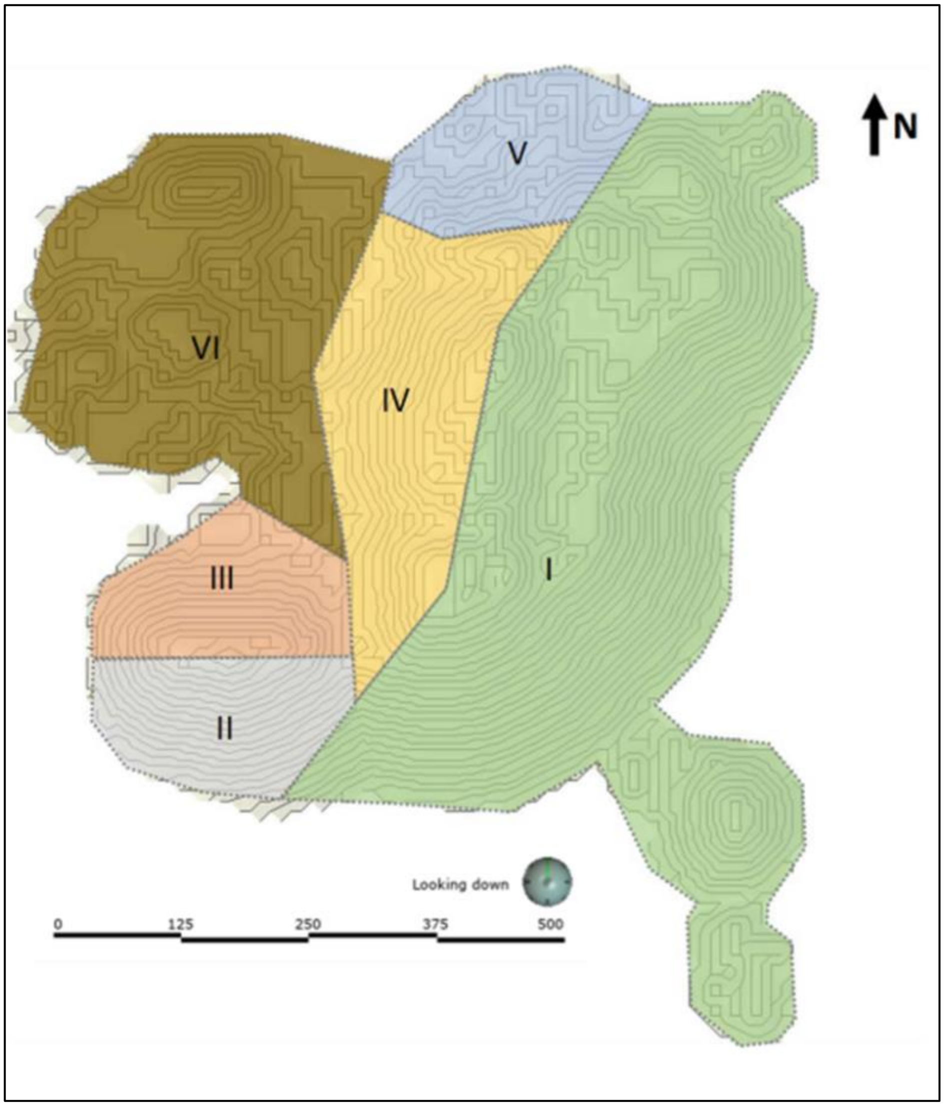
Domain	Lithology	BFA (°)	Bench Height (m)	Planned Berm Width (m)	Design IRA (°) (toe to toe)	Max. Stack Height (m)	Geotechnical Berm Width (m)	Maximum Overall Slope Angle (°)*
I	Vol	80	15	7.0	57	105	15	49
	S2/S1/BBX	80	10	5.8	53	100	15	
II	S2/S1/BBX	80	10	5.8	53	100	15	52
	SLS	80	10	4.5	58	100	15	
III	SLS	80	10	4.5	58	100	15	50
	S1/S2	80	10	7.5	47	100	15	
IV	S1/BBX	80	10	7.5	47	100	15	44
V	SLS	80	10	7.5	47	100	20	44
	Weathered material	75	5	5.5	36	-	-	-

Additional

- 15 m wide berm at the weathered-fresh rock contact for all domains
- 15 m wide geotechnical berm between the volcanic unit and underlying units (S2 & SMR)
- 15 m wide geotechnical berm between S1 and SLS units
- Strip on steep bedding in Domain V
- Weathered material should be about 5m to 35m thick (shallowest over the volcanic rocks in Domain I)
- *Overall angles are based on 22.5 m ramps and the scoping study pit plan geometry, to be updated

(Source: SRK Consulting - Appendix I, pg. 680)

Figure 16.7 – Plan View of the Ultimate Pit Shell Showing the Section Lines and Design Domains for Bigar Hill Deposit



(Source: SRK Consulting - Appendix I, pg. 680)

16.3.1.3 Korkan Pit Design

A multi-component site investigation program was completed to provide data for the Korkan pit design work. Data from eleven geotechnical drill holes (consisting of approximately 1,203 m of drilling) was used to divide the Korkan Zone into five geotechnical domains.

Laboratory testing of core samples from the completed geotechnical drilling included:

- Uniaxial compressive strength (48 tests);

- Triaxial (17 tests);
- Brazilian tensile strength (44 tests);
- Static Modulus (47 tests);
- Direct shear testing (9 tests);
- Wave velocity (40 tests).

The slope designs assume final walls will be excavated using controlled blasting, consistent with the approach proposed for the Bigar Hill pit. The recommended inter-ramp slope angles vary from 36° to 53° based on wall orientation, overall wall height, rock mass quality, and structural controls on slope stability. Inter-ramp slope heights are limited to 100 m after which a geotechnical berm (or ramp) with a minimum width of 20 m is required. Depressurization of the pit slopes is required and should be achievable with a combination of vertical wells and horizontal drains.

Table 16.4 and Figure 16.8 outlines the Korkan open pit slope designs.

Table 16.4 – Korkan Design Recommendations

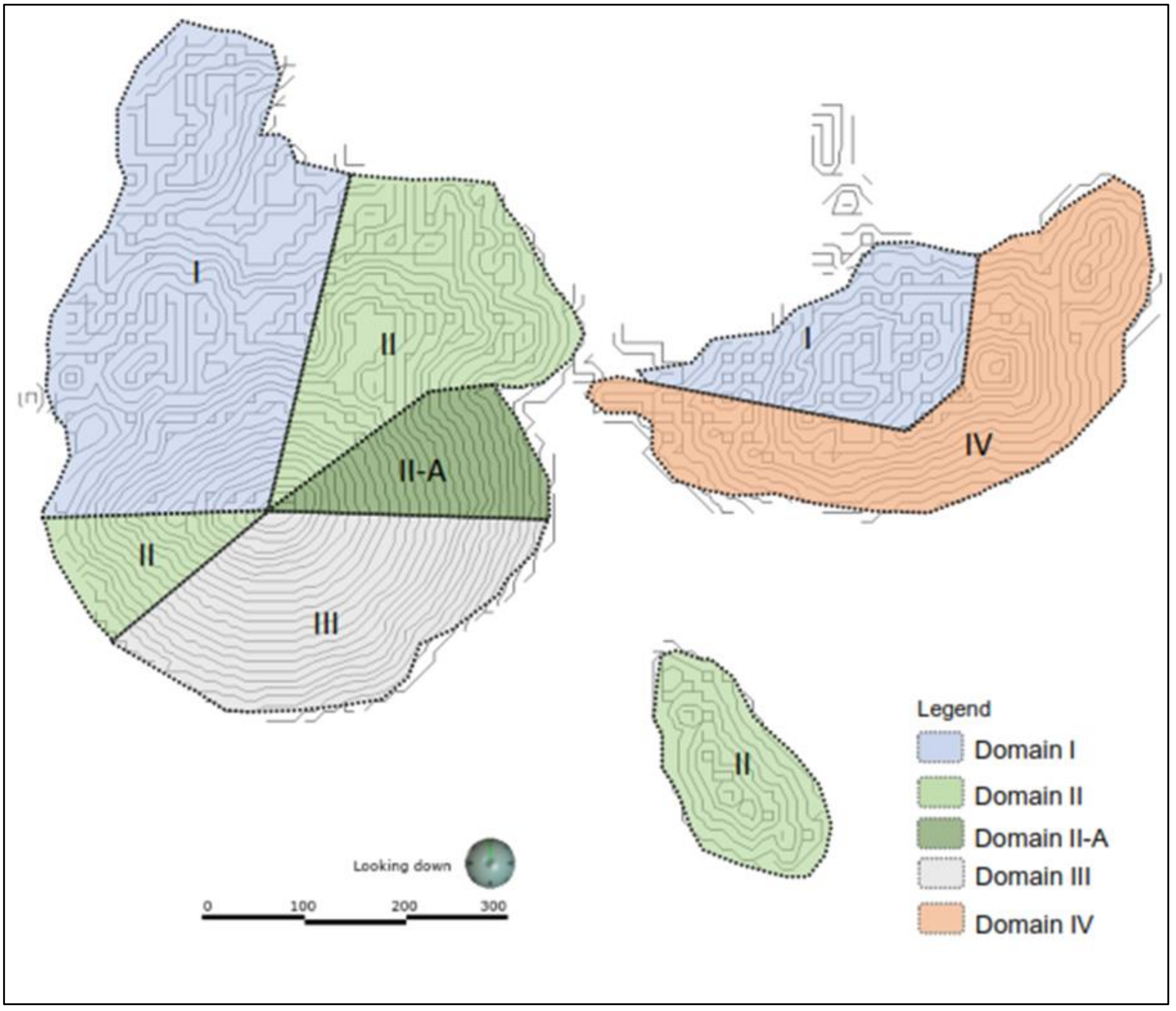
Domain	Lithology	BFA (°)	Bench Height (m)	Planned Berm Width (m)	Design IRA (°) (toe to toe)	Max. Stack Height (m)	Geotechnical Berm Width (m)	Maximum Overall Slope Angel (°)*
I	KLS Trans	80	10	13.3	34	101	20	34
II	S1/KLS	80	10	5.8	53	100	20	46
II-A	S1	75	10	10	38	100	20	35
III	S1 low RQD	80	10	5.8	53	100	20	45
IV	S1/KLS	80	10	5.8	53	100	-	48
	Weathered material	75	5	5.5	36	-	-	34

Additional

- 15 m wide berm at the weathered-fresh rock contact for all domains
- 15 m wide geotechnical berm between S1 and SLS units
- Weathered material is expected to be about 5m to 30m thick (based on modelled oxide and core photos)
- *Overall angles are based on 22.5 m ramps and the scoping study pit plan geometry, to be updated
- Domain II-A is governed by high POF of bench failures and bench break-back

(Source: SRK Consulting - Appendix I, pg. 687)

Figure 16.8 – Plan View of the Ultimate Pit Shell Showing The Section Lines And Design Domains For Korkan Deposit



(Source: SRK Consulting - Appendix I, pg. 687)

16.3.1.4 Korkan West Pit Design

A site investigation program including ten geotechnical drill holes (consisting of approximately 1,061 m of drilling) and hydrogeological testing was completed. Data from the site investigation was used to divide the Korkan West zone into five (5) geotechnical domains.

Laboratory testing of core samples from the geotechnical drilling included:

- Uniaxial compressive strength (14 tests);

- Triaxial (8 tests);
- Brazilian tensile strength (11 tests);
- Static Modulus (15 tests);
- Direct shear testing (7 tests);
- Wave velocity (10 tests).

The slope designs assume that final walls will be excavated using controlled blasting. The recommended inter-ramp slope angles vary from 28° to 54°; based on overall wall height, wall azimuth, rock mass quality, and geological structures. Inter-ramp slope heights are limited to 50 to 100 m after which a geotechnical berm (or ramp) with a minimum width of 15 to 20 m is required. Depressurization of the pit slopes is required and should be achievable with a combination of vertical wells and horizontal drains.

Korkan West open pit slope designs are presented in Table 16.5 and Figure 16.9.

Table 16.5 – Korkan West Design Recommendations

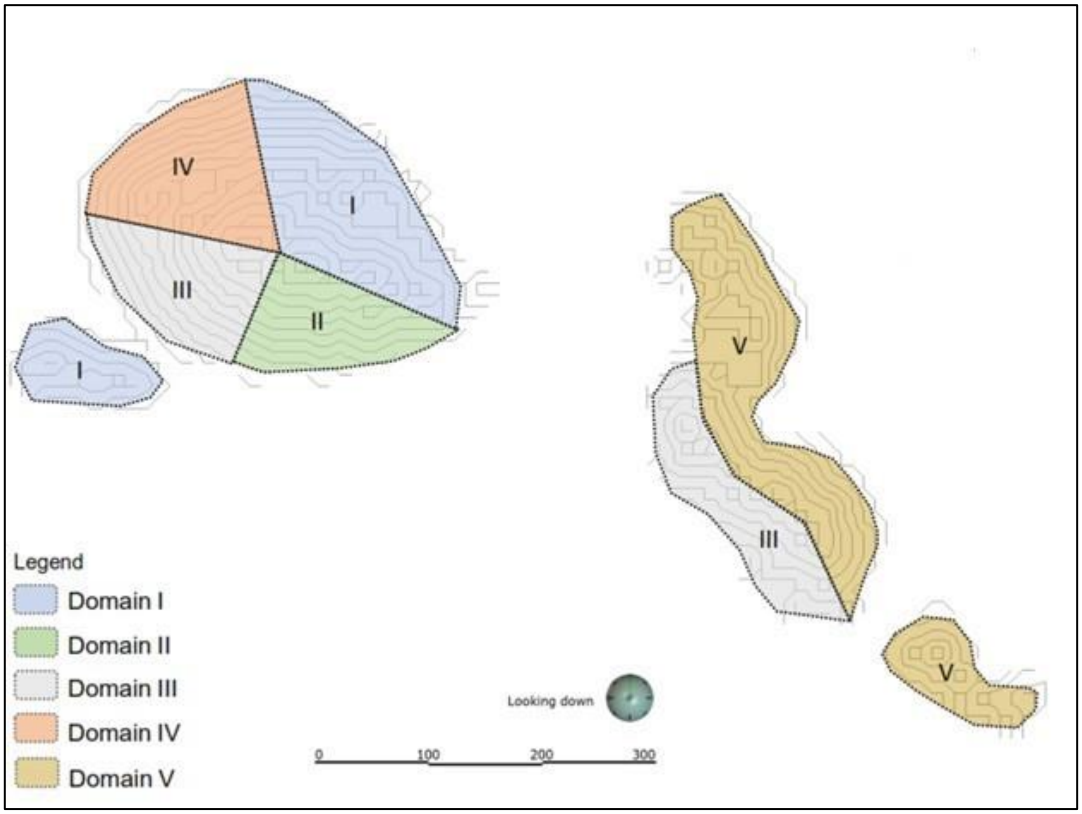
Domain	Lithology	BFA (°)	Bench Height (m)	Planned Berm Width (m)	Design IRA (°) (toe to toe)	Max. Stack Height (m)	Geotechnical Berm Width (m)	Maximum Overall Slope Angel (°)*
I	S1 (fresh-Oxide)	70	10	11.5	33	50	20	32
II	JLS	75	10	4.5	54	100	15	47
III	S1-Oxide	70	10	5.5	48	100	20	45
IV	S2-S1 (fresh-Oxide)	70	10	15.5	28	50	20	28
V	SMR-S2-S1	70	10	5	49	50	15	46
Weathered material		75	5	6.2	34	-	-	-

Additional

- 15 m wide berm at the weathered-fresh rock contact for all domains
- *Overall angles are based on 22.5 m ramps and the scoping study pit plan geometry – geotech to verify the practical pit design
- Two geotechnical berms are required for Domain IV but the haul road can serve as one of these berms

Source: SRK Consulting - Appendix I, pg. 684)

Figure 16.9 – Plan View of The Ultimate Pit Shell Showing Section Lines And Design Domains For Korkan West Domain



(Source: SRK Consulting - Appendix I, pg. 684)

16.3.2 Slope Design Implementation

Achieving the proposed design criteria will require depressurization of the pit walls through the use of vertical wells and horizontal drains. Geological structures may affect bench and inter-ramp scale slope stability and therefore depressurization of these structures will be required.

Monitoring of pit slope displacements at various scales will be required. Inter-ramp and overall scale slopes should be monitored for deformations. It is recommended that a state-of-the-art monitoring system be installed for all three (3) pits. This monitoring system should include multiple robotic-theodolites and survey prisms, mobile slope stability radar units, slope inclinometers, piezometers, and extensometers. The system would be computerised and use radio telemetry or a similar technology to provide real-time data to on-site geotechnical and mining staff. The recommended state-of-the-art monitoring system has not been included in the Capex.

It will be important to manage geological hazards during mining operations.

16.4 Pit Dewatering

The effective radius of influence of pit dewatering on the surrounding groundwater level in the rock was determined using standard analytical equations (Kyrieleis & Sichardt, 1930; Niccoli, et al, 1998). This radius was then used to determine the groundwater inflow rate to the pit (Marinelli & Niccoli, 2000); surface water run-off to the pit is outline in Section 18.15 The calculation results are for the steady-state inflow with the full extent of the pit excavated.

In order to determine the transient inflow rate during both the initial stages of dewatering and as the pit depth is increased, a numerical groundwater model of the pit would need to be developed. This model would allow modelling of the impact on dewatering rates of annual changes in groundwater level as well as providing an input into any water quality modelling undertaken. The development of a groundwater model is recommended for the next stage of Project engineering.

16.5 Mine Planning

A mine plan (or schedule) was prepared to estimate a probable production scenario for the Project and assess the mine equipment fleet requirements as well as mine capital and operating costs for the Project's financial model. The mine plan was based on an ore production rate of 2.5 Mtpa to the crusher/leach pad as determined by DRA. Only oxide and transitional ores are included in the mine plan for the heap leach process. Any sulphide material mined is sent for stockpiling for possible future processing.

Mine planning was performed using the XPAC software based on the final pit designs, the intermediate phases described in the following section and the DRA mineral resource block model. The mine plan was estimated monthly for the pre-production period as well as the first two (2) years of production; the remaining five years of production were estimated on a yearly basis. An additional year (Year 8) is added for re-handling material stockpiled during the first seven (7) years of production, sending that material to the crusher/heap leach. The current proposed mine plan considers a rate of 2.5 Mtpa of ore fed to the crusher/heap leach feed.

16.5.1 Phase Design

DRA defined 3 phases (or pushbacks) for each pit. Results of these phases are given in Table 16.6 and in Figures 16.10 to 16.12.

Table 16.6 – Reserves by Phase

Description	Unit	Bigar Hill			Korkan			Korkan West		
		Phase 1	Phase 2	Phase 3	Phase 1	Phase 2	Phase 3	Phase 1	Phase 2	Phase 3
Oxide Ore Tonnage	kt	4,191	3,804	89	2,251	531	323	2,774	49	603
Oxide Ore Grade	g/t Au	1.04	1.52	0.73	0.99	0.79	0.99	1.12	0.56	0.76
Transitional Ore Tonnage	kt	623	1,050	101	692	261	157	144	5	84
Transitional Ore Grade	g/t Au	0.88	1.33	1.06	1.12	0.97	1.14	0.80	0.66	0.77
Marginal Oxide	kt	350	317	7	188	44	27	232	4	50
Marginal Oxide Grade	g/t Au	0.24	0.24	0.24	0.24	0.24	0.24	0.24	024	0.24
Marginal Transitional	kt	52	88	8	58	22	13	12	0	7
Marginal Transitional Grade	g/t Au	0.24	0.24	0.24	0.24	0.24	0.24	0.24	0.24	0.24
Total Ore	kt	5,217	5,260	206	3,189	858	520	3,162	58	744
Total Ore Grade	g/t Au	0.96	1.39	0.85	0.96	0.80	0.98	1.04	0.54	0.72
Waste	kt	8,404	21,491	531	8,371	1,532	2,404	3,974	164	1,474
Total Material Mined	kt	13,621	26,750	737	11,559	2,390	2,925	7,136	222	2,219
Strip Ratio	SR	1.61	4,09	2,58	2,63	1,78	4.62	1.26	2.82	1.98

Figure 16.10 – Bigar Hill Phases

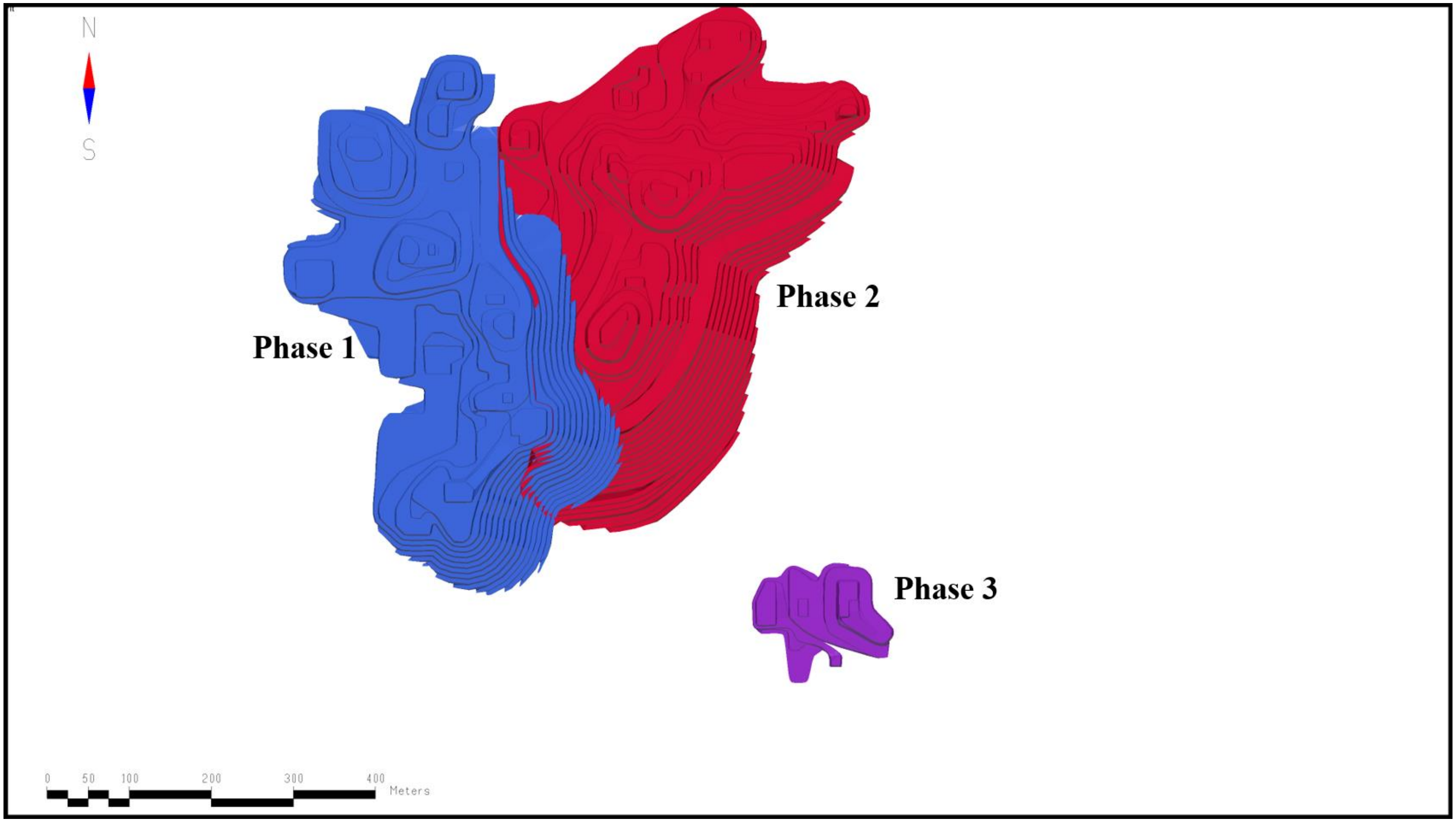


Figure 16.11 – Korkan Phases

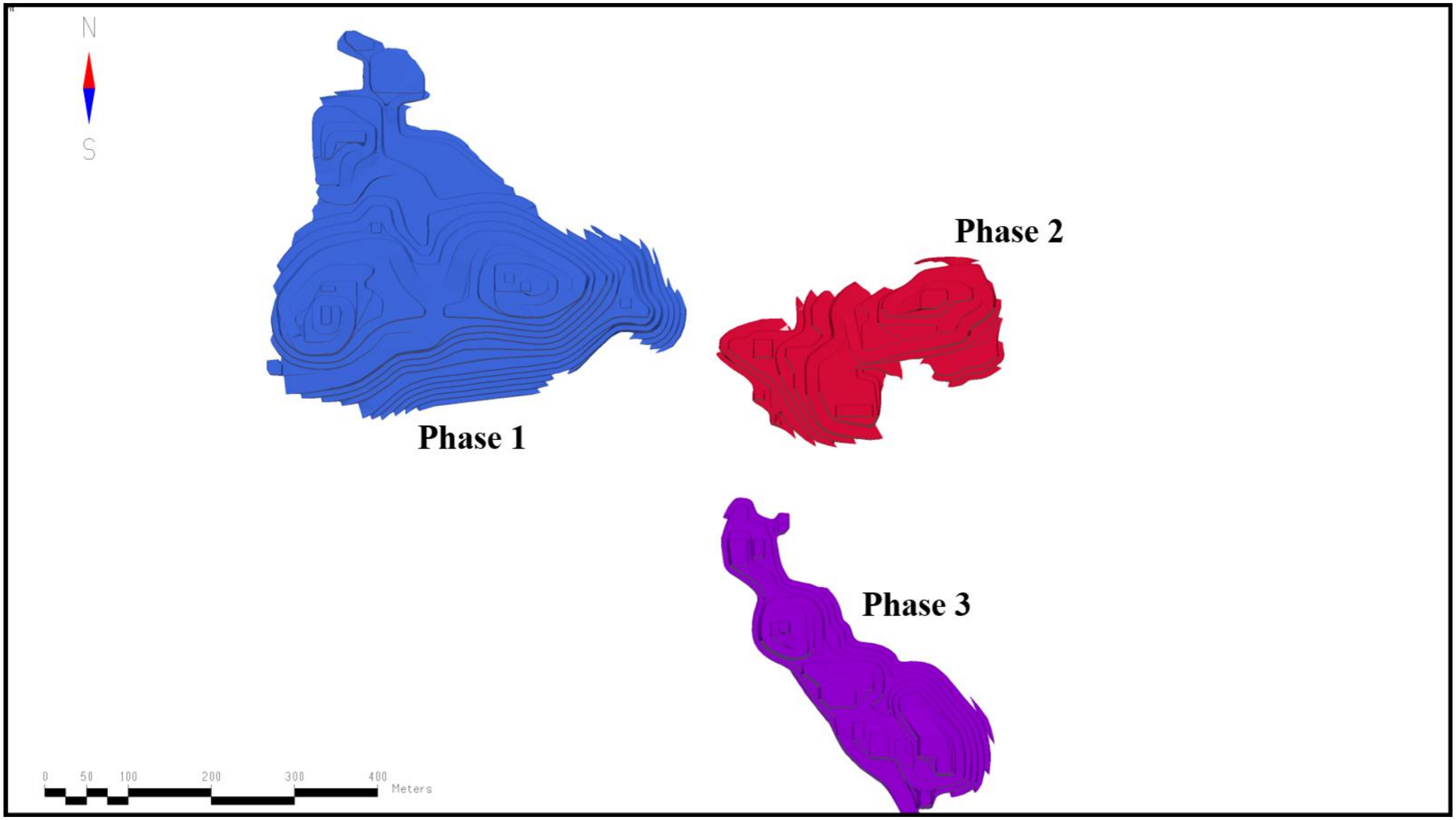
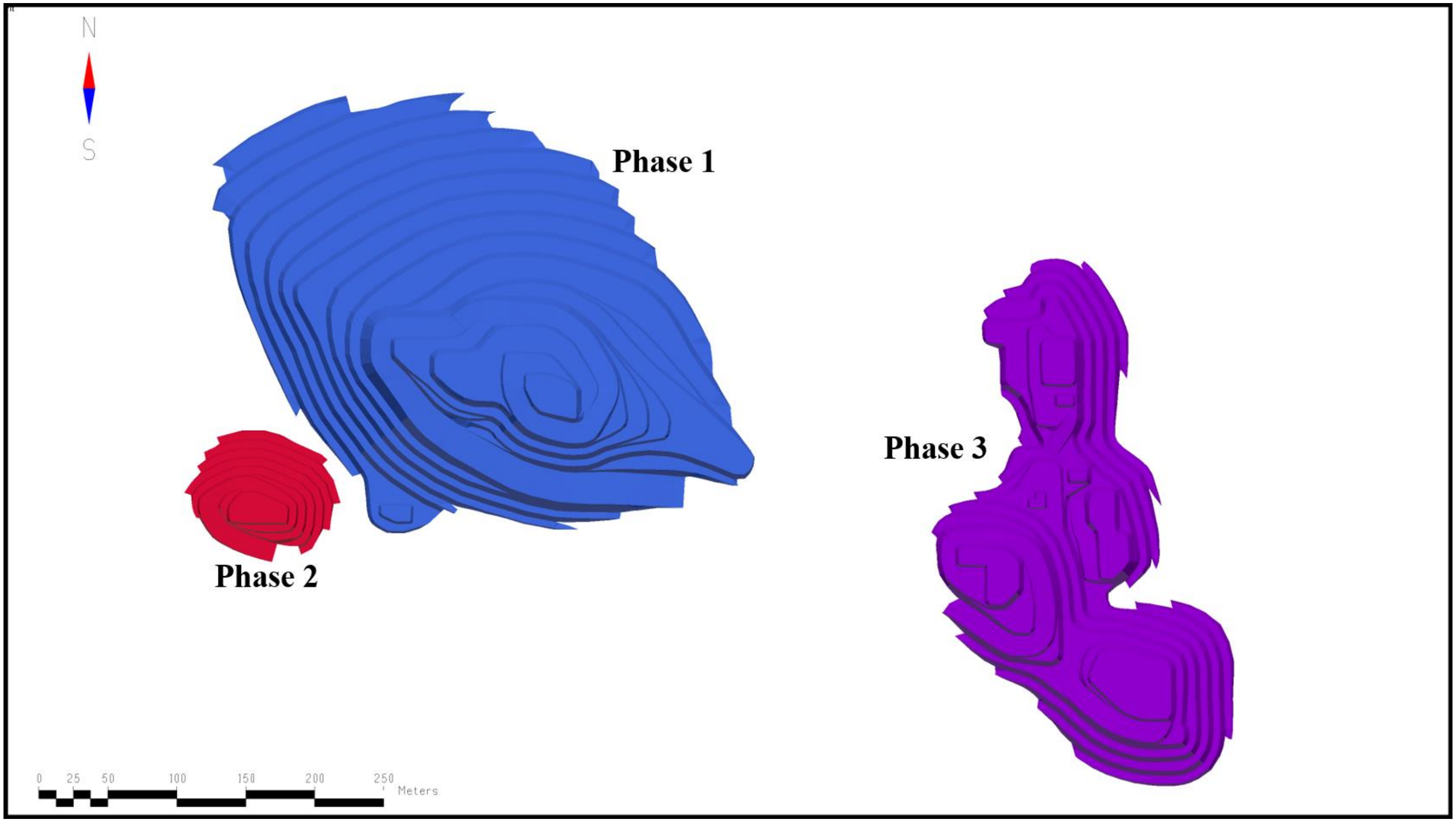


Figure 16.12 – Korkan West Phases



16.5.2 Mine Production Schedule

The mine production schedule (mine plan) was established using the XPAC mine planning software to determine the most productive mining sequence while maximising Au production and minimizing material movement.

The oxide material has a higher recovery than the transitional material, therefore XPAC favoured the higher recovery oxide ore over the lower recovery transitional ore; material could also be stockpiled to be processed later. Marginal ore was also stockpiled near the pits to be processed at the end of the mine life. Up to 3.5 Mtpa of ore could be mined, with a maximum of 2.5 Mtpa sent to the crusher; the remaining material is sent to the stockpile. Waste stripping was scheduled according to the ore production schedule, with a maximum of 10 Mtpa of material to be moved.

The resulting mine production schedule is presented in Table16.7 and in Figures 16.13 to 16.15. The schedule of the material sent to the crusher/leach pad is presented in Table 16.8 and Figure 16.16. The end of period maps shown in Figures 16.17 to 16.24 depict the benches mined during the pre-production period and the production periods of Year 1 to Year 7.

Table16.7 –Mine Production Schedule (by Pit)

Pit	Ore/Waste Tonnage by Pit (kt)								Total
	Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7	
BH Ore	44	1,390	1,644	2,375	1,736	2,082	1,339	72	**10,683**
KO Ore	0	0	0	0	801	657	1,547	1,562	**4,567**
KW Ore	29	1,349	1,842	0	0	529	216	0	**3,964**
Total Ore	74	2,739	3,486	2,375	2,537	3,267	3,102	1,634	**19,214**
Au Grade (g/t)	0.90	1.19	1.00	0.81	1.37	1.12	1.01	0.98	**1.07**
BH Waste	2,178	5,277	5,442	7,625	5,189	2,777	1,806	132	**30,426**
KO Waste	0	0	0	0	2,274	2,677	4,897	2,459	**12,307**
KW Waste	1,082	1,985	1,072	0	0	1,280	195	0	**5,612**
Total Waste	3,260	7,261	6,514	7,625	7,463	6,733	6,898	2,591	**48,345**
Total Material	3,333	10,000	10,000	10,000	10,000	10,000	10,000	4,226	**67,559**
Strip Ratio	44.2	2.7	1.9	3.2	2.9	2.1	2.2	1.6	**2.5**

Figure 16.13 – Mine Production Schedule (By Pit)

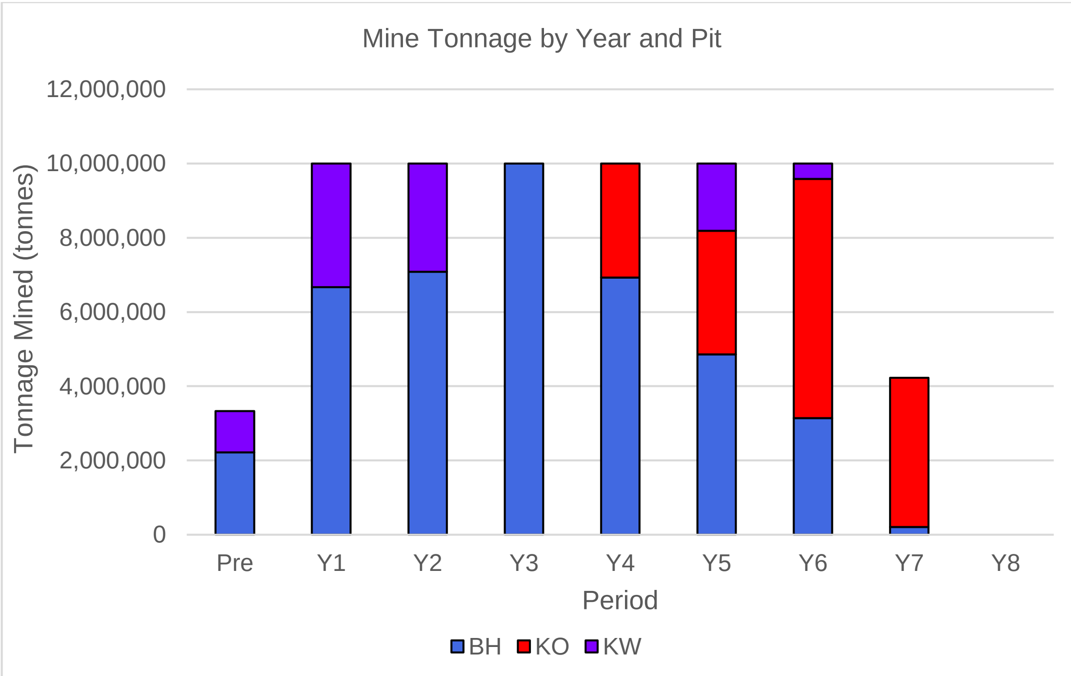


Figure 16.14 – Mine Production Schedule (Ore/Waste)

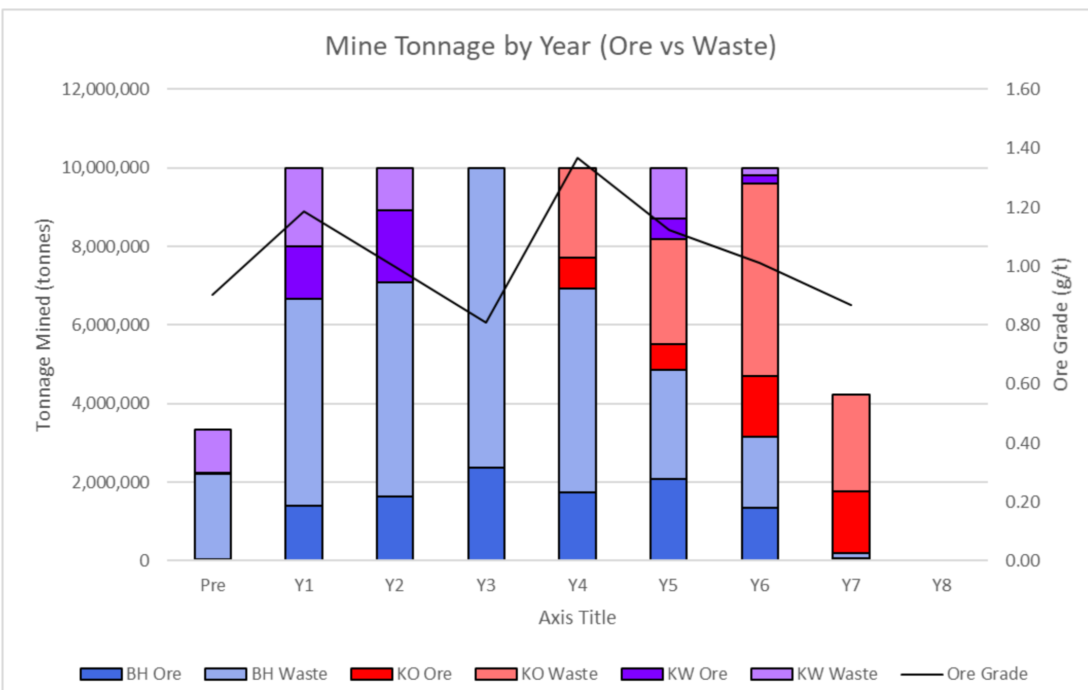


Figure 16.15 – Mine Production Schedule (By Ore Type)

Table 16.8 – Production Schedule to Leach Pad

Period	Pit to Leach Pad		Pit to Stockpile		Stockpile to Leach Pad		Leach Pad Feed		Contained Au	Recovered Au	Waste	Total Mined
	kt	g/t Au	kt	g/t Au	kt	g/t Au	kt	g/t Au	k oz	k oz	kt	kt
Pre-P	0	0.00	74	0.90	0	0.00	0	0.00			3,260	3,333
Y1	2,000	1.21	739	1.07	0	0.00	2,000	1.21	79	67	7,261	10,000
Y2	2,303	1.16	1,183	0.69	197	1.74	2,500	1.20	97	81	6,514	10,000
Y3	1,825	0.92	550	0.45	675	1.28	2,500	1.01	82	68	7,625	10,000
Y4	2,021	1.45	516	1.03	479	0.53	2,500	1.28	103	87	7,463	10,000
Y5	2,500	1.28	767	0.62	0	0.00	2,500	1.28	103	85	6,733	10,000
Y6	2,237	1.22	866	0.46	263	1.67	2,500	1.27	103	83	6,898	10,000
Y7	1,508	1.04	126	0.24	992	0.66	2,500	0.89	71	57	2,591	4,226
Y8	0	0.00	0	0.00	2,214	0.36	2,214	0.36	24	19	0	0
Total	**14,393**	**1.19**	**4,821**	**0.69**	**4,821**	**0.69**	**19.214**	**1.07**	**662**	**547**	**48,345**	**67,559**

Figure 16.16 – Production Schedule to Leach Pad

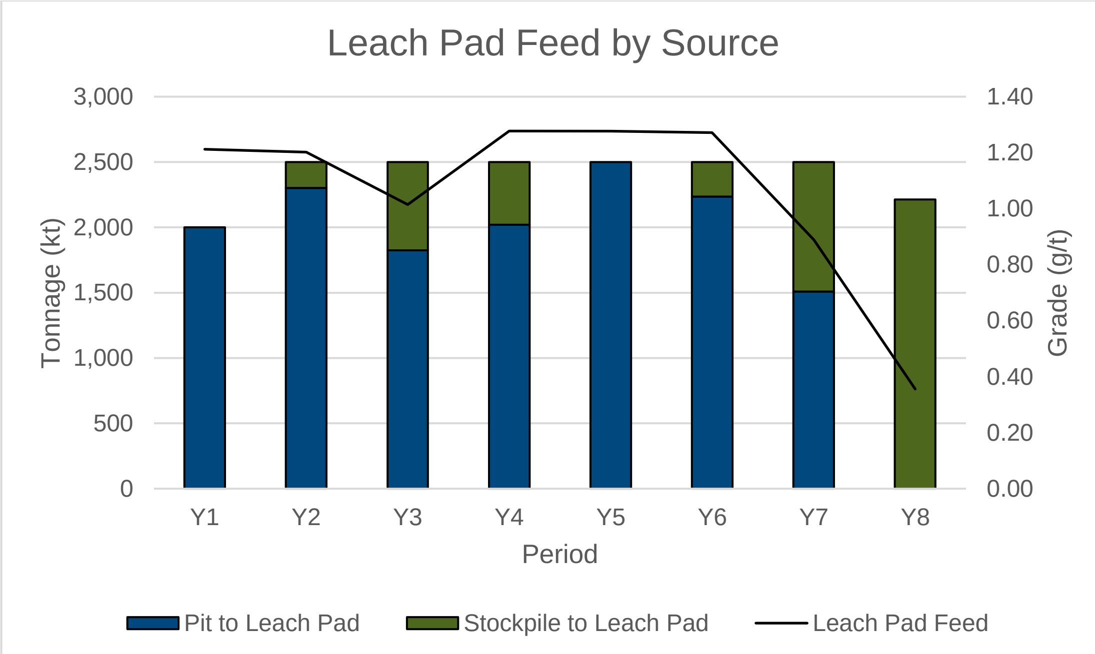


Figure 16.17 – Pre-Production

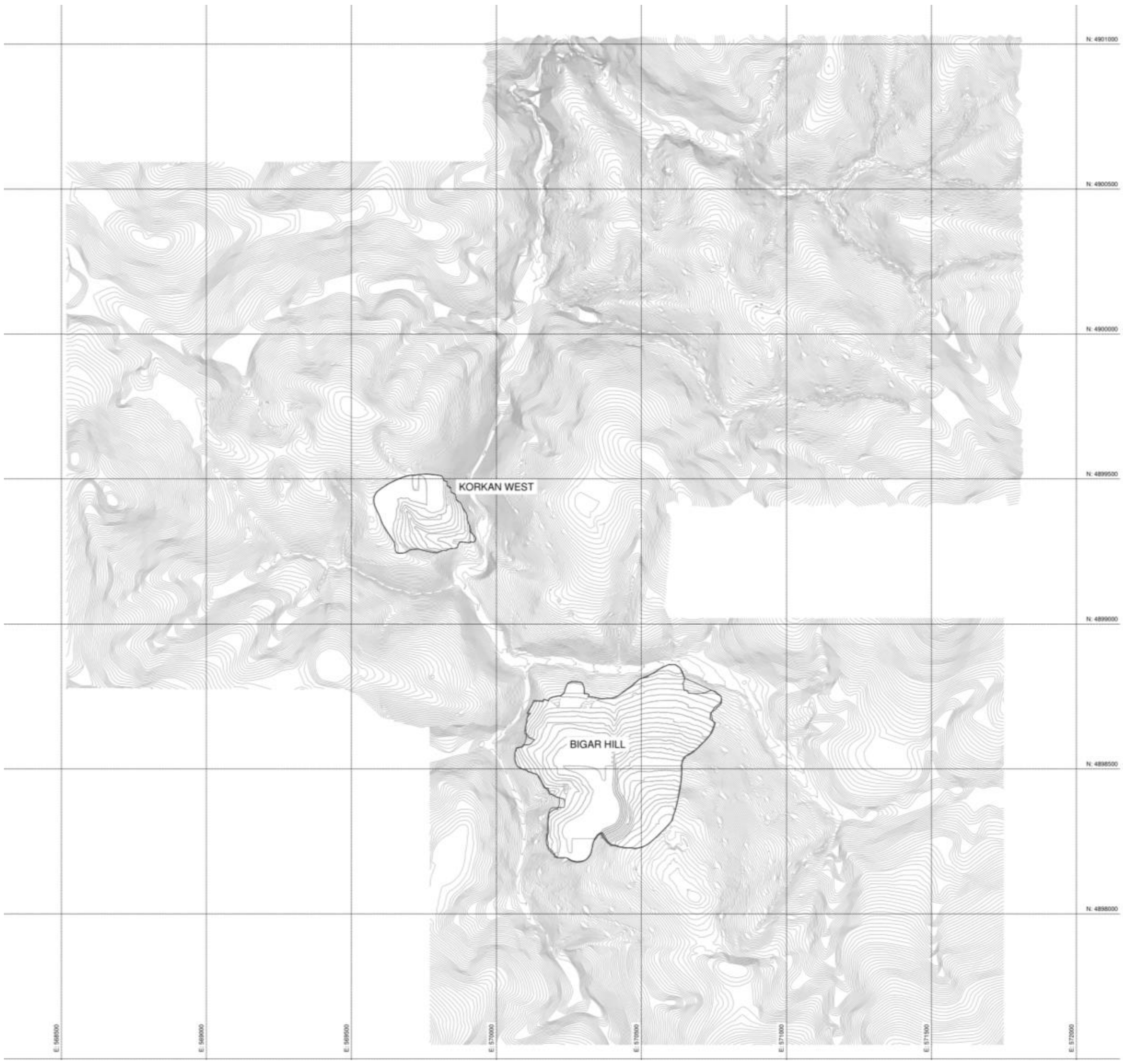


Figure 16.18 – Year 1

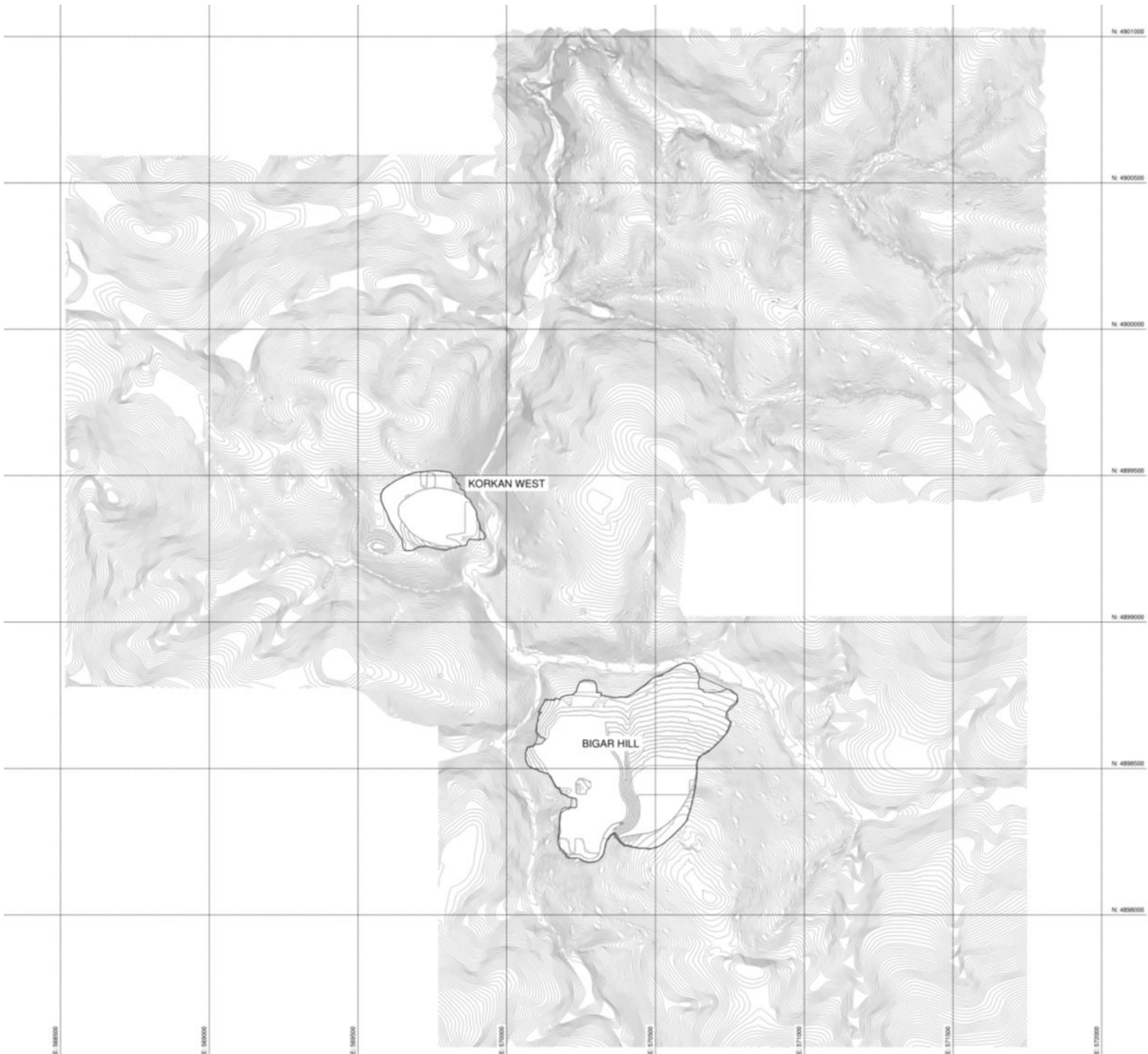


Figure 16.19 – Year 2

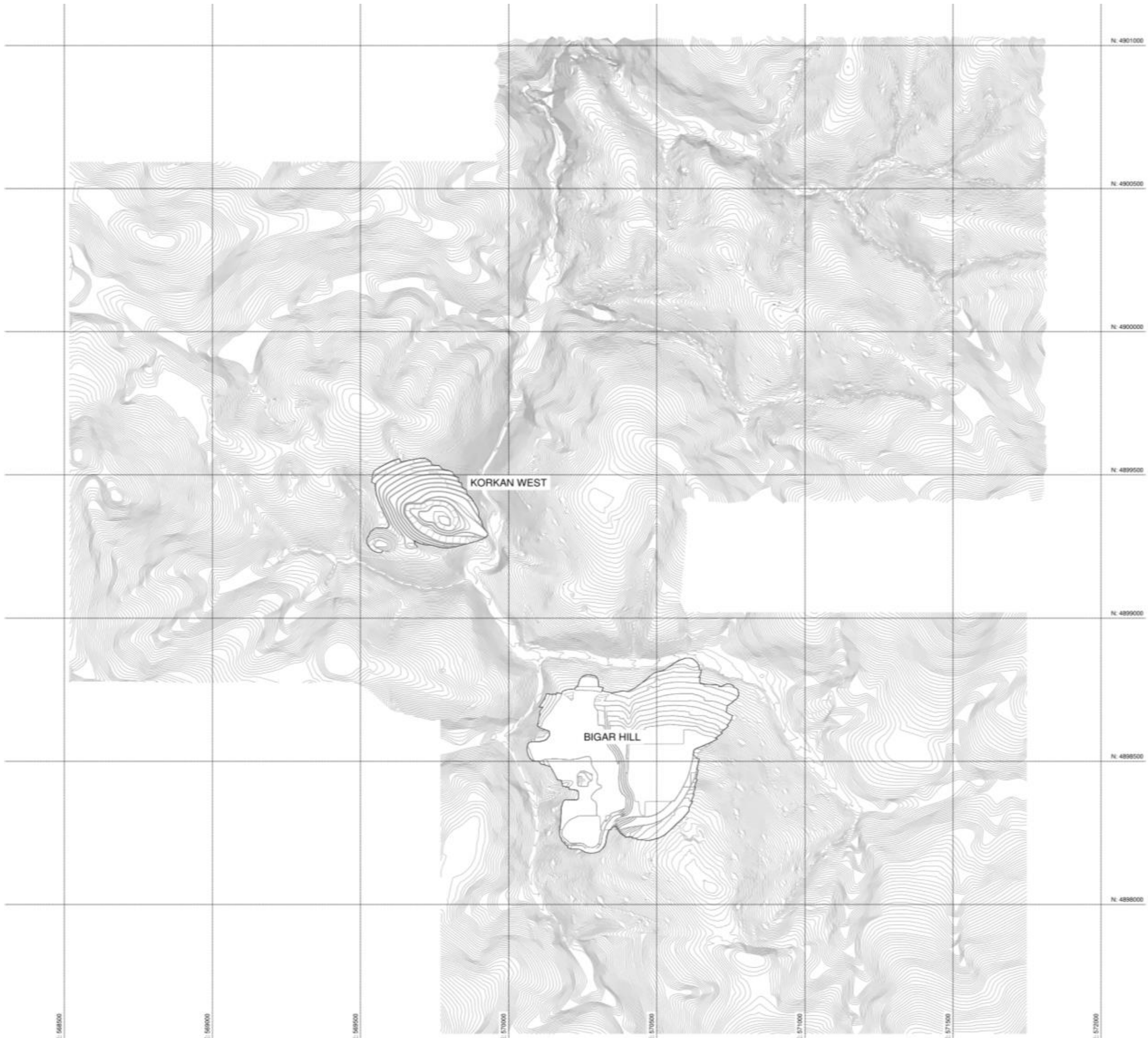


Figure 16.20 – Year 3

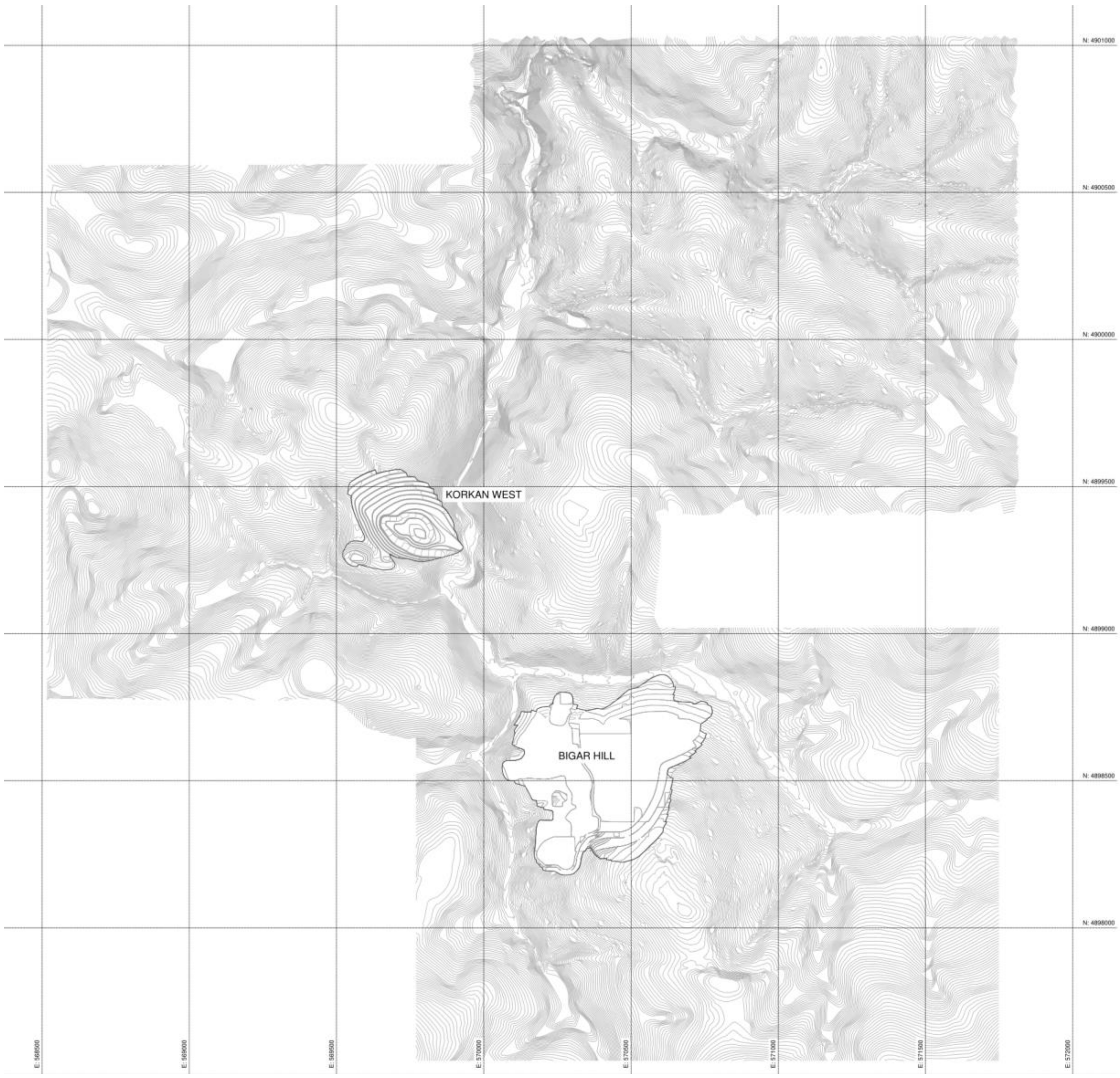


Figure 16.21 – Year 4

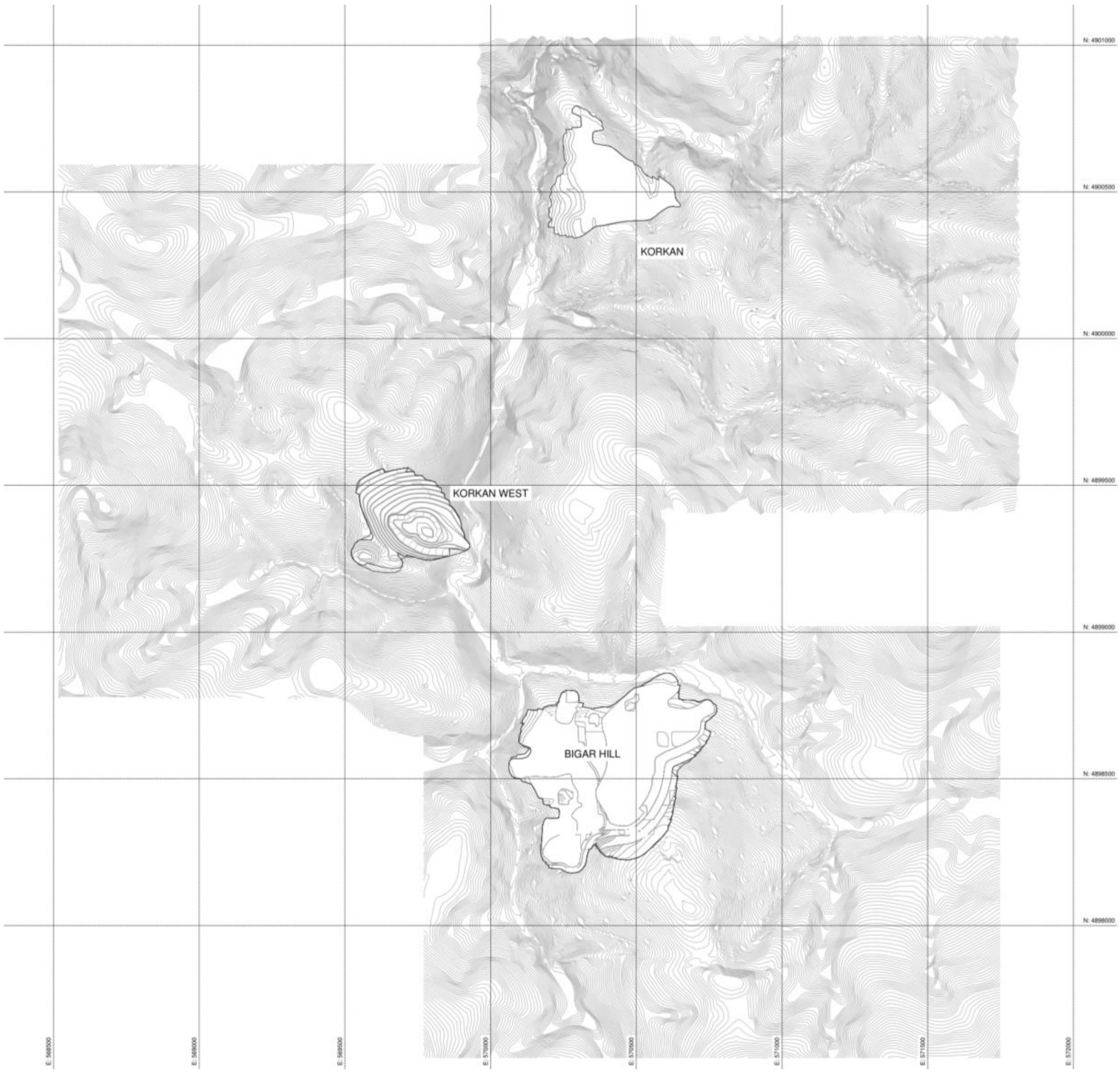


Figure 16.22 – Year 5

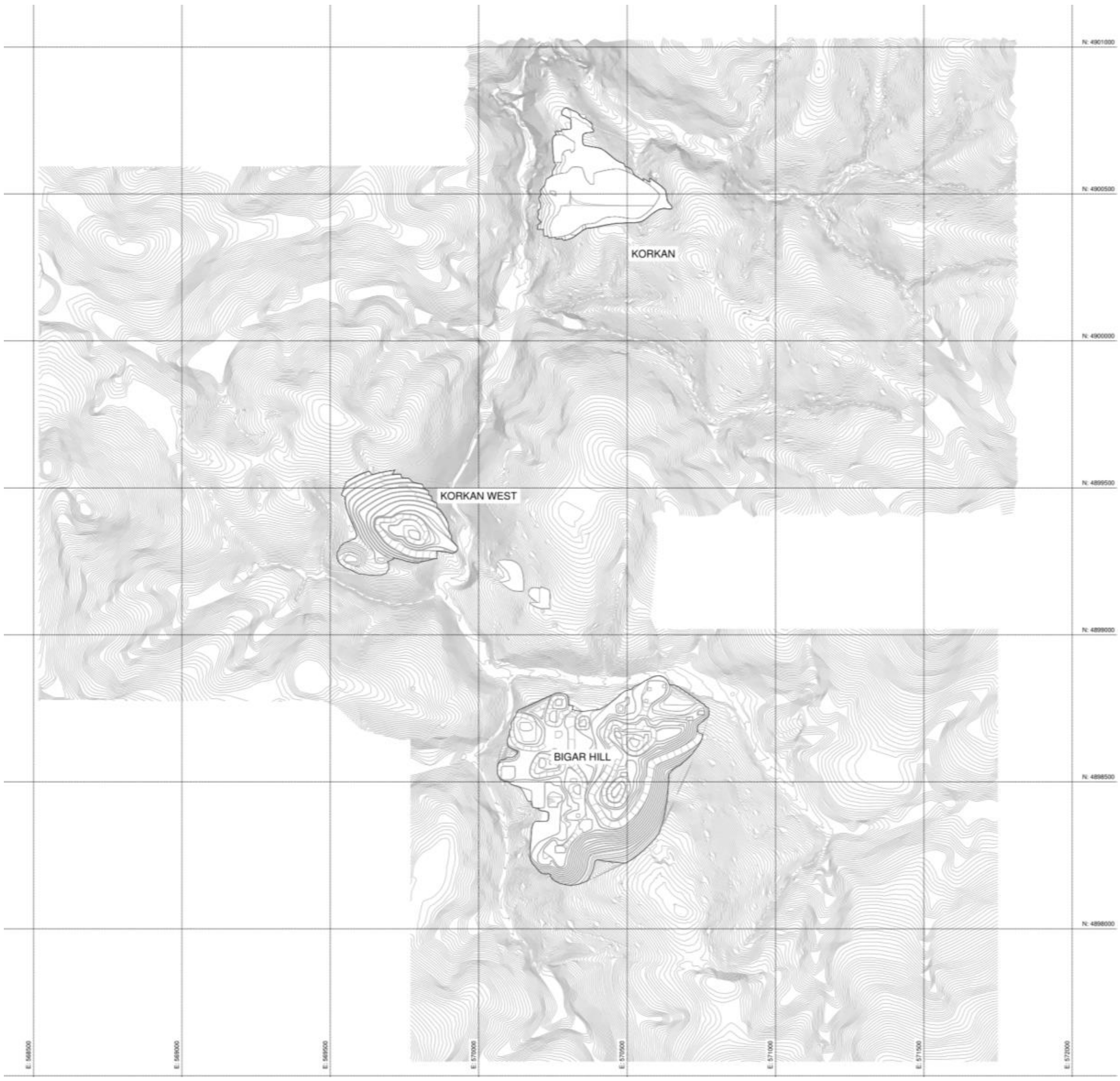


Figure 16.23 – Year 6

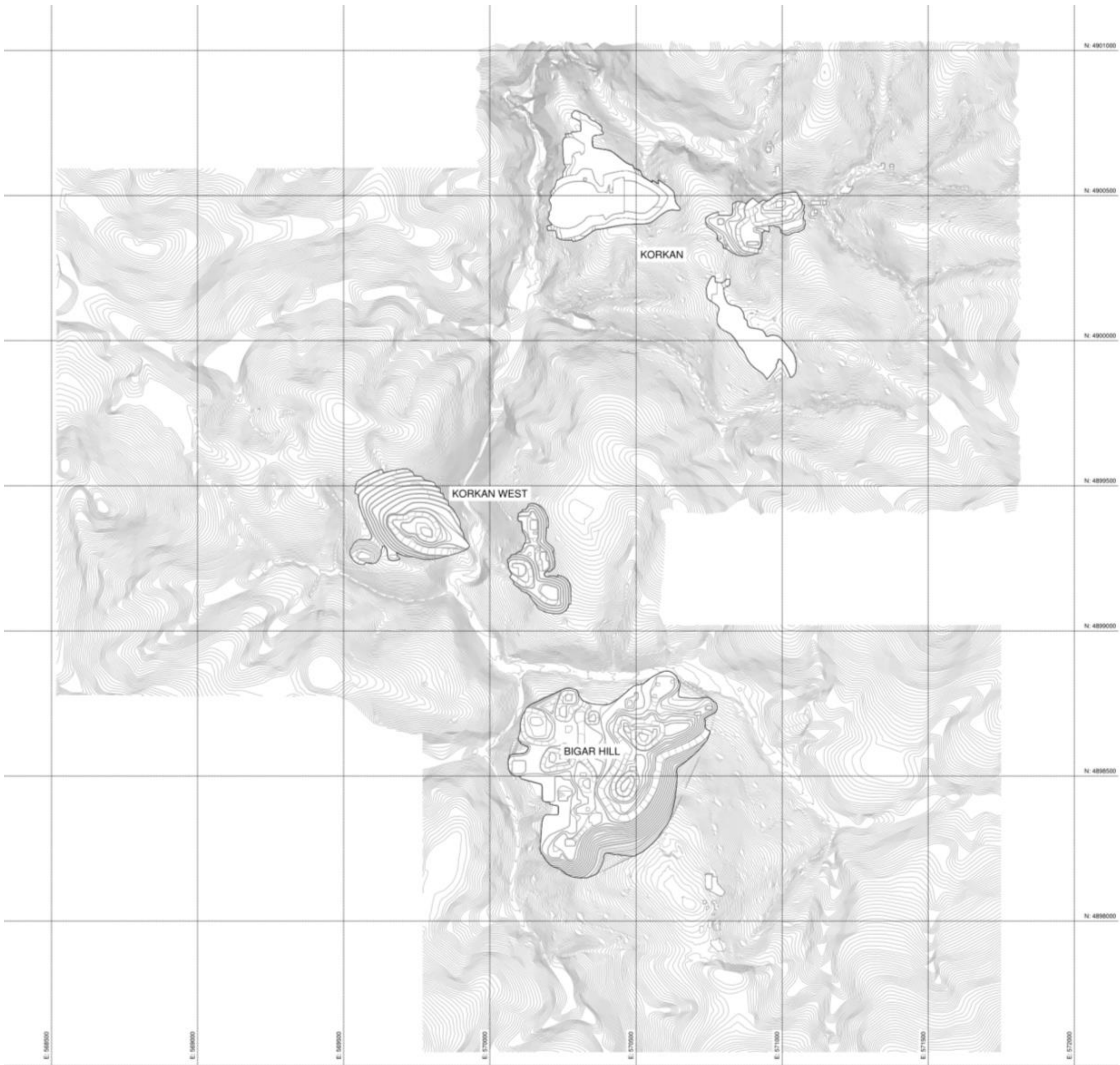
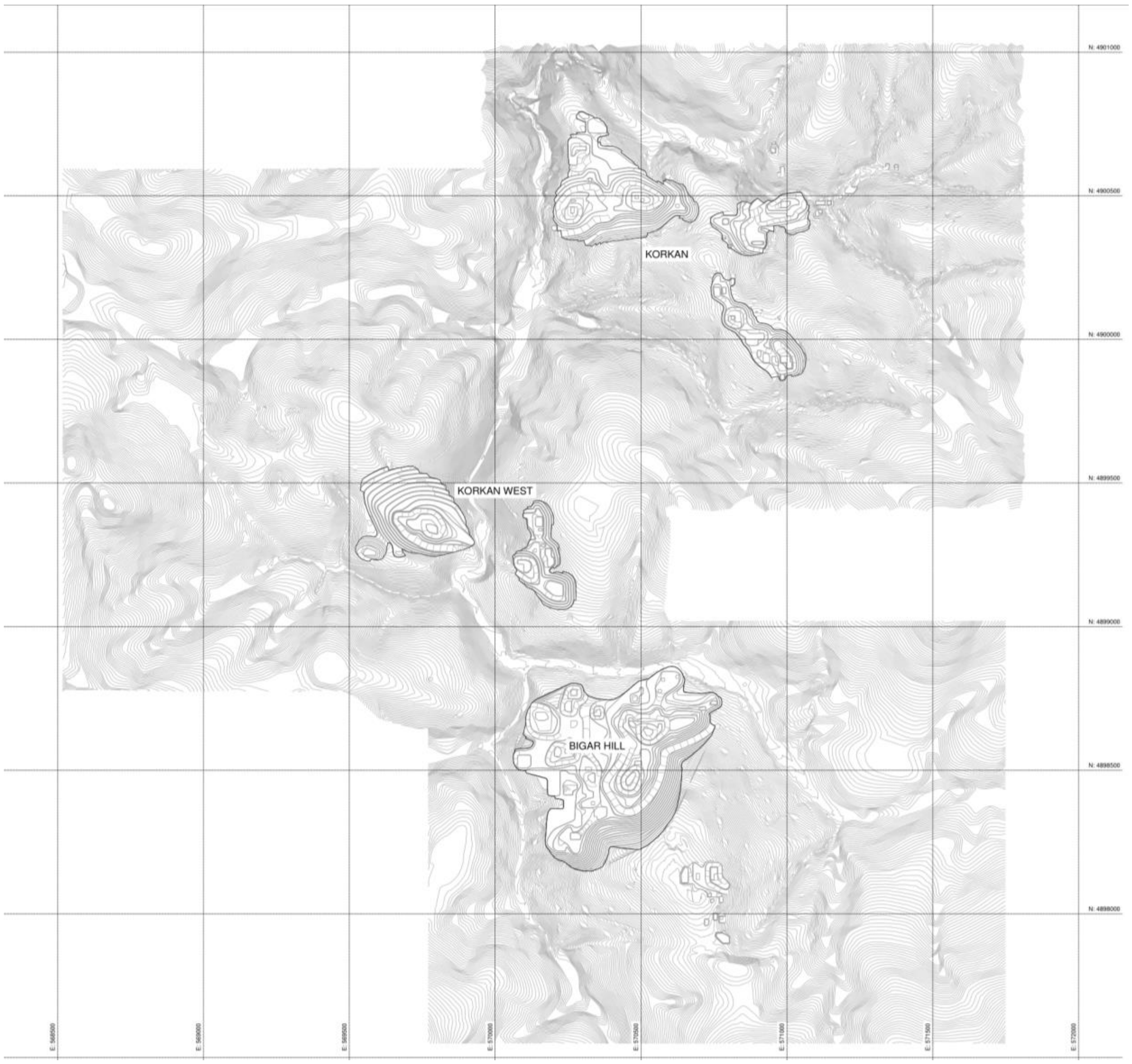


Figure 16.24 – Year 7



16.6 Mine Equipment

The following section discusses the fleet requirements to carry out the proposed mineplan.

16.6.1 Haulage Trucks

Haul truck requirements were estimated for transporting ore from the pit to the crusher and/or ore stockpiles, waste from the pit to the waste stockpiles, and from ore stockpiles to the crusher. The following parameters were used to calculate the number of trucks required to carry out the mine plan and resulting in 5,354 working hours per year for each truck:

- Mechanical Availability: 86%;
- Truck size: 60 t;
- Utilisation: 92%;
- Shift Schedule: two (2), twelve (12) hour shifts per day, 350 days per annum;
- Operational Delays: 140 minutes per shift, including lunch break, fueling, inspections and shift changes;
- Rolling Resistance: 3%.

Haul routes were designed for each period of the mine plan to calculate truck cycle times between the different pits, the stockpiles and the crusher, which were used to estimate productivity and truck requirements. These cycle times are a function of the truck spot time (at the loader or excavator), travel time, queuing time, and dump time. Travel times were calculated for each type and size of truck using the Talpac© software based on the different period routings. The travel distances per period and location are presented in Table 16.9 and the resulting haulage times are presented in Table 16.10. Truck productivity (tonnes per hour) is presented in Table 16.11. Truck hour requirements were calculated by applying the tonnages hauled to the productivity for each haul route.

Table 16.12 and Figure 16.25 present the haul truck requirements for each period of the life-of-mine.

Table 16.9 – Haulage Distances

Pit	Destination	Distance (km)							
		Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Bigar Hill	Crusher		4.14	4.00	3.81	4.03	5.61	3.77	4.40
	Waste Stockpile	1.95	4.02	4.51	5.03	5.71	7.74	4.56	5.13
Korkan	Crusher					5.69	6.30	8.59	3.99
	Waste Stockpile					3.95	4.97	3.43	4.13
Korkan West	Crusher		5.83	6.62			5.11	5.65	
	Waste Stockpile	1.95	2.31	3.41			3.62	4.41	

Table 16.10 – Cycle Times

Pit	Destination	Cycle Time (minutes)							
		Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Bigar Hill	Crusher		14.00	15.18	13.90	15.43	19.36	16.01	18.23
	Waste Stockpile	14.12	16.28	18.09	18.72	21.05	23.53	16.20	17.44
Korkan	Crusher					18.41	18.00	21.15	13.48
	Waste Stockpile					14.72	18.50	17.07	15.18
Korkan West	Crusher		18.00	20.42			16.99	18.22	
	Waste Stockpile	9.60	11.33	15.15			14.88	16.75	

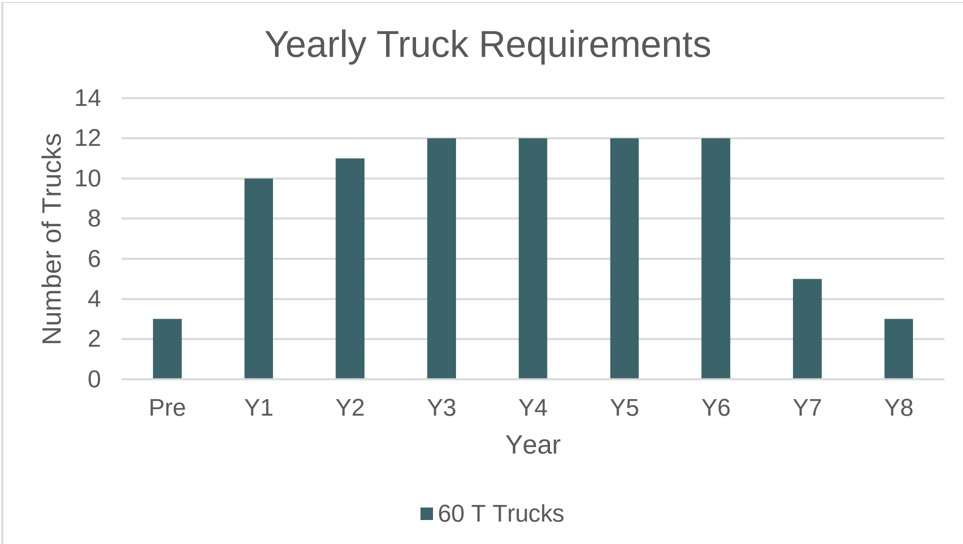
Table 16.11 – Productivity

Pit	Destination	Productivity (t/h)							
		Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Bigar Hill	Crusher		215	198	216	195	155	188	165
	Waste Stockpile	205	178	160	155	138	123	179	166
Korkan	Crusher					163	167	142	223
	Waste Stockpile					197	157	170	191
Korkan West	Crusher		167	147			177	165	
	Waste Stockpile	302	256	191			195	173	

Table 16.12 – Haulage Truck Requirements

Capacity	Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
54t	3	10	11	12	12	12	12	5	3

Figure 16.25 – Annual Truck Requirement



16.6.2 EXCAVATORS

The main ore and waste loading equipment selected are 6.8 m³ hydraulic excavators. Throughout the life-of-mine, 11.5 m³ loaders are used for bench cleanup, temporary replacement of hydraulic excavators during breakdowns, stockpile reclamation as well as for cleanups and road maintenance if required. The following parameters were used to calculate the number of excavators and loaders required to carry out the mine plan and resulting in 3,492 available working hours per year for each excavator and loader.

- Mechanical Availability: 86%;
- Utilisation: 60%;
- Shift Schedule: two (2), twelve (12) hour shifts per day, 350 days per year
- Operational Delays: 140 minutes per shift, including lunch break, fueling, inspections and shift changes.

Table 16.13 presents the excavator and loader requirements throughout the life-of-mine for the Project.

Table 16.13 – Loading Equipment Requirements

Equipment	Capacity	Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Excavators	6.8 m³	1	3	3	3	3	3	3	2	
Wheel Loaders	11.5 m³	1	2	2	2	2	2	2	2	1

16.6.3 DRILLING AND BLASTING

Production drilling will be carried out with down-the-hole (DTH) rotary drills producing 114 mm diameter, 5 m deep blast in both ore and waste. The following parameters were used to calculate the number drills required to carry out the mine plan and resulting in 5,354 available working hours per year for each drill.

- Mechanical Availability: 86%;
- Utilisation: 92%;
- Shift Schedule: two (2), twelve (12) hour shifts per day, 350 days per year
- Operational Delays: 140 minutes per shift, including lunch break, fueling, inspections and shift changes.

Based on these parameters, the drill pattern specifications presented in Table 16.14 and the drill productivity calculations presented in Table 16.15, a maximum of 3 drills are required during the mine life. Table 16.16 presents the number of drills required for each period of the life-of-mine.

Table 16.14 – Drill Pattern Specifications

Specification	Unit	Value
Bench Height	m	5
Sub drilling	m	1
Blasthole Diameter	mm	114
Blasthole Effective Diameter	mm	103
Spacing	m	3.3
Burden	m	3.3
Hole Length	m	6

Table 16.15 – Drilling Productivity

Specification	Unit	Value
Penetration Rate	m/min	0.38
Hole Length	m	6
Drill Time	min	15.17
Move, Spot, and Collar Hole	min	2.50
Level Drill	min	0.50
Pull Drill Rods	min	0.50
Total Drill Time per Hole	min	19.2
Drill Productivity	t/h	461

Table 16.16 – Drill Requirements

Equipment Type	Capacity	Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Production Drill	165 mm	2	4	4	4	4	4	4	2	N/A

The cost of the emulsion was estimated to be $143/100kg, with a powder factor of 0.28 kg/t.

16.6.4 SUPPORT EQUIPMENT

The fleet of production equipment requires service and auxiliary equipment for support, such as road maintenance, stockpile management, fuelling and equipment maintenance and employee transport between the different pits. Table 16.17 presents the support equipment requirements throughout the life-of-mine.

Table 16.17 – Support Equipment Requirements

Equipment Type	Pre-P	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Grader	1	2	2	2	2	2	2	2	1
Track Dozer	2	3	3	3	3	3	2	2	1
Grade Control Drill	1	1	1	1	1	1	1	1	1
Water Truck	1	2	2	2	2	2	2	2	1
Support Backhoe	1	1	1	1	1	1	1	1	1
Light Plant	2	3	3	3	3	3	3	2	2
Pick-Up Trucks	20	20	20	20	20	20	20	20	5
Man Bus	1	1	1	1	1	1	1	1	1
Lowboy and Tractor	1	1	1	1	1	1	1	1	1
Lube/Fuel Truck	1	1	1	1	1	1	1	1	1
Tire Manipulator	1	1	1	1	1	1	1	1	1
Mechanic Truck	1	1	1	1	1	1	1	1	1

16.7 Manpower Requirements

Manpower requirements for the Project are based on management and supervision requirements, operation of the mine equipment fleet 24 hours per day and 350 days per year, maintenance of the equipment and technical services (engineers, geologists and surveyors). The manpower requirements are presented in Table 16.18. They are expected to peak in Years 3 to 5.

Table 16.18 – Manpower Requirements

Position	Number of Employees								
	Pre	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Mine Operations									
Technical Superintendent (Expatriate)	1	1	1	1	1	1	1	1	1
Mine General Foreman	1	1	1	1	1	1	1	1	1
Mine Shift Foreman	4	4	4	4	4	4	4	3	1
Road Crew/Service Foreman	1	1	1	1	1	1	1	1	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1
Truck Operators	12	40	44	48	48	48	48	20	12
Shovel Operators	4	12	12	12	12	12	12	8	0
Loader Operators	4	8	8	8	8	8	8	8	4
Drill Operators	8	16	16	16	16	16	16	8	0
Grader Operators	4	8	8	8	8	8	8	8	4
Dozer Operators	8	12	12	12	12	12	8	8	4
Water Truck Operators	4	8	8	8	8	8	8	8	4
General Mine Equipment Operator	3	3	3	3	3	3	3	3	1
Blaster	1	1	1	1	1	1	1	1	
General Mine Labourer	5	10	15	15	15	15	15	10	2
Road/Pump Crew	1	1	1	1	1	1	1	1	1
Trainee	1	1	1	1	1	1	1	1	1
Subtotal	*63*	*128*	*137*	*141*	*141*	*141*	*137*	*91*	*38*

Position	Number of Employees								
	Pre	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Engineering and Geology									
Chief Engineer (EXPAT)	1	1	1	1	1	1	1	1	1
Senior Mining Engineer	1	1	1	1	1	1	1	1	
Open Pit Planning Engineer	2	2	2	2	2	2	2	1	
Geotech Engineer	1	1	1	1	1	1	1	1	1
Surveyor/Mine Technician	1	2	2	2	2	2	2	2	1
Surveyor Helper/Mine Technician Helper	1	2	2	2	2	2	2	2	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1	1	1	
Grade Control/Modeller Geologist	1	1	1	1	1	1	1	1	
Grade Control Technician	1	4	4	4	4	4	4	3	
Sampling/Geology Technician	1	4	4	4	4	4	4	3	1
Clerk/Secretary	1	1	1	1	1	1	1	1	
Subtotal	*14*	*22*	*22*	*22*	*22*	*22*	*22*	*19*	*7*

Position	Number of Employees								
	Pre	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Maintenance									
Maintenance Shift Foreman	4	4	4	4	4	4	4	4	1
Maintenance Planner/Contract Admin	1	1	1	1	1	1	1	1	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1
Lube Truck Driver	1	1	1	1	1	1	1	1	1
Tyre Man	1	1	1	1	1	1	1	1	1
Light Duty Mechanic	1	4	4	4	4	4	4	3	1
Heavy Duty Mechanic	1	4	4	4	4	4	4	3	1
Welder	1	2	2	2	2	2	2	1	1
Electrician	1	1	1	1	1	1	1	1	1
Apprentice	1	1	1	1	1	1	1	1	
Subtotal	*13*	*20*	*20*	*20*	*20*	*20*	*20*	*17*	*9*
Total Mine Employment	**90**	**170**	**179**	**183**	**183**	**183**	**179**	**127**	**54**

17 RECOVERY METHODS

Oxide and transitional ores from BH, KO, and KW deposits will be processed using conventional heap leach technology, adopting a Heap Leach Fill (HLF) design. The design stacking rate is based on processing 2.5 Mtpa, at an average gold grade of 1.07 g/t, and an overall discounted gold recover of 84.9% and 71.8% for the oxide and transitional ore types respectively.

The HLF pad will be constructed in two (2) separate phases: 9.9 million tonnes in Phase 1 with an addition of 9.3 million tonnes during Phase 2, totalling to 19.2 million tonnes of ore processed during 8 years of mine life. Phasing of the Project includes the earthworks and pad construction and lining and reduces initial capital expenditure.

Ore will be crushed in the three-stage crushing plant and then trucked to the HLF pad. An agglomeration circuit, considered for the high clay ore, will be used intermittently on an "as required" basis. The leach pad will be irrigated with Barren Leach Solution (BLS) at a solution irrigation rate of 467 m^3/h using a single stage irrigation plan. Pregnant Leach solution (PLS) will be recovered at a flow rate of 423 m^3/h (irrigation rate less evaporation). A 4.5 tonne ADR Plant has been designed to treat the PLS over the entire LOM and produce doré bars.

The process flowsheet is based on proven technology operated by numerous gold operations around the world. Key process plant design criteria were derived from the metallurgical test work programs conducted and have allowed for the sizing of major equipment items. Metal production estimates were prepared based on metallurgical test results, and the mine plan developed for the PFS. The plant layout and mechanical equipment selected ensure access for proper constructability, operability, and maintainability.

In the opinion of the QP, the metallurgical testwork conducted to date supports the selected process flowsheet and associated recovery factors for this pre-feasibility study Mineral Resource / Reserve estimate, and project evaluation.

17.1 Process Design Criteria

The heap leach operation was designed to crush and stack 2.5 Mtpa of ore to a crush size of 100% passing 25 mm. The design criteria were developed based upon testwork results described in Section 13, and the life of mine plan.

The processing facility is designed to operate 365 d/a. The crushing plant and agglomeration have been specified with an operating availability of 75%, equivalent to 18 h/d of operation. The leaching and ADR plant have been designed to operate at an availability of 95%, equivalent to 22.8 h/d.

Key process design criteria can be found in Table 17.1.

It should be noted that an average plant feed gold grade and a weighed average discounted gold recovery has been updated based on the ore tonnage and blending ratio developed in the latest mine plan. For a heap leach operation, the inputs to the plant are run-of-mine (ROM) ore, make-up water, sodium cyanide solution, powdered lime, powdered cement, and other reagents. The only output from the HLF will be doré bars.

Table 17.1 – Key Process Design Criteria

Parameter	Unit	Phase 1	Phase 2
Plant Throughput, Design, Dry	t/a	2,500,000	
Bigar Hill Oxide Gold Grade	g/t	1.19	
Bigar Hill Oxide Gold Recovery, Discounted	%	85.1%	
Korkan Oxide Gold Grade	g/t	0.9	
Korkan Oxide Gold Recovery, Discounted	%	86.2%	
Korkan West Oxide Gold Grade	g/t	0.99	
Korkan West Oxide Gold Recovery, Discounted	%	83.4%	
Bigar Hill Transitional Gold Grade	g/t	1.09	
Bigar Hill Transitional Gold Recovery, Discounted	%	68.8%	
Korkan Transitional Gold Grade	g/t	1.02	
Korkan Transitional Gold Recovery, Discounted	%	74.6%	
Korkan West Transitional Gold Grade	g/t	0.74	
Korkan West Transitional Gold Recovery, Discounted	%	81.0%	
Gold Head Grade, Weighed Average Design (for Oxide ore)	g/t	1.08	
Gold Recovery, Weighted Average Discounted (for Oxide ore)	%	84.9%	
Gold Head Grade, Weighed Average Design (for Transitional ore)	g/t	1.04	
Gold Recovery, Weighted Average Discounted (for Transitional ore)	%	71.8%	
Gold Head Grade, Weighed Average Design (for Oxide and Transitional ore)	g/t	1.07	
Gold Recovery, Weighted Average Discounted (for Oxide and Transitional ore)	%	82.6%	
Ore Moisture Content	w/w %	8%	
Crushing and Stacking Availability	%	75%	
Crushing and Stacking Operating Hours/Year	h/a	6,570	
Crushing and Stacking Ore Processing Rate	t/h	386	
Crushing Feed Top Size (F_{100})	mm	600	
Crushing Feed Size (F_{80})	mm	440	
Crushing Product Top Size (P_{100})	mm	25	
Crushing Product Size (P_{80})	mm	17	

Parameter	Unit	Phase 1	Phase 2
Crushing Work Index (CWi), Design	kWh/t	14.4	
Solution Irrigation and Process Plant Availability	%	95%	
Solution Irrigation and Process Plant Operating Hours/Year	h/a	8,322	
Heap Leach Pad Footprint	m^2	146,000	131,000
Heap Leach Pad Thickness	m	63	73
Heap Leach Pad Stacked Density	t/m^3	1.65	
In-Heap Solution Pond Volume	m^3	26,576	
Event Pond Volume	m^3	36,373	
Irrigation Type		Single Stage	
Leach Cycle Solution Irrigation Rate	$l/h/m^2$	10	
Leach Cycle Time	days	90	
Heap Leach Solution Application Flux Rate (t of solution per t of ore)	t/t	1.64	
Pregnant Leach Solution Design Flow Rate	m^3/h	423	
Sodium Cyanide Consumption [1)]	kg/t	0.14	
Cement Consumption [2)]	kg/t	10	
Quicklime Consumption [3)]	kg/t	0.32	

Notes:
1) Weighted average. Leach plus Elution and Agglomeration
2) Weighted average. Cement is only used during agglomeration
3) Weighted average

17.2 Gold Production Schedule

Gold recovery rates for a heap leach operation are dependent on leach kinetics, crushed ore size, irrigation rates, lift height, and leach cycle times. The metal production schedule was prepared based on the mine plan, ore stacking plan, irrigation schedule, and metallurgical data from the deposit.

The annual gold production schedule is shown in Table 17.2.

Table 17.2 – Annual Gold Production Schedule

Item	Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Total
Stacked Ore										
Oxide	kt	2,000	2,500	2,272	2,374	2,244	2,028	1,198	1,220	15,836
Gold Grade	g/t	1.23	1.20	0.96	1.32	1.23	1.03	0.96	0.24	1.08
Transitional	kt	0	0	228	126	256	472	1,302	994	3,378
Gold Grade	g/t	0.00	0.00	1.59	0.55	1.66	2.37	0.82	0.47	1.04
Total	kt	2,000	2,500	2,500	2,500	2,500	2,500	2,500	2,214	19,214
Gold Grade	g/t	1.23	1.20	1.01	1.28	1.28	1.28	0.89	0.34	1.07
Stacking Schedule										
Oxide										
Gold Stacked	k oz	79.3	96.6	69.9	100.4	89.0	67.3	37.1	9.4	549.0
Gold Recovered	k oz	66.8	81.3	59.3	85.6	75.7	57.6	32.0	8.0	466.2
Transitional										
Gold Stacked	k oz	0.0	0.0	11.6	2.2	13.6	35.9	34.4	14.9	112.7
Gold Recovered	k oz	0.0	0.0	8.3	1.6	9.7	25.8	24.8	10.7	80.8
Total										
Gold Stacked	k oz	79.3	96.6	81.5	102.6	102.6	103.3	71.5	24.3	661.7
Gold Recovered	k oz	66.8	81.3	67.6	87.2	85.4	83.3	56.8	18.6	547.0
Cumulative Values										
Oxide										
Gold Stacked	k oz	79.3	175.8	245.7	346.2	435.2	502.5	539.6	549.0	
Gold Recovered	k oz	66.8	148.1	207.4	293.0	368.7	426.3	458.2	466.2	
Gold Recovery	%	84.3%	84.2%	84.4%	84.7%	84.7%	84.8%	84.9%	84.9%	

Item	Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Total
Transitional										
Gold Stacked	k oz	0.0	0.0	11.6	13.9	27.5	63.4	97.8	112.7	
Gold Recovered	k oz	0.0	0.0	8.3	9.9	19.6	45.4	70.2	80.8	
Gold Recovery	%			71.4%	71.3%	71.3%	71.5%	71.8%	71.8%	
Total										
Gold Stacked	k oz	79.3	175.8	257.4	360.0	462.6	565.9	637.4	661.7	
Gold Recovered	k oz	66.8	148.1	215.8	302.9	388.3	471.6	528.4	547.0	
Gold Recovery	%	84.3%	84.2%	83.8%	84.1%	83.9%	83.3%	82.9%	82.6%	

17.3 Process Description

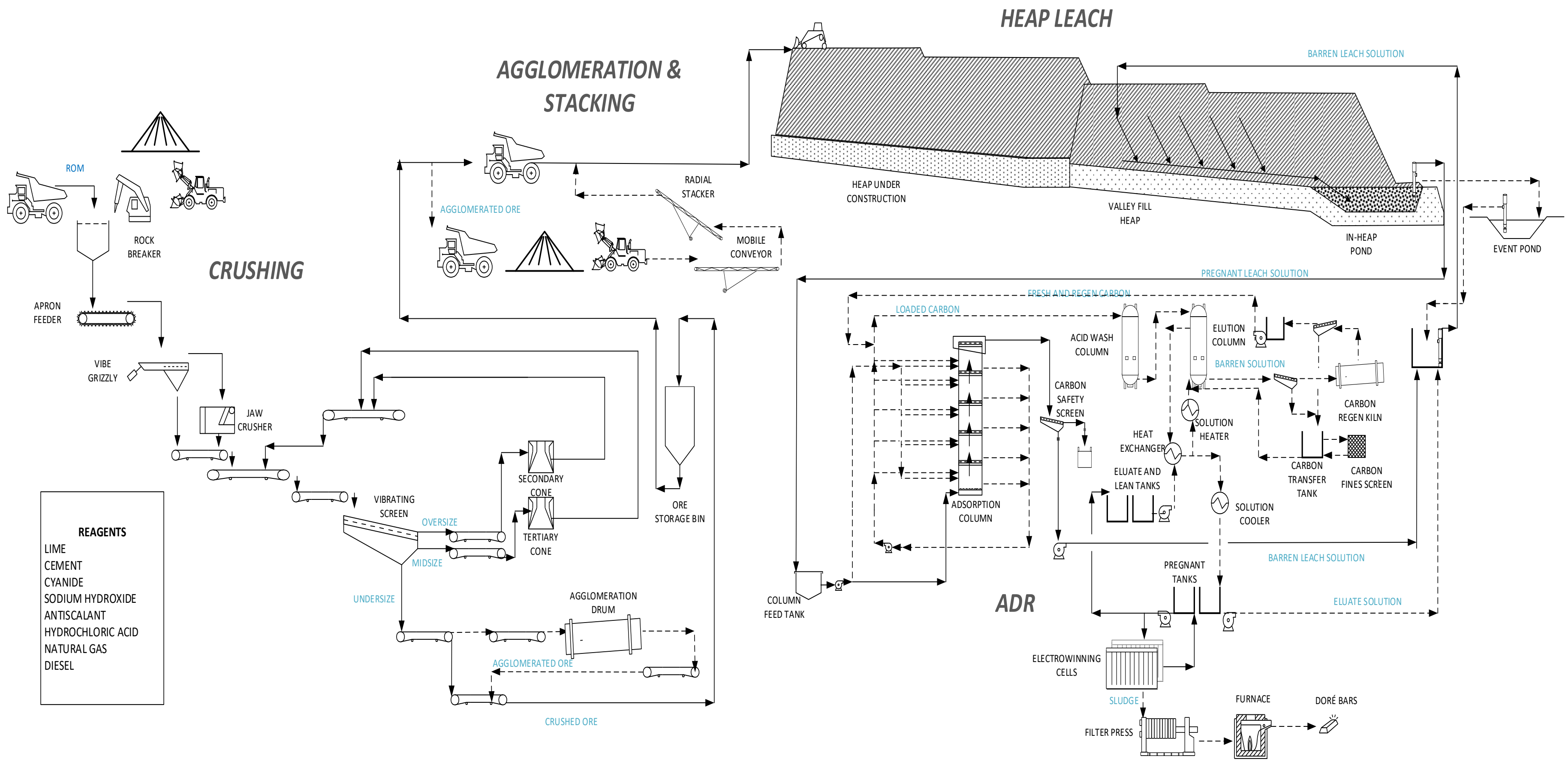
The overall process flow diagram is summarised in Figure 17.1.

17.3.1 Process Overview

The process facilities will consist of the following major areas and unit operations:

- Crushing - a three-stage crushing will be employed to produce a crush size P_{80} of 17 mm (P_{100} of 25 mm).
- Conveying - crushed agglomerated ore will be transported using belt conveyors to the ore crushed storage bin from which it will be loaded into the trucks by means of the feeder, or simple direct dumping from a loadout bin.
- Agglomeration - the agglomeration circuit will operate intermittently during processing of the high clay ore types. In this regime, the crushed ore will be diverted to the agglomeration circuit and mixed with lime, cement and cyanide in the agglomeration drum, and then conveyed to the ore storage bin for further placement onto the heap leach pad.
- Stacking - crushed ore stacking will be accomplished by direct truck dumping on the leach pad. The truck dumped ore will then be spread by a bulldozer and ripped ahead of placing the solution irrigation pipes. The agglomerated ore will be delivered to a pad area by truck, unloaded onto the ground and distributed over the leach pad by means of the front-end loader, mobile conveyor and a mobile radial stacker.
- Heap Leach Pad - the HLF pad will have a composite liner system which includes a low permeability underliner layer overlain by a geomembrane liner to prevent any leakage of cyanide solution. During Phase 2, the leach pad footprint will be expanded and additional lifts will be added to stack ore on.
- Irrigation/Leaching - BLS will be pumped to the irrigation piping network and released onto the heap using drip emitters placed in a grid pattern on the heap surface.
- Solution Collection – a solution collection system will be installed within the HLF, which consists of a piping network installed directly on the geomembrane liner and covered by with coarse crushed overliner material. The purpose of the solution collection system is to collect and facilitate the solution conveyance within the HLF to the solution pumping system in a timely manner and reduce the potential for solution head to be generated on the liner. The solution collection system is comprised of a perforated piping network embedded within a free draining granular overliner material. PLS will be collected in the PLS pond and pumped to the ADR plant.

Figure 17.1 – Overall Process Flow Diagram



Source: DRA, 2020

- The integrated PLS pond is located at the foothill (toe) of the heap leach pad and consists of a double lined solution storage equipped with two HDPE liner to prevent any leakage of cyanide solution into the environment. The PLS pond is constructed with a leak detection system installed between the liners to alert the operators if any leaks develop in the upper liner. The lower liner is referred to as the "secondary" liner and the upper liner the "primary" liner. Layered between the two liners is the leak detection system, also referred to as a leak detection and recovery system. The pond liner and a leak detection system details can be found in Section 18.1.2.2.
- The solution is then pumped to the ADR facility. In the event of any overflow from the PLS pond solution will gravitate into the event pond located downstream.
- Stormwater Containment - the integrated PLS pond and solution event pond which will be located downstream of the PLS pond will provide containment of leach solution drain down during a 24-hour power outage and runoff from a 100-year, 24-hour storm event. Each of the ponds allows for at least a 0.5 m freeboard.
- Adsorption - PLS will be pumped through a Vertical-Carbon-In-Column (VCIC) circuit to allow for the adsorption of gold and any silver onto activated carbon. BLS will discharge from the VCIC circuit and will be pumped to the BLS tank.
- Desorption - acid washing of the loaded carbon with dilute hydrochloric acid will be used to remove most inorganics and siliceous material from the carbon prior to eluate stripping using a hot caustic-cyanide solution. A split Anglo-American Research Laboratories (split-AARL) elution circuit will be used to recover gold and silver from the loaded activated carbon into the pregnant eluate solution. Gold and silver in the pregnant eluate solution will be recovered onto stainless steel cathodes via electrowinning and removed from the surface using a high-pressure washer as a high gold-grade electrowinning sludge. The eluted carbon will be regenerated at temperatures between 650 and 850ºC using a natural gas fired regeneration kiln before being returned to the VCIC circuit. This regeneration process will remove organic foulants from the stripped (eluted) carbon.
- Recovery - electrowinning sludge is washed from the cathodes using a high-pressure washer and will be filtered using a filter press.
- The filter cake will be dried and any off-gas treated in a mercury retort to remove and capture any vaporized mercury. Any vaporized mercury will be recovered by condensers, and securely stored at the site. The dried cake will then be mixed with fluxes and smelted in a natural gas smelting furnace and poured into final doré bars.

17.3.2 PRIMARY CRUSHING

ROM ore will be transported in mine trucks to the crushing plant where it will be dumped into a 150-tonne feed bin equipped with a static grizzly. Any oversized rocks not passing through the grizzly will be broken with the use of a rock breaker. An apron feeder will reclaim ore from the bin, and a vibrating grizzly feeder will feed the primary jaw crusher.

The jaw crusher will be a 1200 mm x 830 mm opening crusher, model CJ412 or equivalent, and will operate with a closed side setting (CSS) of 90 mm. The jaw crusher and vibrating grizzly feeder undersize will be combined and conveyed to the vibrating screen deck.

17.3.3 SECONDARY AND TERTIARY CRUSHING

The vibrating screen will be a 2.4 m by 6.2 m double-deck 10 deg inclined linear motion step deck screen with 70 mm and 25 mm screen deck apertures.

The vibrating screen classifies the combined feed stream combined from the primary, secondary and tertiary crushers discharge into:

- Top deck oversize: secondary crusher feed;
- Bottom deck oversize: tertiary crusher feed; and
- Bottom deck undersize: crushed product.

Each of the cone crushers is fed from a dedicated 60 t crusher feed bin via a dedicated belt feeder (one (1) bin and one (1) belt feeder per crusher).

Secondary and cone tertiary crushers will be CH660 or equivalent machines with 25 mm and 15 mm CSS respectively and will provide a sufficient size reduction to the feed material. The cone crushers discharge onto a belt conveyor, combined with the primary crusher discharge, and are conveyed to the vibrating screen for the size separation.

A dust collection system will be installed to minimise dust generation throughout the crushing and screening circuits while provisions for dust suppression water sprays have been allowed for in the ROM ore bin, transfer chutes, and in the screen-house area.

17.3.4 CONVEYING AND AGGLOMERATION

The crushed ore (crusher screen undersize) will be reclaimed by the belt conveyor and discharge into the 150-t ore storage bin. Powdered quicklime will be metered onto the discharge conveyor for pH control in the heap and leach solutions.

The ore will be loaded from the bin onto the trucks by means of a feeder.

An agglomeration circuit will be installed prior to the ore storage bin and will be used intermittently to treat the high clay ores.

Powdered Portland cement will be metered onto the conveyor, and the ore, quicklime, and cement will be fed to the 2.5 m diameter by an 8 m long agglomeration drum. The agglomeration drum will include a sodium cyanide solution spray to provide moisture and to assist in pre-leaching of the ore. In the agglomeration drum, the cement, lime, ore, and solution mixture will tumble and produce the agglomerates. Cyanide addition in the agglomeration circuit is a globally accepted practice. The cyanide delivery to the agglomeration sector will comply with safety requirements with regards to pipe specifications and design, valves, controls, and leak prevention and detection safety protocols.

During the feasibility stage, the design of agglomeration area should be reviewed from the point of compliance with the ICMC to ensure safe operation.

Agglomeration allows fines to adhere to larger particles to prevent their mobilisation within the heap and help maintain heap permeability and stability. The cement binding the agglomerates is permeable to the leach solution, which allows for proper leaching of the agglomerated particles. The agglomeration drum discharge will be conveyed to the crushed ore load-out bin.

Lime and cement will be delivered via truck and will be transferred to their respective silos via on-board compressors. A 20-t cement silo will be installed to provide sufficient storage capacity for agglomeration circuit. A lime silo of same size will be installed to provide lime supply to agglomeration.

17.3.5 ORE STACKING

The crushed ore will be delivered to the heap leach pad area by trucks which will dump the ore in the dedicated placement area. The ore then will be spread over the leach pad by means of a bulldozer. The ore needs to be ripped ahead of placement of the solution irrigation pipes.

Agglomerated ore will be trucked from the load-out bin to the leach pad and unloaded on the ground. The agglomerates placement and distribution around the heap leach is carried out by means of the front-end loader which reclaims agglomerates and feeds them into a feed hopper of the horizontal mobile conveyor. The horizontal mobile conveyor feeds the agglomerates to a radial mobile stacker to be distributed over the leach pad. Both the horizontal mobile conveyor and a radial mobile stacker are located on the heap under construction.

Ore will be stacked on the heap in accordance with the stacking plan. The ore and agglomerates will be stacked in 8m lifts to a maximum height of 80 m.

Once an area has finished leaching and is sufficiently drained, a new lift can be stacked over the top of the old lift. The old lift will be cross-ripped prior to stacking new material on top of any old stacked area, or access road/ramp to break up any compacted or cemented sections.

17.3.6 Irrigation and Solution Recovery

After a sufficient quantity of ore has been stacked, the irrigation system will be installed. The buried dripline emitters will be used to apply a dilute cyanide solution. Solution will be applied using a single stage irrigation regime.

BLS will be pumped from the respective barren solution tank via the vertical multistage pumps. The solution will be transported via the irrigation pipe network to the respective areas undergoing leaching. Since the site terrain is mountainous, the layout of the heap leach process areas results in total dynamic head (TDH) for the solution pumps higher than 100 m. The double lift solution pumping system of the intermediate tanks (20 m^3 for BLS and 12 m^3 for PLS) and pumps is required to avoid high pressures in the solution piping to and from the heap leach area. Standby pumps are not required as the pump type is highly reliable with a sufficient quantity of spares available on-site and planned preventative maintenance.

The leach cyanide solution dissolves the gold and silver as it percolates through the ore and onto pad overliners. The details of the liner system design can be found in Section 18. The PLS will be collected in perforated piping at the base of the leach pad, and gravity drains to the in-heap solution pond. PLS from the in-heap pond will then be pumped to the VCIC circuit adjacent to the ADR facility.

Pregnant solution will flow through the VCIC columns to load the soluble gold and silver onto activated carbon. Barren solution exiting the VCIC columns will be pumped to the BLS tank where make up cyanide, water and anti-scalant will be added, as required.

The in-heap pond is located at the toe of the pad. The event pond is located outside the pad and downstream of the in-heap pond. The event pond will be used to capture overflow from the BLS tank and in heap pond during an upset condition. Any solution collected within the event pond will be pumped to the VCIC circuit and returned to the circuit.

17.3.7 Adsorption

The adsorption circuit consists of a single counter-current VCIC adsorption column 3.8 m in diameter with an overall height of 10.4 m. The column will consist of six stages, each with 4.5 tonne carbon capacity of 6–12 mesh granular activated carbon. Each stage will be separated by a bubble cap plate, each of which will consist of a solid steel plate with holes. Each hole will have an offset bubble cap installed above it. The design of the overall column and individual plates is such that it will allow for upward solution flow and no backflow of carbon.

The adsorption column will be used for extracting gold and silver contained in the PLS. The leached gold and silver will be adsorbed onto the activated carbon. The PLS will enter the bottom of the columns where it will first contact activated carbon with the highest carbon gold grade.

As the solution flows from stage to stage, the carbon gold grade will incrementally decrease, leaving the lowest-grade carbon in the final or top stage of the VCIC column. The upward flow of the solution in the column will fluidize the carbon, which will ensure uniform adsorption of gold and silver onto carbon. The solution, overflowing the top of the column, will be BLS and will report to the carbon safety screen-located above the BLS tank.

A provision has been included to allow for a recirculation of solution to adjust the fluidization velocity within the columns, if necessary. The carbon safety screen will collect any gold-bearing carbon that may have been carried over the top of the column. After passing through the screen, the solution will flow by gravity back to the BLS pump box.

Carbon advancement will be performed using a single centrifugal-type recessed-impellor carbon transfer pump. The pump will receive and distribute the carbon using a series of valves and piping, allowing for proper transfer between stages. The carbon will be advanced down the column in a counter-current arrangement. Loaded carbon will be removed from the circuit and pumped to the gold processing circuit. The overall circuit is designed to send 4.5 tonnes of loaded carbon to the elution circuit per strip.

17.3.8 DESORPTION

The desorption circuit includes acid washing and rinsing, elution, electrowinning, and carbon regeneration. Loaded carbon from the adsorption columns will be transferred to the acid-wash column eight times a week. The acid-wash column will be 1.27 m in diameter by 8.25 m tall with a 4.5 tonne capacity. The carbon will be washed in a dilute hydrochloric acid solution prior to being rinsed with water. The acid wash and rinse will help prevent scale or inorganic compound build-up on the carbon and allow for increased gold adsorption and desorption kinetics. The acid wash solution will be stored to be reused for a second cycle before being sent to the operational pond, while the rinse solution will be sent directly to the operational pond. The washed loaded carbon will be transferred to the elution column for gold and silver desorption.

Elution will be performed using the split-AARL process. The elution column will be 1.27 m in diameter by 8.25 m tall, with a 4.5 tonne capacity. The eluate solution containing 3% w/w sodium cyanide and 2% w/w sodium hydroxide will be recirculated through the column and heaters as the column heats up. Once the column reaches the target temperature of 120°C, the full volume of hot eluate solution and a full tank of hot lean solution will be pumped through the column and transferred to the pregnant eluate tanks. As pH conditioning occurs within the elution circuit with sodium hydroxide, the pregnant eluate tanks receive a product with the correct pH for electrowinning. Once this is complete, the column will be rinsed with water and the water collected in the lean solution tank for the next cycle.

Once elution is complete, the column will be filled with water, and the carbon transferred to the carbon dewatering screen to be dewatered and transferred into the carbon regeneration kiln feed hopper. 100% of the carbon will be regenerated in a natural gas fired 250 kg/h carbon regeneration kiln at temperatures between 650°C and 850°C. The regenerated carbon will discharge from the kiln onto the carbon fine screen to remove any carbon fines generated during the elution and regeneration processes before being quenched in raw water. Any fresh makeup carbon will be introduced into the carbon attrition tank where it will be agitated in water and discharged onto the carbon fine screen at this stage and screened prior to use.

All carbon fines will be washed into a tank and continuously pumped through the carbon filter. The regenerated and fresh carbon will be pumped back to the adsorption circuit via a carbon transfer pump. The carbon regeneration circuit is sized to regenerate eight cycles per week. In periods with increased elution cycles, regeneration will be performed every second cycle.

Spillage within the modular plant's gold desorption and gold recovery sections, namely elution, electrowinning, will be collected within a common area sump pump.

17.3.9 Recovery

The contents of the pregnant eluate tank will be pumped to two electrowinning cells, each containing 12 stainless steel cathodes (800 mm x 800 mm). During this process, gold and silver will be electrolytically plated onto stainless steel mesh cathodes as a sludge. The solution will continue to circulate through the electrowinning cells until the eluate gold concentration decreases to an acceptably low target level (<5 ppm Au). The solution will then be pumped to the eluate or lean tank for reuse or will be discharged to the pregnant solution tank. The cathode mesh will be removed from the electrowinning cell and pressure washed.

The sludge will be collected and pumped to the sludge filter press. The filtered sludge will be placed on trays and transferred to the dedicated mercury retort oven where the sludge will be dried, and mercury vaporised, condensed, and collected. The dried sludge will be weighed and mixed with fluxes prior to being smelted in the baring furnace. Once smelted, the furnace contents will be poured into doré bars, and, once cooled, cleaned, and weighed, the doré bars will be transferred to the vault.

All fumes, off-gases, or vapors from electrowinning and smelting will be extracted and passed through a wet scrubber. All mercury vapors will be collected for proper disposal before final off-gas is released to the environment, as assays indicate ore Hg content of 1-3 g/t and Hg leachability testwork is not currently available. The risk of mercury vapor emission is mitigated by including these measures in the design.

The gold room will be a dedicated area with increased security, containing all required gold-refining equipment. It will also include the vault and a secure enclosed area for transferring doré bars to the security transport vehicles.

17.4 Reagents

The average reagent consumption rates are listed in Table 17.3 is calculated based on metallurgical testwork and first principles. The heap leach reagent consumptions by ore type were determined based on the metallurgical testwork results discussed in Section 13.

Table 17.3 – Reagent Consumption

Reagent	Usage	Delivery	Annual Consumption t/a 1)	Consumption per tonne of Ore kg/t 1)
Sodium Cyanide	Leaching, elution, and agglomeration	Truck, SLS system or pellets in soft containers	361	0.14
Cement	Agglomeration	Bulk tanker, powdered Portland cement	25,000	10.0
Quicklime	pH modifier for heap leach and agglomeration	Truck, powdered quick lime, 92% CaO	811	0.32
Sodium Hydroxide	pH modifier for elution, VCIC, electrowinning, cyanide preparation	Bulk tanker, 40-50% solution	214	0.09
Hydrochloric Acid	Carbon washing prior elution	Bulk tanker, 33% acid	129	0.05
Coconut Shell Granulated Activated Carbon 6 x 12 mesh	Gold adsorption	Truck, soft containers	38	0.015
Anti-scalant	Leaching	Solution, delivered in 1 m^3 tote bins	25	0.01

Notes:
1) Weighted average based on ore type and ore tonnage

17.4.1 Sodium Cyanide

A cyanide system will be based on the SLS technology which considers delivery of isotainer with a solid sodium cyanide briquettes to site. Once an SLS container is delivered to site, it is connected to the dissolution station (supplied by CyPlus) via two (2) hoses (inlet/outlet). The automatic dissolution process of the solid cyanide can then be started. During this process, a high pH water is directed through the container, generating cyanide solution, which then flows directly into the site cyanide storage tank. When the dissolution process is finished, the whole SLS container is automatically rinsed with water and purged with air.

The concentration of the cyanide solution can be adjusted according to the process requirements. A schematic depiction of the SLS system can be seen in Figure 17.2.

Figure 17.2 – CyPlus SLS System Schematic

Source; CyPlus 2020
1 – SLS container; 2 – dissolution station; 3 – cyanide storage tank

The cyanide preparation system was designed for preparation of the solution using SLS technology and from the cyanide pellets delivered in soft containers or drums in case of the SLS containers supply disruption.

The 30% strong cyanide solution will be stored in the site storage tank and diluted to the required solution strength prior to distribution to heap leach, elution, and agglomeration area as per the plant flowsheet.

Any spillage will be contained within the area bund and collected by means of the spillage pump. Hydrogen peroxide will be stored in 1 m³ totes in case emergency cleanups are needed in sectors where cyanide is used. Sodium hydroxide solution will be applied for pH control during the spillage decontamination. The spillage containment system will be designed and constructed according to the International Cyanide Management Code guidelines.

17.4.2 Anti-Scalant

Anti-scalant tote bins will be delivered by a flatbed truck. Anti-scalant metering pumps will distribute the reagent to the process.

17.4.3 Sodium Hydroxide

Sodium hydroxide will be used to raise the pH within the elution and the VCIC. Sodium hydroxide is also added to the cyanide mixing tank to control the pH.

A tanker truck will transport 50% w/w sodium hydroxide solution to the processing plant. Upon arrival, sodium hydroxide solution will be transferred to the sodium hydroxide tank. The sodium hydroxide will be pumped to the elution circuit via the sodium hydroxide distribution pump. Prior to use, it will be diluted with water at the discharge of the distribution pump. To control any spills, the system will be contained in a bunded area equipped with a sump pump.

17.4.4 HYDROCHLORIC ACID

A tanker truck will transport 33% w/w hydrochloric acid solution to the processing plant. Upon arrival, the acid will be transferred to the hydrochloric acid tank. The acid will be pumped to the acid-wash column via the hydrochloric acid distribution pump. Prior to use, the acid will be diluted with water at the discharge of the distribution pump. To control any spills, the system will be contained in a bunded area equipped with a sump pump.

17.5 Utilities and Services

17.5.1 NATURAL GAS

Liquified natural gas will be transported by tanker truck to the processing plant. Upon arrival, the natural gas will be transferred to a natural gas day tank. A natural gas vaporiser is included to provide adequate flow of natural gas to the process.

17.5.2 DIESEL

Diesel will be transported by tanker truck to the processing plant. Upon arrival, the diesel will be transferred to a diesel storage tank and distributed to the plant area tanks by means of the site fuel trucks.

17.5.3 WATER SERVICES

The primary source of fresh makeup water for the site will be from the local water source outside the plant envelope. Raw water will be used as process makeup, mining, dust suppression, and feed water to the potable water treatment plant. Surface water from various contact water ponds and mine dewatering waters will act as an additional source of water and will be pumped to the event pond and will be reclaimed to the heap leach.

Makeup requirements for the process facilities are estimated at 42 m^3/h on average with 61 m^3/h maximum for the dry season.

Raw water pumps will be used to distribute the raw water throughout the facilities.

A potable water treatment system will be included to produce potable water on site and distribute throughout the site.

The site will contain a fire-water tank and a set of fire water pumps to provide the fire suppression systems of the site. The fire-water tank will be filled from the raw water pumps.

17.5.4 Air Services

The process facilities will include two (2) air compressors to provide plant air and instrument air to the processing plant.

One mobile compressor will be used for the crushing plant. The compressors will include on-board air driers. The instrument air will also be dried with a dedicated air drier. The plant air and instrument air each will have dedicated air receivers.

17.5.5 Electrical Power

Power will be provided by the local utility. Please refer to Section 18.11 for electric power system details

The total process installed electrical power is estimated at 4.1 MW, while the total process absorbed electrical power is estimated at 2.3 MW.

Electrical power consumption was calculated using expected absorbed power draw as determined for individual equipment items after applying use and electrical correction factors.

17.5.6 Laboratory Services

The site will include an assay and metallurgical laboratory equipped to perform sample preparation and assays, including fire assays, atomic adsorption, and cyanide analysis. The laboratory facility will support the process plant, as well as mining, environmental, and water monitoring. Certain metallurgical and environmental samples will be sent off-site to an external laboratory for confirmatory testing.

The following equipment and system are required for the plant laboratory:

- Crushing system and pulverisers;
- Drying oven, ventilation and fume hoods;
- Atomic Absorption (AA) machine;
- Equipment for cyanide and pH analysis;
- Columns for leaching;
- Columns for percolation tests;
- Sieve shakers;
- Bottle roll machine.

17.5.7 PROCESS CONTROL, SAMPLING

Field instruments will provide inputs to a set of programmable logic controllers (PLCs). Process control cubicles are located in the motor control centres and will contain the PLC hardware, power supplies, and input/output cards for the instrument monitoring and loop control. The PLCs will perform the control functions by:

- Collecting status information of drives, instruments, and packaged equipment;
- Providing drive control and process interlocking;
- Providing proportional-integral-derivative control for process control loops.
- Standard personal computers are located in the main control room and the crusher control room. The computers are networked to the PLCs and operate a supervisory control and data acquisition (SCADA) system that provides an interface to the PLCs for control and monitoring the heap leach process facility. The SCADA system is configured to provide outputs to alarms, control the function of the process equipment, and provide logging and trending facilities to assist in analysis of process facility operations; and
- Uninterrupted power supplies will provide operating control stations with 20 minutes of standby power.

The general control strategy adopted for heap leach processing facility is as follows:

- Integrated control via process control system (PCS) for areas where equipment requires sequencing and process interlocking;
- Hardwired interlocks for safety of personnel;
- Motor controls for starting and stopping of drives at local control stations, via the PCS or hardwired depending on the drive classification;
- All drives can be stopped from the local control station at all times. Local and remote starting is dependent on the drive class and the control mode;
- Control loops via the PCS except where exceptional circumstances apply;
- Monitoring of all relevant operating conditions on the PCS and recording select information for data logging or trending; and
- Trip and alarm inputs to the PCS is fail-safe in operation (i.e., the signal reverts to the de-energized state when a fault occurs).
- Drives that form part of a vendor package are controlled from the vendor's control panel. At a minimum, "Run" and "Fault" signals from each vendor control panel is made available to the SCADA system via the PLC. Where practical, the PCS will interface with the vendor control panel to provide full operating status, including state of all drives, alarms, and instrument outputs.

- Sampling of the ore, solutions, carbon, doré, water and chemicals will be performed as per the approved process control scheme continuously or discretely as necessary.
- Conveyor belt scales will be used to control the crushed ore tonnage shipped for the stacking, and manual sampling of the ore will be organized for the grade control and metal accounting.
- Solutions and water flow within the plant will be controlled by means of the flowmeters appropriate for the duty.
- Solution sampling within the heap leach and the VCIC will be performed by means of the continuous wire samplers.
- Sampling within the ADR facility will be organized through the vendor supplied samplers and sampling points within the equipment supplied.
- Sampling and assaying within the gold refinery will be conducted as per the approved methodologies and security protocols.
- Environmental sampling will be organised as per the local regulations and requirements of the International Cyanide Management Code.

18 PROJECT INFRASTRUCTURE

This section describes the infrastructure required to deliver the mining and mineral processes described in this Report. This section describes the main project elements related to process, followed by support infrastructure.

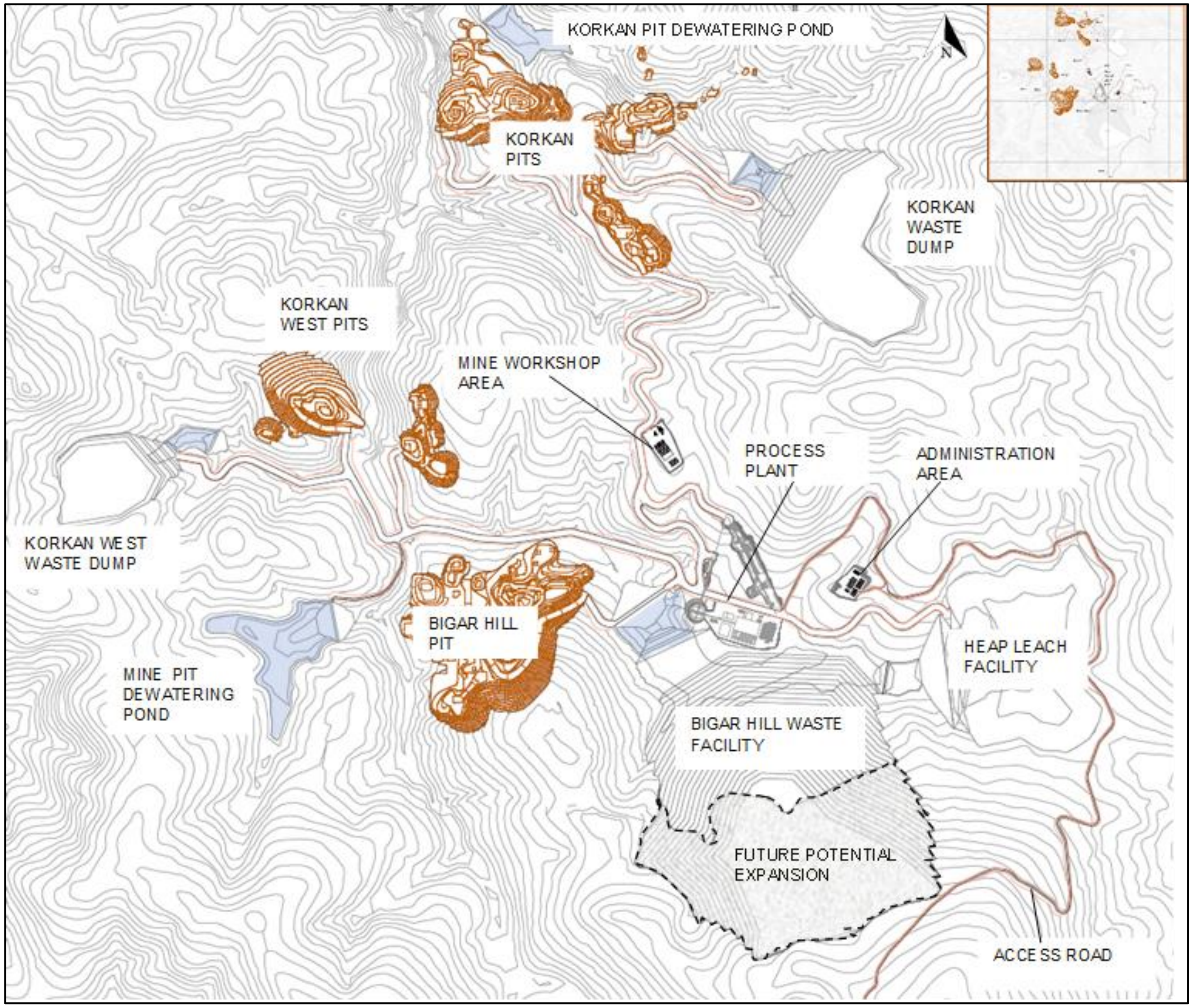
Figure 18.1 depicts the overall site layout.

This section commences with a description of the heap leach facility location and design detail as a priority processing element around which other infrastructure will be located along material flow paths. Following the heap leach location confirmation, the process plant location and design is described along with plant related services. Site terrace locations and associated process support building infrastructure descriptions follow.

The haul road network interconnects the various site elements with material flow optimisation the main priority. The positioning of the haul roads provide opportunity for the development of the electrical power reticulation network infrastructure layout that will maximise access through positioning along the haul roads.

Electrical automation, control and instrumentation is also discussed. The waste management facility location and design descriptions are described next. Finally, site wide water management design description ensures the correct water containment, treatment and control discharge infrastructure.

Figure 18.1 – General Site Layout



Source: DRA Global, 2020

18.1 Heap Leach Facility

18.1.1 Heap Leach Facility Location

A site selection study was performed to find a suitable location for the Heap Leach Facility (HLF) within the vicinity of the project mine pits namely Korkan, Korkan West and Bigar Hill. The mountainous topography of the site allowed the identification of potentially suitable locations. Singular and distributed facility options were compared against an encompassing criteria suite that included, social, environmental, economic, safety, operability, emergency response, and closure considerations. Leach pad locations were not feasible due to extensive earthworks requirements. AHLF concept was adopted, in which ore is placed and leached upstream of an embankment constructed across a valley.

A suitable location near the Bigar Hill open pit was determined as the most feasible. The final selected location for the HLF and ponds provided advantages in terms of haulage distances, surface area and liner optimisation, suitable process facility proximity, safe handling and emergency containment of cyanide solutions, and a positive economic analysis.

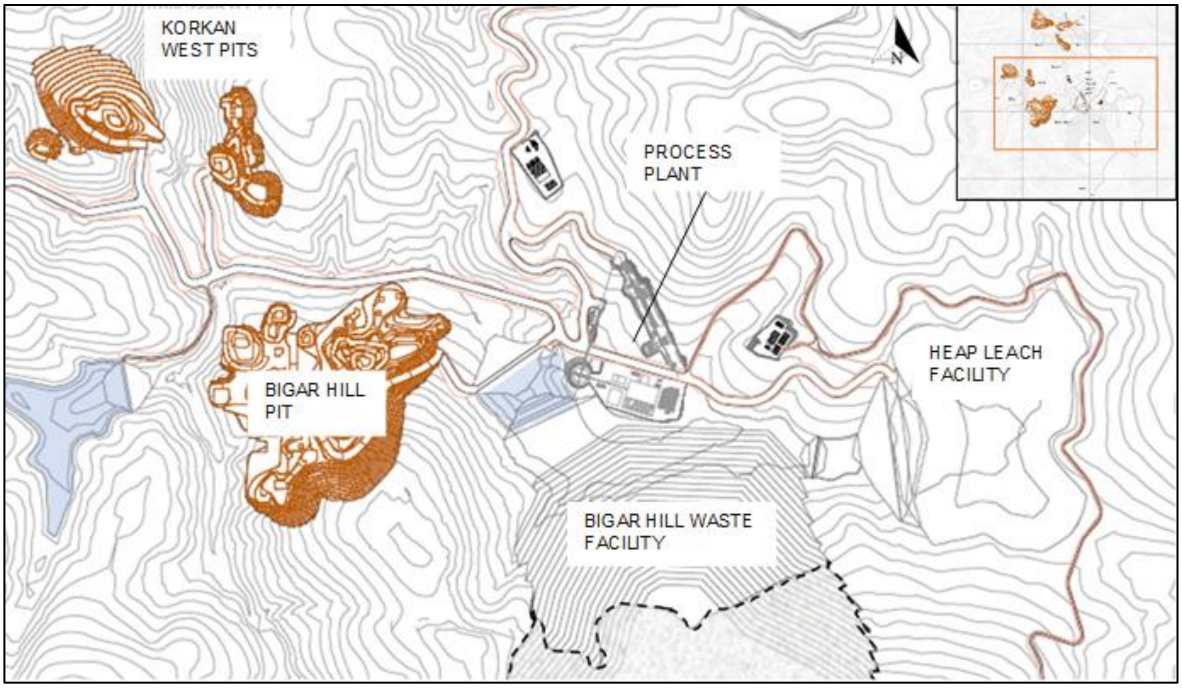
The final selected location for the HLF is indicated in Figure 18.2. Of note, no geotechnical investigations have been conducted to characterise the foundation or borrow materials for any of the facilities presented in this report, which includes the waste rock dumps, tailings co-disposal facility, HLF and associated embankments.

No geotechnical characterisation testing has been conducted on ore samples from the Project at the time of this Report.

The current location designs require the completion all related environmental studies and hydrogeological studies associated with the area. Baseline water monitoring plan has commenced, but still require actual field data to be collected and interpreted.

As no geotechnical information was available at the time of developing the design, a field and laboratory investigation program will need to be carried out as part of the next project phase to confirm the assumptions made, or if changes to the design need to be made. This program will include geophysics, drilling and test pitting in the designated area, as well as taking samples for geotechnical laboratory testing.

Figure 18.2 – Process Plant Element Locations



Source: DRA Global, 2020

18.1.2 HEAP LEACH FACILITY DESIGN

For the PFS project definition, oxide and transitional ore will be crushed, placed on the HLF and leached with a cyanide solution in order to recover gold and silver. Crushed ore is placed and leached upstream of an embankment constructed across a valley and serves as a buttress to improve the slope stability and provide containment for solution.

For this Project, the design included both an Internal Solution Pond and External Event Pond for overflow protection and liquor dilution control during storm events. The internal pond was selected on the basis of environmental and temperature benefits.

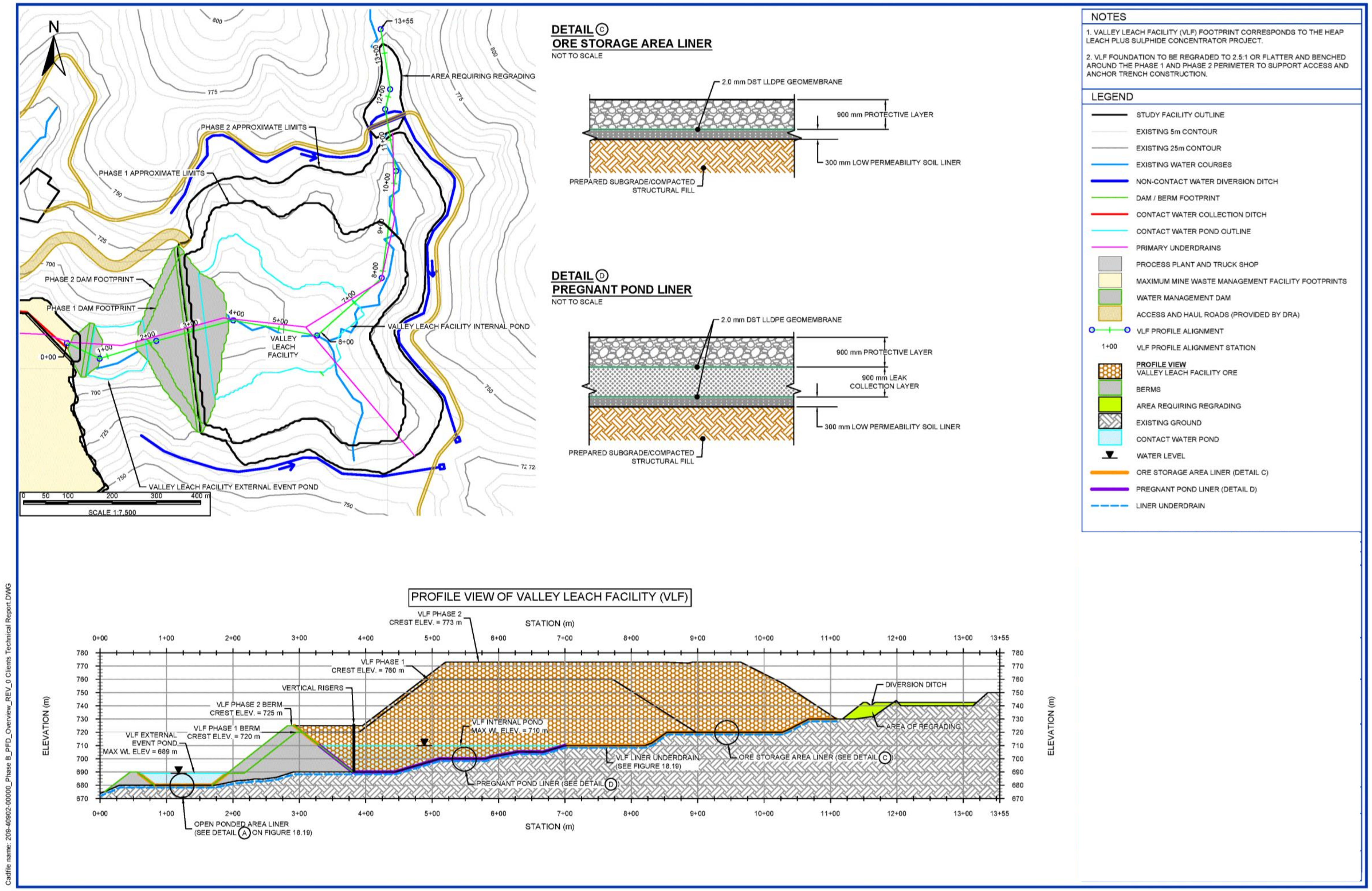
18.1.2.1 Design

The HLF, Solution Pond and Event Pond, illustrated on Figure 18.3, were designed based on the following key criteria and parameters:

- Provide capacity for 20 million tonnes (Mt) of oxide and transitional ore, with potential for future expansion.
- The ore is durable and free draining, and can be stacked to a maximum ore height of 80 m with a permeability two (2) orders of magnitude greater than the design solution application rate of 10 litres per hour per square metre (l/hr/m^2);

- Provide for secured containment of all process fluids within a liner system that protects surface and ground water, with solution pond water balance and sizing requirements provided by DRA;
- Minimise surface water runoff entering the HLF;
- Meet slope stability requirements using peak ground acceleration parameters and provide a minimum static and pseudo-static Factor of Safety (FOS) of 1.5 and 1.1, respectively. The most current reference for seismicity (AMEC, 2012), only presented a single value for the Peak Ground Acceleration (PGA) for the 1:475 return period event, which was used in the current design. Performing a site specific Probabilistic Seismic Hazard Assessment (PSHA) is recommended to determine the appropriate PGA to be used in the pseudo-static stability analysis in the FS design.
- Provide a configuration that could be phased to minimise initial capital requirements.

Figure 18.3 – Valley Fill Heap Leach Facility Phases 1 & 2 - Section and Details



Source: SLR HLF Design Report, 2020

- The Solution Pond and Event Pond have a combined capacity to contain the runoff volume resulting from the 1-in-100-year 24-hour storm event.

An underdrain system, comprised of a free draining granular material wrapped in a geotextile, will be installed under and within the HLF and External Event Pond limits, in the area of any springs or seeps to protect, the groundwater and minimise any uplift pressures on the liner system. This underdrain system will gravity flow to a point outside of the HLF where it can be monitored and discharged.

An embankment will be constructed with engineered fill at the downstream toe of the HLF to serve as a buttress and increase the overall stability, and provide containment for the storage of solution within the void spaces of the ore within the Internal Solution Pond limits.

A Solution Collection System (SCS) will be installed within the HLF, which consists of a piping network installed directly on the geomembrane in a herringbone pattern and covered by an overliner material. The purpose of the SCS is to collect and facilitate the solution conveyance within the HLF to the solution pump system in as timely a manner as possible and reduce the potential for solution head to be generated on the liner system. The SCS is comprised of a perforated piping network embedded within a 900 mm thick overliner, which is comprised of a free draining granular material.

DRA performed the Heap Leach Facility water balance that was used as a basis for pond sizing. DRA estimated the capacity of the Internal Solution Pond to be 26,576 cubic metres (m^3) and the capacity of the External Event Pond to be approximately 40,000 m^3.

18.1.2.2 Liner System

As solution is not expected to pond in the Ore Storage Area, a composite liner system is proposed for the majority of the HLF area in which ore will be placed and leached, in an area referred to as the Ore Storage Area.

The Internal Solution Pond will have solution stored on a regular basis, and have the potential to generate leakage into the environment. In order to facilitate Leak Detection monitoring from the upper geomembrane within the Internal Solution Pond, a Leak Collection Layer was included between the two (2) geomembranes. The Leak Collection Layer is monitored to remain dry through the application of level sensors within the relevant wells as well as regular well pump activation, flow determination and sampling.

- Any solution collected will gravity drain to a low point and be pumped from a Leak Collection Recovery System (LCRS). The internal pond will be lined with a double layer using LLDPE geomembrane.

As the External Event Pond will only have solution stored in the event of an extreme storm event, it will be lined with a single layer using HDPE geomembrane.

18.1.2.3 HLF Decommissioning and Closure

According to the ICMC, decommissioning considers the steps that address the cyanide remaining on site upon cessation of production, preparing the site for its closure period. These steps include the treating, neutralizing, or management of cyanide containing process solutions remaining in production facilities so that they do not present a risk to people, wildlife, or the environment upon closure. The ICMC requires a schedule for carrying out decommissioning plans. The plan does not require specific dates, but rather an order of steps to be conducted from the point in time that the operation ceases production. The operation is expected to make a reasonable attempt at scheduling its decommissioning activities.

Steps for closure of the HLF will begin with the final irrigation cycle: an irrigation of as much as possible of the facility at a lower irrigation rate, reaching the maximum capacity of the VCIC column. Irrigation would continue until the value of gold recovered is less than the cost to recover, including fixed overhead costs. The heap would then be rinsed with fresh water, topped up as required for evaporation losses. A low irrigation rate should continue to be applied. The solution would be circulated through the heaps and ponds until the pH from the heap is neutral and total cyanide concentration is low, less than the 0.5 ppm CN WAD (discharge limit according to the IFC EHS Guidelines for Mining and the ICMC). The heap would be sampled until it is determined to be detoxified of any residual cyanides or heavy metals, which would conclude the rinsing stage.

Once rinsing is complete, the heap can be capped and revegetated as per environmental specifications, and the drain down monitored until it reaches an acceptable level.

Upon the cessation of leaching, infiltration will be allowed to accumulate within the HLF. Therefore, minimising infiltration into the HLF will be critical. Installation of an Enhanced Low Permeability Cover (low permeability layer overlying the regulating layer and in turn overlain by a geomembrane, drainage layer and an additional 600 mm of subsoil and soils) is proposed, in addition to monitoring of post mining solution levels within the HLF, and possibly pumping should the solution levels reach certain thresholds.

18.2 Process Plant

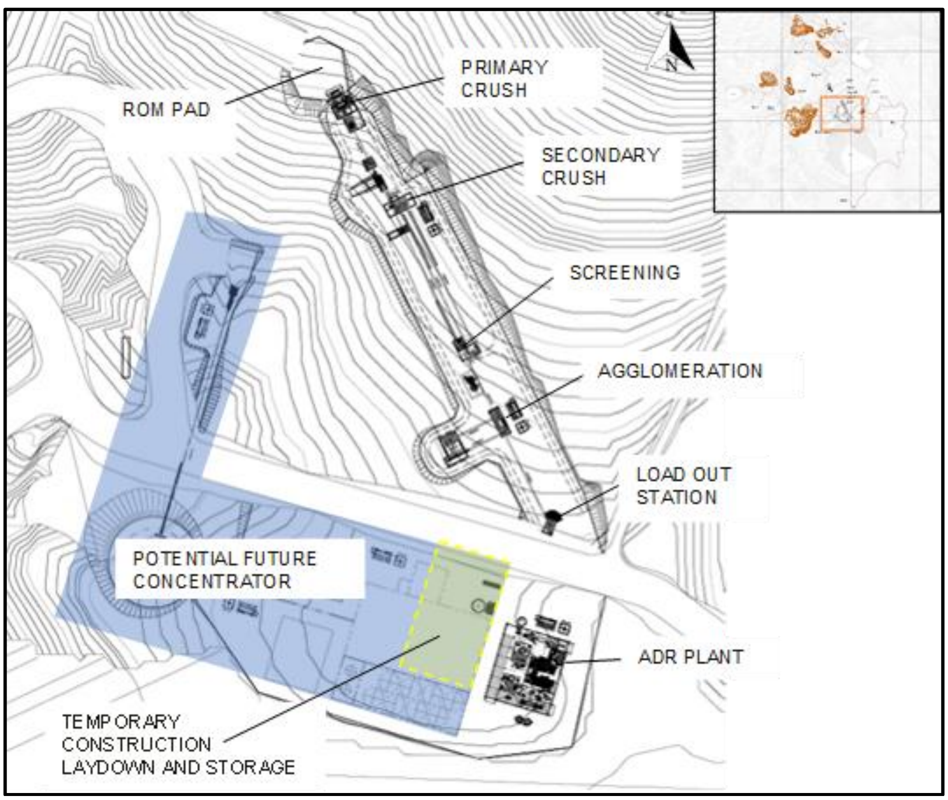
18.2.1 Plant Location

The process plant site location considered the following main criteria:

- Proximity to heap leach facility.
- Available topographical areas that will minimise the amount of earthwork for the establishment of plant terraces and connecting of roads.
- Central location and elevation from major mining pits to optimise haulage cost.

The mountainous terrain limited plant location options. A plant location closest to the main Bigar Hill mine pit and HLF was chosen. The terrain proved amenable to practical access and reasonable terrace earthwork. Refer to Figures 18.1 and 18.4 which show the process plant location and layout.

Figure 18.4 – Process Plant Layout



Source: DRA Global, 2020

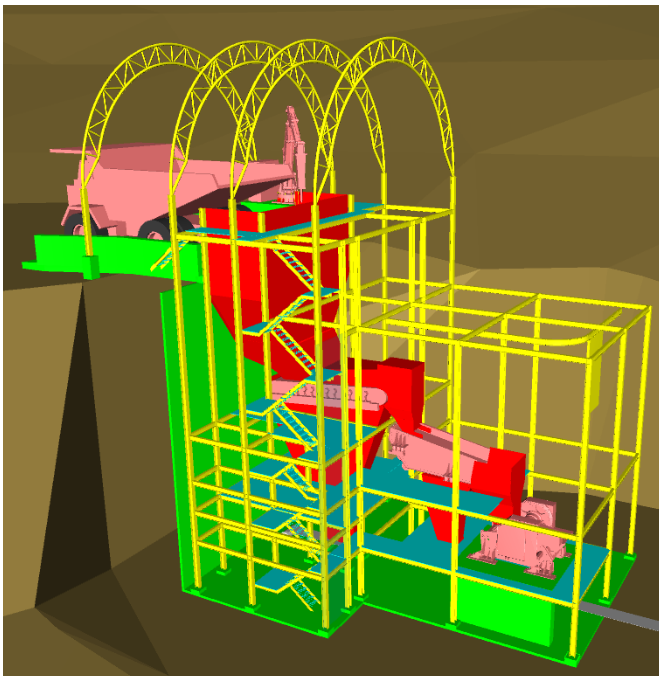
The layout of the heap leach crushing facility was selected to optimise available sloped area for larger building terraces, while taking advantage of the terrain to minimise height requirements between the process plant elements. The main processing elements associated with the heap leach processing facility will be described in the next sections.

18.2.2 Primary Crushing

The primary crusher building allows the direct tipping of mine haul trucks into a suitably sized Run of Mine (ROM) ore bin, followed by a gravity fed crushing process. The structure is sheeted, but not insulated or heated.

The primary crusher building requires the construction of a retaining wall to allow for direct truck tipping. The associated earthworks and concrete have been allowed for in the design and costing of this facility. Refer to Figure 18.5.

Figure 18.5 – Primary Crushing



Source: DRA Global 3D Model, 2020

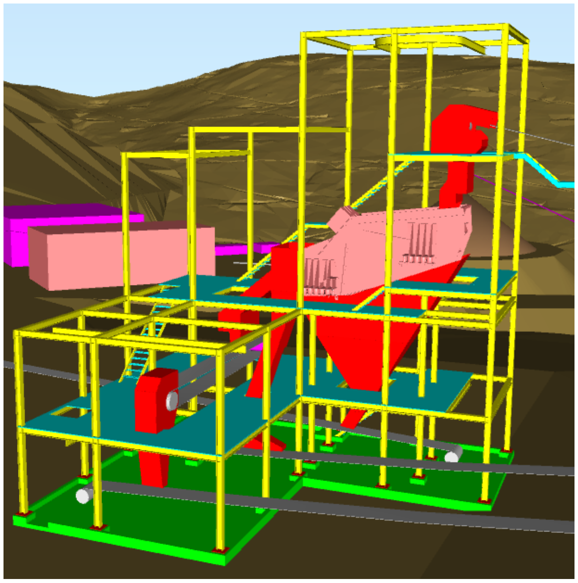
18.2.3 Screen-House

The screening building houses a double deck screen that classifies the primary crushed feed material into three (3) sized products. The building is sheeted, but not insulated or heated. Refer to Figure 18.6.

18.2.4 Secondary and Tertiary Crushing

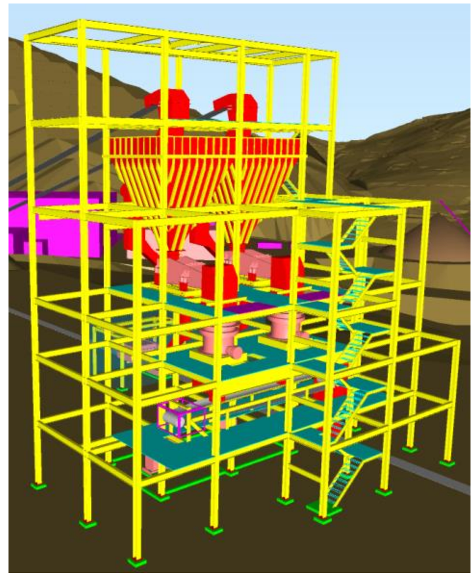
The secondary and tertiary cone crushers have been arranged such that the crushed products discharge directly on to the primary crusher feed belt, circulating back to the screening building. Refer to Figure 18.7.

Figure 18.6 – Screening Building



Source: DRA Global 3D Model, 2020

Figure 18.7 – Secondary / Tertiary Crushing Building

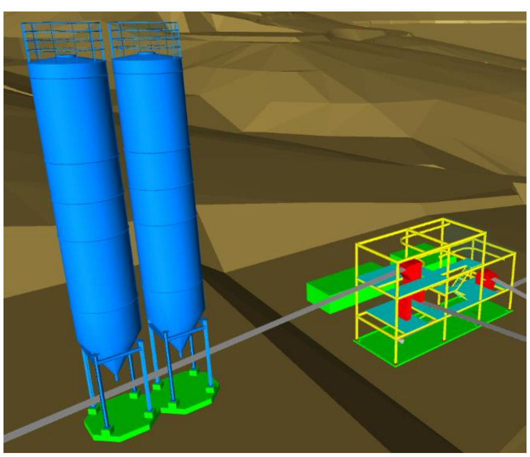


Source: DRA Global 3D Model, 2020

18.2.5 AGGLOMERATION CIRCUIT

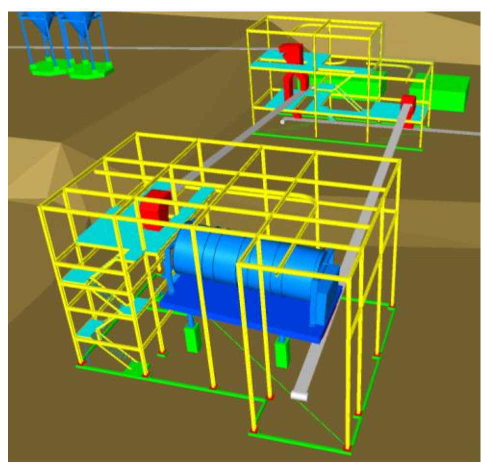
The agglomeration circuit has been included to be able to stack ores with high clay content to ensure that this material can be processed without impacting the heap leach process. The agglomeration circuit can be bypassed through a diversion gate arrangement when ore constituency does not require the agglomeration circuit. The agglomeration building will be sheeted, but not insulated or heated. Refer to Figures 18.8.

Figure 18.8 – Lime and Cement Silos



Source: DRA Global 3D Model, 2020

Figure 18.9 – Agglomeration Drum

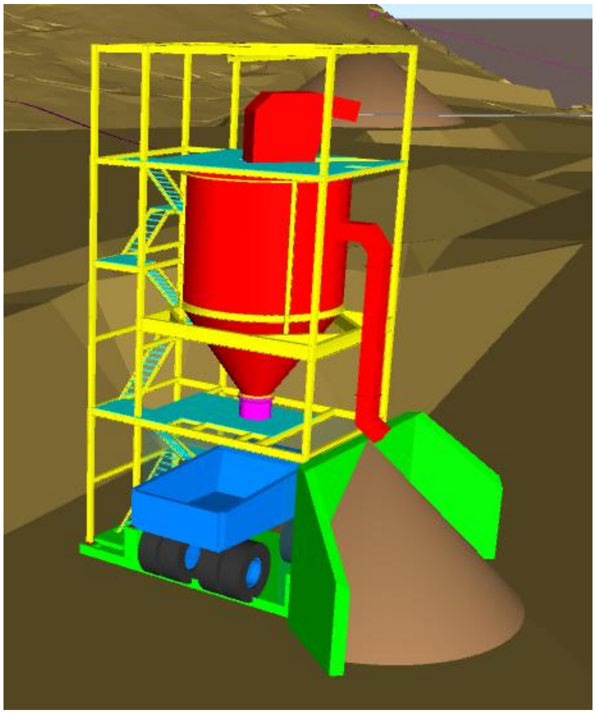


Source: DRA Global 3D Model, 2020

18.2.6 150-t Ore Storage Bin

The 150-t load out bin provides buffering capacity between the crushing process and haul truck load out. The load-out bin is sufficiently sized to accommodate 90ton trucks and include a clam shell arrangement for controlled loading of trucks. Trucks will park in a specified location and remain stationary during the loading process for safety purposes. The bin is equipped with an overflow chute to account for any non-standard operating conditions. Heap leach material addition measurement will be either via load cells installed on the load-out bin, or belt scale installed on the feed conveyor. Refer to Figure 18.10.

Figure 18.10 – 150-t Storage Silos



Source: DRA Global 3D Model, 2020

18.2.7 Gold Room / ADR

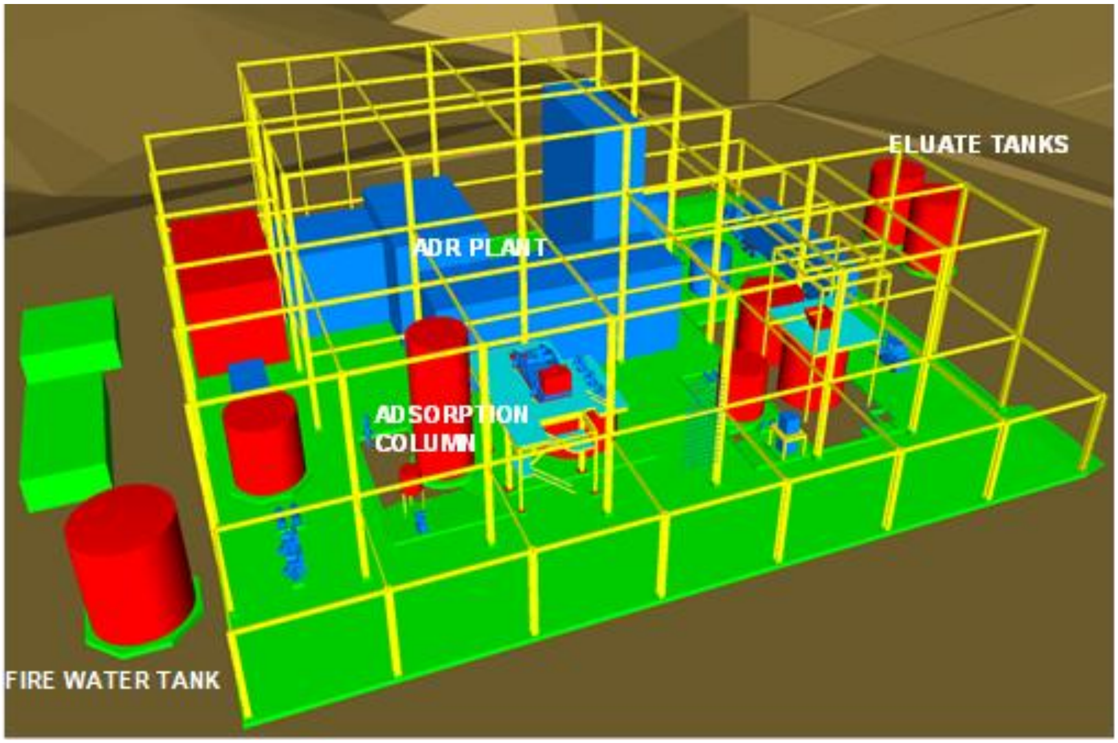
The integrated Gold Room/ADR building houses the entire refinement process for the gold-silver bearing PLS returning from the heap leach pad as depicted in Figure 18.11.

The building has drive-in areas for reagent off loading and doré loading, integrated security and gold processing areas. Refer to Section 18.3 for security descriptions.

The building is full sheeted and insulated for heating purposes. The building is approximately 2,365m^2 under roof measuring 42 x 52 m.

External to the building a raw water and pregnant eluate tanks are present.

Figure 18.11 – Gold Room / ADR Facility



Source: DRA Global 3D Model, 2020

18.3 Site Security

The Project site will require security measures to prevent unauthorised access. The Project site encompasses a wide area and numerous large project elements. The mountainous terrain does not easily allow for the fencing of the entire site.

The area remains rural farmland and is sparsely and seasonally occupied. Multiple points of ingress via forest tracks and roads exist. DPM is planning to purchase all properties within the affected mine area toward consolidating and provide sufficient boundaries between mine operations and inhabited properties.

18.4 Camp Site Accommodations

No accommodation camp will be present on-site. Sufficiently established towns are located within reasonable distance and travel time from the Project site. Operational staff will be required to relocate to a local town of their choice.

18.5 Site Buildings

Site buildings are required to provide secure work and storage areas. Fit for purpose modular structures are selected over stick-built brick and steel structures to minimise cost and construction man-hours at site.

18.6 Site Services

18.6.1 Process Raw Water Supply

Process related raw water supply will be drawn from the Bigar Hill waste dump pond and has been noted in the Site Wide Water Balance (Section 18.15).

The raw water system will consist of a floating barge with a submerged pump capable of operating during the winter period as well. Process raw water will be added to the process water tank which will also double as the fire water tank.

18.6.2 Potable Raw Water Supply

Potable water supply will be for human consumption use only.

A suitable borehole location will be identified from which water can be extracted. A small-scale water treatment and monitoring plant will be installed at the plant area along with a header tank and buried distribution system.

The mining and administration terraces will each have an additional header tank with its own buried distribution network, fed from the water treatment plant.

18.6.3 Compressed Air

Cost allowance for compressed air distribution has been included. Compressed air will be individually produced at the process plant and mine workshop terrace areas using dedicated compressors to avoid the requirement for buried piping.

18.6.4 Fire Systems

A buried fire water system has been allowed for in the capital costs. The plant raw water tank will be segmented to provide a suitable volume of dedicated fire water, attached to suitably arranged duty/standby fire water pump manifold.

Suitably insulated fire hydrants will be present throughout the site and distributed terraces, connected to the underground distribution system.

Buildings will be equipped with fire hoses and fire extinguishers at every floor.

18.6.5 Sewage Treatment

A centralised packaged sewage treatment plant has been included at the process plant terrace. The sewage treatment plant will process sewage and grey water run-off from all ablution facilities. The use of a piped or septic tank system will be evaluated in the next phase toward optimizing the design. A cost allowance has been included for a sewage treatment plant system of sufficient capacity for the site personnel compliment.

18.6.6 Waste Disposal

A suitable domestic waste disposal site has not been identified as part of this Report. A waste management strategy will be developed as part of the next project phase as to the correct on and off-site disposal for waste. It is recommended that a potential domestic waste dump facility be identified early on in the next study phase to ensure the location is recorded and real estate reserved. Specialised third-party contractors will be identified that can safely dispose of hazardous waste. Refer to Section 20 for further details.

Industrial and hazardous waste will be stored in suitably contained areas for off-site disposal. It is assumed suitable facilities are available in the region for disposal of hazardous waste due to ongoing heavy industry in the region. It is recommended to identify and negotiate with such facilities during the next project phase.

18.7 Terraces

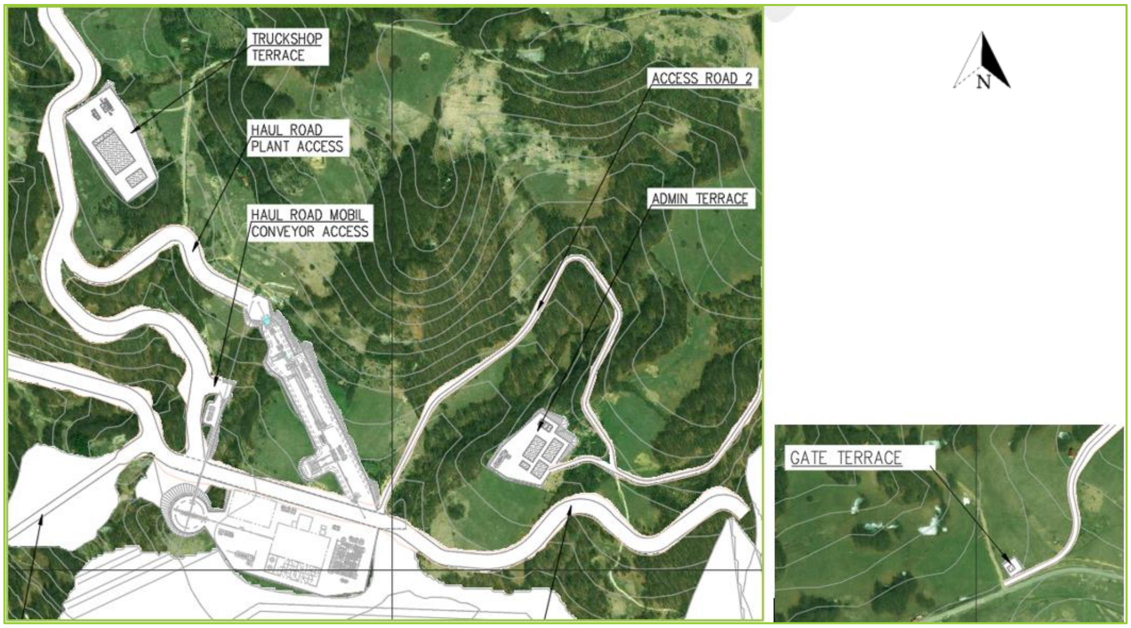
The Project site requires terraces areas to accommodate the project infrastructure and equipment. Terraces have been designed through an iterative process between project element layout and suitable topography and terrain.

18.7.1 Terrace layout

There will be some independent terraces, the process area with the crushing circuits (fix and mobile) and some surrounding terraces for Truck shop, administration buildings and gate house. Figure 18.12 depicts the layout of the site terraces.

The terraces are designed with grading and a drainage system that will allow the runoff water to reach the collection ponds and natural drains in accordance with the site water management requirements.

Figure 18.12 – Site Terraces



Source: DRA Global, 2020

18.7.2 Parking Facilities

There will be several gravel parking facilities in different areas across the sites, such as visitor parking at the gatehouse, and personnel /bus parking at the administration buildings. Parking is incorporated into the terrace areas.

18.7.3 Terrace Water Management

The surface drainage system will consist of a combination of cut-off berms, drainage trenches, culverts, and elevated roads throughout the site. Stormwater controls have been designed to route up-gradient runoff around the proposed process and mine infrastructure to accommodate and contain on-site runoff from design storm events. The intent of the stormwater controls is to:

- Divert non-contact water (i.e., water that has not encounter disturbed ground or process solutions) around the process and mine facilities and discharge to downstream watercourses.
- Convey sediment-laden runoff (i.e., water that comes off stripped surfaces and roadways), as necessary to sediment collection basins prior to discharging to downstream watercourses.
- Contain precipitation from a design storm event that has encounter process solution or disturbed ground.
- The contact water ponds will receive all water falling within the process and mine internal catchment areas. The contact water pond levels will be controlled and pumped to be used as make-up to the process.

- All water related designs have been sized to withstand a one-in-a-100 year, 24-hour storm event.

18.8 Roads

18.8.1 Site Access Roads

The Project site will be accessed via a new road coming from the south of the existing public road 105 to the administration area. The road will be oriented north-south and will provide gravel access with a typical width of 6 m and will be approximately 4.5 km long. An alternative access road route for the first 1.8 km of the access road was proposed following the initial design and quantification of the access road. The alternate route is preferred by DPM and will be incorporated during the next phase of the Project. The quantities estimated for the initially designed road will remain within the estimate as it is deemed accurate and sufficient for either access road routings.

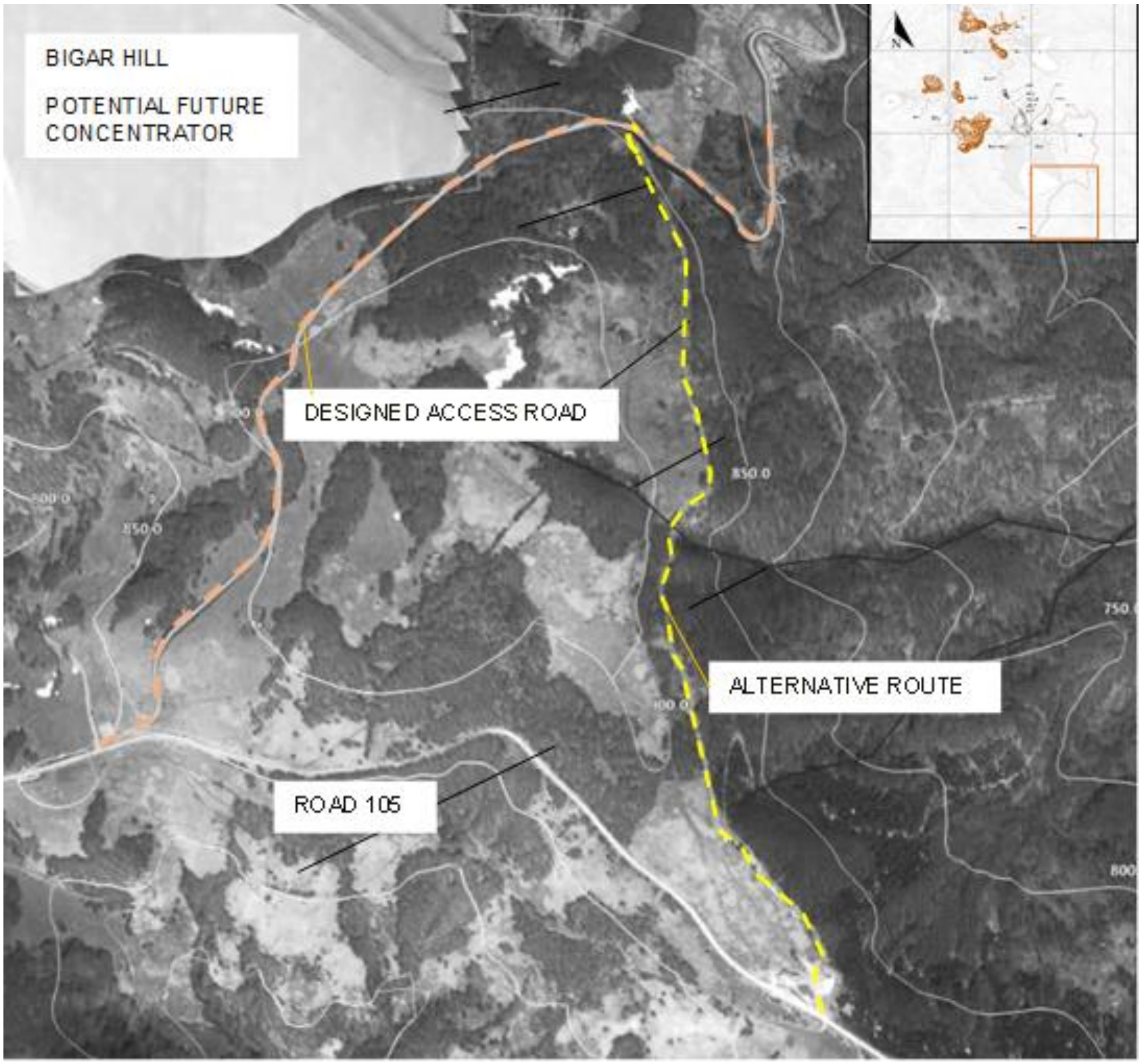
18.8.2 Internal Site Roads

The Project will have internal roads to connect multiple locations with independent access to the process plant and approximately 2 km to the dams. All internal roads will be constructed in compacted gravel layers with a width of 6 m and will include ditches and culverts according to drainage requirements. Refer to Figure 18.4.

18.8.3 Haul Roads

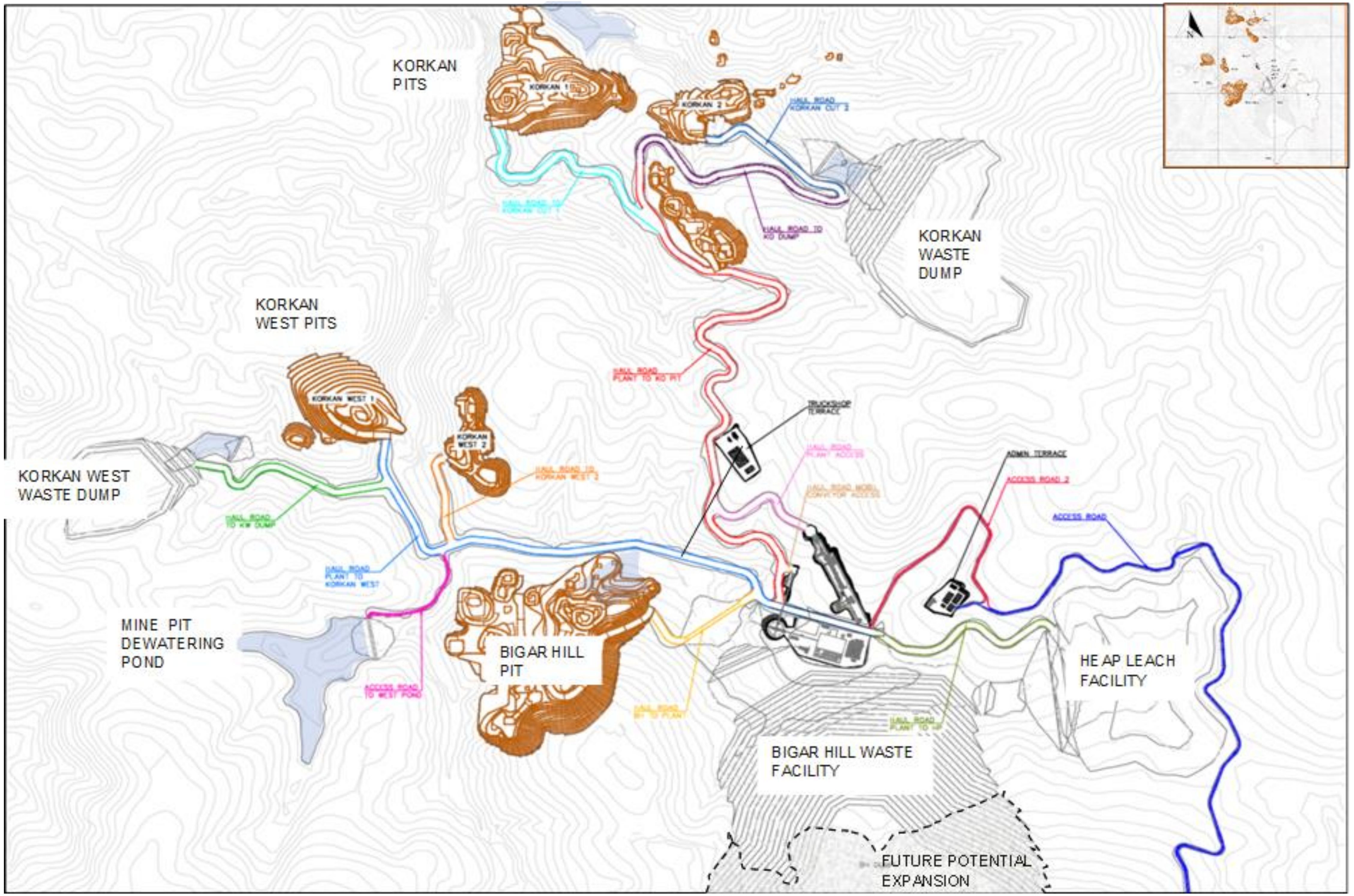
A network of gravel haulage roads, totalling approximately 9.4 km, outside the pits have been planned. The gravel haulage roads will connect the Korkan, Korkan West, and Bigar Hill pits with the respective waste dumps and process areas.

Figure 18.13 – Site Access Road - Proposed Alternative Routing



Source: DRA Global, 2020

Figure 18.14 – Haul Road Layout and Segmenting



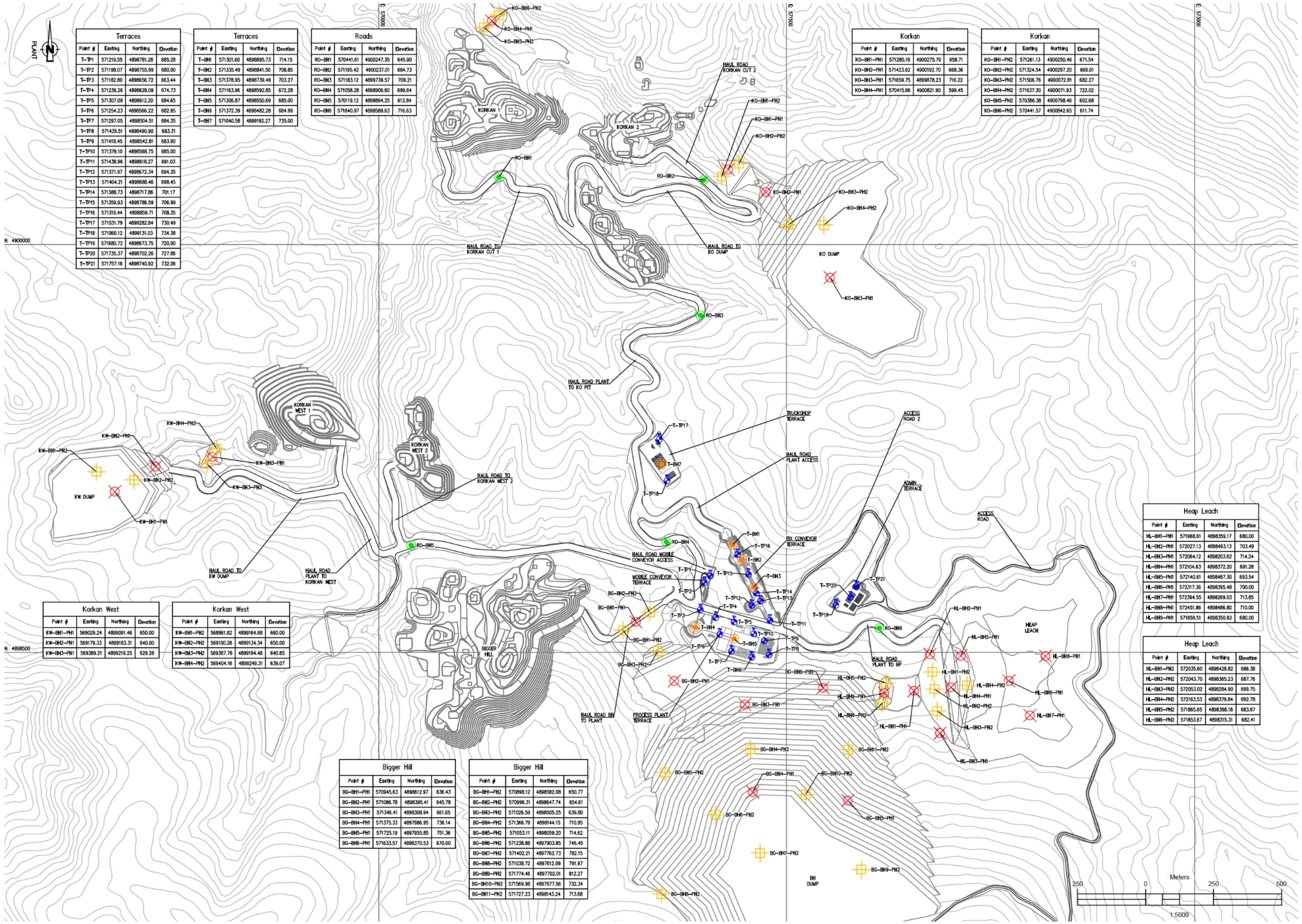
Source: DRA 2020

18.9 Geotechnical Investigations

18.9.1 Geotechnical Drilling

The PFS progressed the geographical placement of various project element based on limited geotechnical data of the founding ground conditions. Although local lithologies were taken into account, specific test pits and boreholes are required to confirm local ground conditions associated with each project element. A borehole and test pit program were compiled to provide an overview of the geotechnical requirements.

Figure 18.15 – Drilling and Test Pit Overview



Source: DRA 2020

18.10 Waste Acid Rock Drainage (ARD) Potential

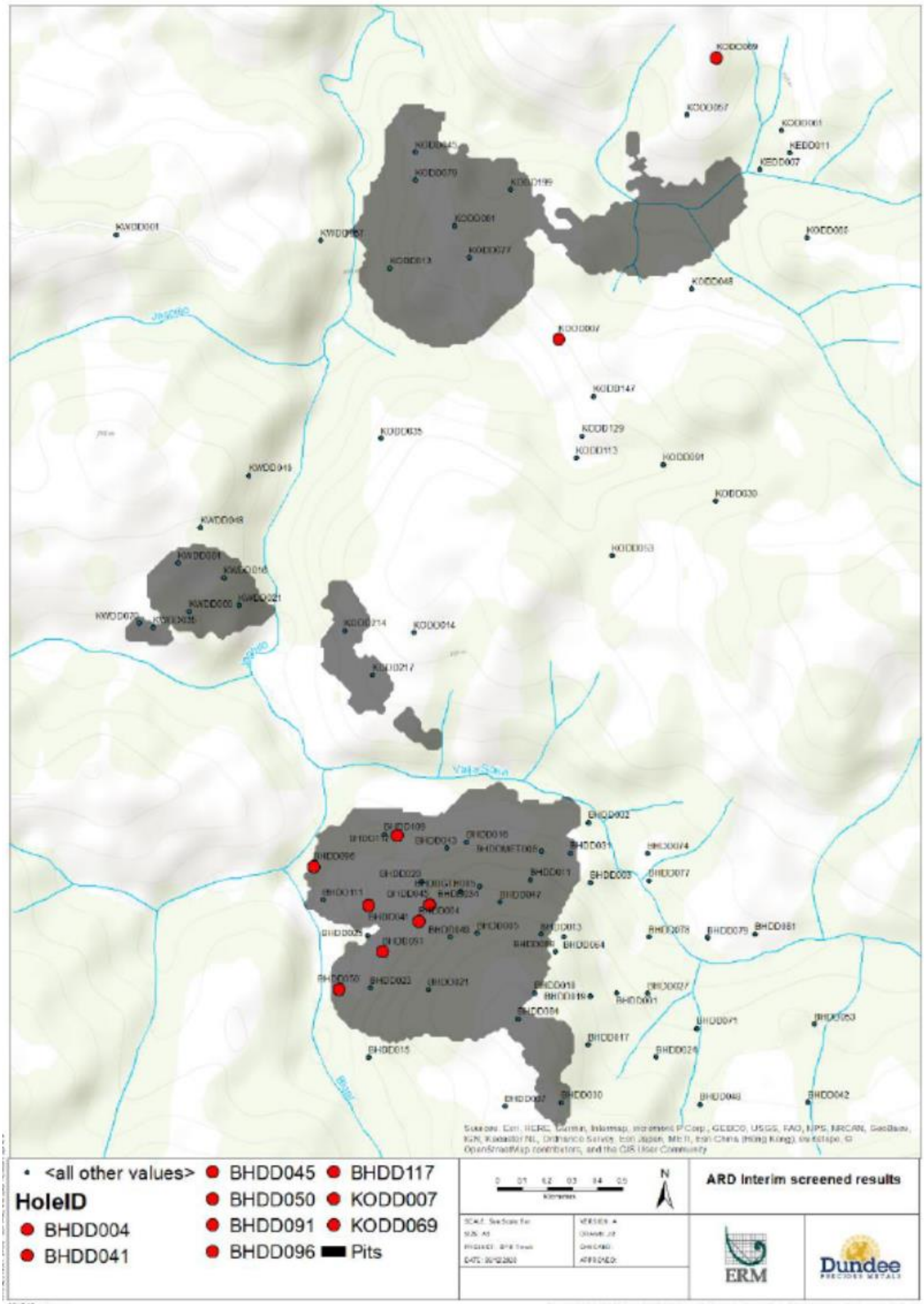
The potential for waste rock to generate acid drainage water was initially analyzed during the PEA (Avala, 2014). A low potential for ARD was indicated and subsequently no water treatment or containment measures were incorporated.

During the PFS, further ARD testwork was undertaken on drill core samples to further the ARD understanding and determine the need for water treatment and containment. An interim report has been compiled that indicates some ARD potential for less than 10% of the tested samples.

The locations of the ARD potential samples are mapped in Figure 18.16, indicating ARD prevalence in the Bigar Hill and Korkan waste areas. Treatment and containment recommendations for the ARD potential waste rock is still to be determined. For the PFS phase, an allowance has been included for the partial lining of the Bigar Hill facility, able to contain 10% of the total waste equivalent to approximately 4.8 Mt.

Additional to the facility location geotechnical studies, the HLF further requires the characterisation of the ore to support detailed heap leach design during the next phase of the project. Characterisation detail will be confirmed by the designer during the next phase.

Figure 18.16 – Potential ARD Sample Geographical Distribution



Source: ERM ARD Interim Memo

18.11 Power Supply and Distribution

18.11.1 Power Distribution

Electrical power for the Project will be provided by the power utility, Elektromreza Srbije, from transmission lines connected to the national power grid. Two (2) transmission lines are located approximately 5 km south of the Project. One (1) transmission line, referred to as 122ab supplies 110 kV and the other transmission line referred to as 122A/5 supplies 35 kV.

Power system studies using ETAP software were conducted to evaluate capacities of both transmission line voltages to supply the project. ETAP simulations were undertaken to determined power draw limits without exceeding 5% voltage regulation on the Project power distribution system. As the mine loads are supplied by the longest overhead lines, voltage regulation limits were applied to supply of the remote mine loads.

The power studies determined the existing 35 kV transmission line 122A/5 does not have sufficient capacity to supply the Project, without exceeding voltage regulation limits. The power studies did indicate the existing 110 kV transmission line 122ab may be sufficient to supply the Project, provided the existing power grid has spare capacity to supply the Project.

Load flow reports listed above indicated the 110 kV transmission line 122ab is a possible source of power for the Project provided the existing power grid has sufficient spare capacity.

For the purposes of this study, it is assumed that electrical power for the Project will be provided by Elektromreza Srbije, via existing 110 kV transmission line 122ab connected to the Project's main substation Elektromreza Srbije will need to be engaged in the during the next study period to ensure the Project's impact on the existing 110 kV power grid is recorded and accepted.

The power systems study listed above provides detailed results of load flow, short circuit and motor starting calculations for the Project power supply system connected to the existing 110 kV transmission line 122ab.

The main substation comprises a 110 kV main incoming breaker, isolation and grounding switches, metering, protection, controls, structures, grounding, and fencing.

A 20/22/30 MVA, 110–10.5 kV power transformer will be installed in the main substation. The transformer sizing accommodates the forecast total load for the Project including initial and future loads listed in Table 18.3 to Table 18.4.

A prefabricated, weatherproof, and portable switchgear house will be installed within the main substation. The switchgear house will be supplied with main 10 kV switchgear and associated battery bank and battery charger, fully serviced with fire alarm panel, lighting, and air conditioning.

Power-factor correction capacitor banks will be installed in the main substation to compensate power factor to 95% as required by the national power utility. Power factor will be controlled by a multiple stage compensator. Power-factor correction equipment will be supplied complete within a weatherproof and portable enclosure.

Power to mining areas and the related infrastructure will be distributed at 10 kV via overhead pole lines connected in 10 kV Loop Primary System with 400 V Radial Secondary System. Other underground cable line loop will distribute power to the crushing and gold area loads. A separate 10 kV feeder will supply administration buildings.

Pole-mounted transformers will step down 10 kV voltage for small service loads or buildings. Power for process and large infrastructure loads will be stepped down from 10 kV to 400 V by pad mounted transformers located adjacent electrical rooms (E-Rooms).

E-Rooms will be installed adjacent to process areas. These will be supplied complete with medium voltage switchgear, low voltage MCC's, VFD's, DCS panels and COM panels, fully serviced with fire-alarm panel, lighting, and air conditioning. The fire-alarm panel will report a fire trouble or alarm to the main control room via the communications network. E-Rooms will be supplied with handheld fire extinguishers. E-Rooms will be installed on reinforced concrete footings and pedestals, elevated 2 meters above grade for bottom-entry cables.

Power transformers are located adjacent to the electrical rooms, separated by concrete block walls with 2-hour fire ratings. Oil filled transformers will be installed on reinforced concrete oil containment pads. Dry type transformers will be installed indoors within ventilated transformer vaults.

Due to the potential for freezing liquids in pipes or tanks, piping and tanks which carry or hold liquids subject to freezing will be electrically heat traced.

Table 18.1 provides a summary of estimated loads for infrastructure based on previous similar projects.

Table 18.1 – Summary of infrastructure Loads

Infrastructure and Services	kW
Local Services, Lighting, HVAC	283
Admin Offices and Change House Transformer	340
Plant Maintenance and Warehouse Transformer	153
Truck Shop Transformer	153
Fuel and Storage	34
Sub-Total	963
Contingency 20%	*193*
Total Estimated	**1,156**

It is assumed to allow **1,200 kW** for infrastructure at this stage of the Project.

The total estimated load including an allowance for a potential future Concentrator is presented in Table 18.2. Separate tables are presented for ease of understanding and as inputs to load flow calculations.

Table 18.2 – Estimated Load with Concentrator

Area	Demand		
	kW	kVAr	kVA
Heap Leach Process	3,463	2,467	5,207
Infrastructure	1,200	900	1,500
Mining	1,046	784	1,307
Concentrator	10,545	7,909	13,181
Contingency 20%	3,251	2,438	4,063
Total	**19,504**	**14,498**	**25,259**

The total estimated load without Concentrator is presented in Table 18.3.

Table 18.3 – Estimated Load without Concentrator

Area	Demand		
	kW	kVAr	kVA
Heap Leach Process	3,463	2,467	5,207
Infrastructure	1,200	900	1,500
Mining	1,046	784	1,307
Concentrator			
Contingency 20%	1,142	856	1,427
Total	**6,850**	**5,007**	**9,442**

The total estimated load excluding Concentrator and contingency is presented in Table 18.4.

Table 18.4 – Estimated Load without Concentrator and Contingency

Area	Demand		
	kW	kVAr	kVA
Heap Leach Process	3,463	2,467	5,207
Infrastructure	1,200	900	1,500
Mining	1,046	784	1,307
Concentrator			
Contingency 20%			
Total	**5,709**	**4,151**	**8,014**

A power system study was conducted using ETAP software to simulate loads listed in Table 18.3.

18.11.2 Emergency Power Supply

Emergency backup power generators will be installed for emergency power supply to essential loads upon loss of utility power. The generator output voltage will be 400V. Average power demand for emergency power is estimated at 170 kVA with largest current motor DOL start at 45kW.

Emergency generators will consist of diesel engine-driven generators. The generators are supplied fully assembled within a weatherproof acoustic enclosure complete with main breaker and 24-hour fuel storage in the steel base.

The emergency generators will be connected to a low voltage standby switchgear assembly supplied from the 10 kV site distribution system via a step-up transformer. Under normal operating conditions the standby power switchboard will be energized from the main 10 kV supply with generators disconnected.

Upon loss of normal power, the main incoming 10 kV breaker will be disconnected from the main substation switchgear and an automatic transfer switch will start the emergency generator. The transfer switch will switch power supply from normal to emergency power once the generator reaches full voltage and the 10 kV bus will be energized from the generator. The PCS will automatically start only loads which require emergency power.

Uninterruptable power supplies (UPS) will provide back-up power to critical control systems. Fire alarm panels and emergency lighting will be supplied with integral rechargeable battery units backed up by normal power.

18.12 Automation

The plant control system hierarchy will be designed in accordance with the concept known as Computer Integrated Manufacturing and will be designed as a multi-level integrated structure.

This multi-level structure contains the following three (3) levels which concur with the levels described as the International Standards Organization (ISO) Open Systems Interconnection, model for degree of integration.

- Level 0 - Field Equipment and Instruments;
- Level 1 - Process Control and Measurements;
- Level 2 - Process Control and Supervision.

The PCS includes Level 1 equipment for process control and measurement. The main PCS will be distributed control system (DCS). The control system network topology for the PCS shall be ring.

The typical architecture for the Level 2 process control and supervision shall include:

- Two (2) PCS system operating station servers;
- One (1) Business network server;
- One (1) PCS system historian;
- One (1) engineering station;
- Two (2) control room human-machine interface (HMI) viewer;
- Field HMI viewer and license for management as per Project-specific needs.

A business network will be provided as a separate network connected via firewall to the industrial network.

All low-voltage MCC starters will be equipped with intelligent motor protection relays with built-in Ethernet communication. Starters will communicate with the control system to transfer information and control data.

All VFDs and soft starts shall be equipped with an Ethernet communication link.

All medium-voltage motor protection relays are supplied with the communication option. For the medium voltage motors, the start/stop commands are hard-wired to the PCS Input/Outputio cards.

UPS will be used to protect specific equipment from unexpected power outages. The plant network active devices will also be supplied from UPS. The UPS units will be located mainly in the electrical rooms, control room, and in critical HMI viewer panels.

18.13 Communication

On-site communications for process plant and mine area personnel will be by hand-held radios and cell phones, using the local wireless cell phone service provider.

In addition, internet and data will be accessed through the local wireless provider. A local wireless provider connection will be required early in the construction stage, providing internet and VPN network connectivity required for construction works. In addition, as soon as the dedicated Project hardwired connection is commissioned, the local wireless network will act as a backup internet connection providing network redundancy.

18.14 Waste Facilities

18.14.1 Mine Waste Management

As part of the technical report and PFS trade-off process, the addition of a potential future sulphide concentrator was explored along with associated waste disposal facilities. The Bigar Hill waste facility was identified as a potential site for co-disposal of tailings should the future concentrator be built. This section will address the Bigar Hill waste facility as a co-disposal waste facility that can potentially include tailings.

The Bigar Hill facility design is based on a conventional dump, with the flexibility to convert to co-disposal in the future should a sulphide processing facility be considered.

18.14.1.1 Sulphide Tailings Management (Future Potential Sulphide Concentrator)

The Sulphide Concentrator Trade-Off Study (ToS) evaluated several tailings management alternatives, including an independent slurry tailings management facility, tailings storage in the HLF, co-disposal with waste rock, in-pit tailings disposal and in-pit co-disposal with waste rock.

The ToS concluded that co-disposal of filtered tailings with waste rock within the Bigar Hill waste rock dump, or in-pit, was the most economically and environmentally attractive option. In the PFS, the Bigar Hill waste rock facility was designed for the heap leach only option, but with the flexibility to allow for potential future conversion to a co-disposal facility. Capex allowances reflect this assumption.

18.14.1.2 Waste Rock Management Location Alternatives

The location of the waste rock dumps in the PEA (CSA Global, 2019) were re-examined for the PFS.

A revised location at the head of the Korkan West valley was selected to manage waste rock from the Korkan West pits because it represents an opportunity to reduce the waste rock dump capital cost and footprint, simplify water management, and overlies the known limestone formations to a lesser extent than the location proposed in the PEA.

A revised location southeast of the Korkan East Pit was selected to manage the waste rock from the Korkan pits because it represents an opportunity to reduce the waste rock dump capital cost and footprint, simplify water management, and is outside of known limestone formations.

18.14.2 Waste Rock Management Facilities

The mine waste facilities, illustrated on Figure 18.1, were designed to provide the following storage capacities:

- Bigar Hill: 90 Mt of waste rock plus 9.9 Mt of filtered tailings as a co-disposal facility (future sulphide concentrator option only);
- Korkan waste rock dump: 16 Mt of waste rock; and
- Korkan West 6 Mt of waste rock.

The designed facilities use an assumed in-place waste rock density of 2.0 t/m³ and an in-place tailings density of 1.5 t/m³ based on a specific gravity of 2.65 and an assumed compacted void ratio of 0.75.

To mitigate potential impacts from potential ARD/ML, the design includes a liner for the full mine waste facility footprints, as well as their associated contact water ditches and ponds. The liner system includes a bedding layer, a 2.0 mm Double-Sided Textured (DST) Linear Low-Density Polyethylene (LLDPE) geomembrane, and an overliner protective layer and drainage blanket. The ponds and ditches are lined with a 1.5 mm Single-Sided Textured (SST) High Density Polyethylene (HDPE) geomembrane without an overliner. Liner underdrains will be provided to relieve pore-pressures to prevent liner uplift.

The mine waste facilities were designed to meet a static and pseudo-static FOS of 1.5 and 1.1, respectively and use 2.5H:1V overall slopes to simplify closure.

18.14.2.1 Possible Tailings Management (Future Potential Sulphide Concentrator)

Per Section 18.14.1.1, co-disposal of concentrator tailings was identified as the preferred option from the tailings alternatives ToS. Tailings produced by the possible future sulphide concentrator would be managed within the Bigar Hill Dump. The compacted tailings were assumed to be non-liquefiable. For the Bigar Hill Co-disposal Facility, the critical surface from the stability model was estimated and used to define the following two zones:

- Zone A (Structural Shell) fully encompass the critical slip surface and contains only waste rock to serve as a buttress and improve stability.
- Zone B (Interior Fill) is beyond (deeper than) of the critical slip surface, consisting of either filtered tailings or waste rock.

Based on the estimated failure surfaces within co-disposal facility, a minimum Zone A width of 250 m from the downstream face is considered stable.

Three (3) co-disposal methods were considered for the sulphide concentrator option and a waffle-like tailings cell structure was selected as the preferred method to provide the most flexibility and structural support within a lined facility. Tailings will be placed within the co-disposal facility in cells, with each cell contained by a waste rock perimeter to support truck traffic for tailings placement. After each tailings cell is filled, it will be graded to shed surface water runoff and covered with a minimum of 1 m of waste rock to inhibit dust generation and erosion by precipitation runoff. The waste rock cover and cell perimeter roads will promote drainage throughout the co-disposal area and improve overall stability. Multiple active tailings cells can be open at a time to allow for operating flexibility to deal with mine schedule variances, weather, and upset tailings conditions.

18.14.2.2 Uncertainties and Future Studies

The key mine waste management risks include limited data available for the mine waste facility design, including such information as foundation and groundwater conditions in the soil and bedrock, the possible presence of karst formations, borrow material characterization and availability, tailings properties (sulphide concentrator option), and mine waste geochemical characterization.

As assumptions were made in support of PFS design based on engineering judgement, the design of the mine waste facilities will need to be reviewed and updated to reflect the site-specific information. These potential changes can impact the design, schedule (both engineering and construction), and construction costs for each of the facilities.

Once the mine scheduling has been finalised, the site hydrogeology characterized, and the mine waste geochemistry completed, the requirement for liners will be re-evaluated and current exclusion of the alternative for backfilling the pits with waste rock and / or tailings to reduce the Project footprint should be reconsidered.

18.15 Water Management

18.15.1 Description of the Water Management System

The water within the basins where the mine facilities will be located, is classified into two categories, "contact" and "non-contact" water. Contact water is surface water that has been exposed to excavated materials (e.g., ore, tailings and waste rock) or mining process facilities (e.g., water within the process plant and HLF circuits). Non-contact water is surface runoff that has not been in contact with any disturbed surface within the Project area (i.e. freshwater) and is diverted around the mine facilities. Any non-contact water that mixes with contact water becomes contact water.

The water management system of the Project includes the following main components:

- Non-contact water diversion ditches;
- Contact water collection ditches;
- Contact water management ponds for the HLF, waste rock dumps (and potential tailings co-disposal); and;
- Sedimentation ponds for management of open pit dewatering.

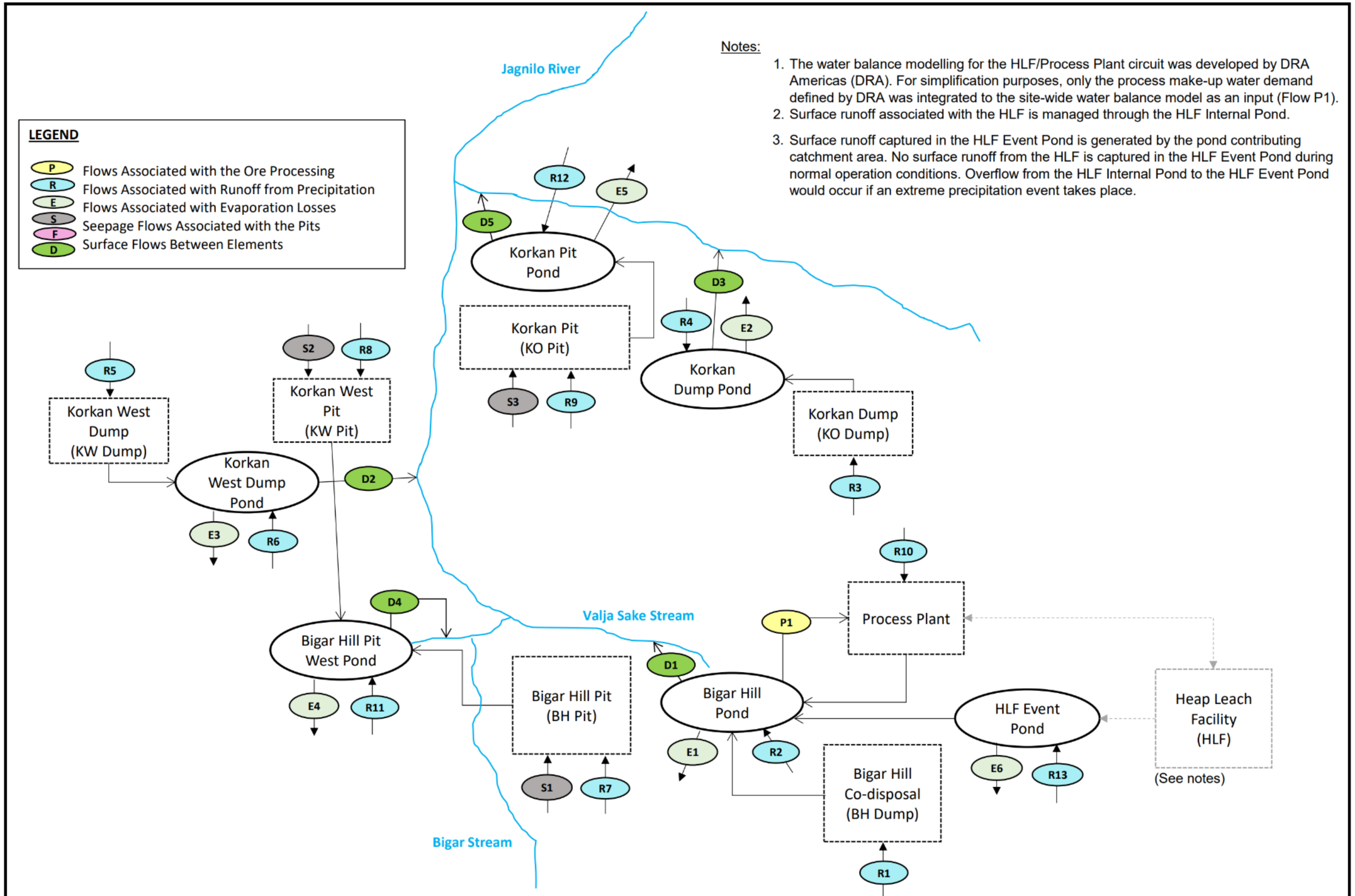
Figure 18.17 illustrates the water management concept proposed for the operating phase of the Project. The overall water management concept is to divert non-contact water to reduce the amount of contact water to be managed at the Project site and collect the contact water for conveyance to water collection ponds.

Figure 18.18 shows the potential maximum footprints of the mine facilities and the footprints of the water management infrastructure, including non-contact water diversion ditches, contact water collection ditches, collection ponds and underdrains. The ponds will be equipped with an emergency spillway to prevent dam overtopping.

The Project site water management plan includes the following:

- Two (2) ponds at the base of the two (2) waste rock dumps (Korkan and Korkan West Dump Ponds), and one (1) at the base of the co-disposal facility (Bigar Hill Pond);
- Two (2) ponds for open pit water management (Korkan Pit Pond and Bigar Hill Pit West Pond);
- An External Event Pond for the HLF;
- Contact water collection ditches;
- Non-contact water diversion ditches; and
- Underdrains for the HLF, and Bigar Hill facility (non-contact water).

Figure 18.17 – Mine Site Water Management Flow Schematic



Source: SLR Site Water Management Design Report, 2020

The water diversion ditches will collect non-contact water from surface runoff upstream of the Project facilities to reduce the amount of water to be managed on site. The diversion ditches will direct the non-contact water away from the Project facilities towards natural streams located downstream.

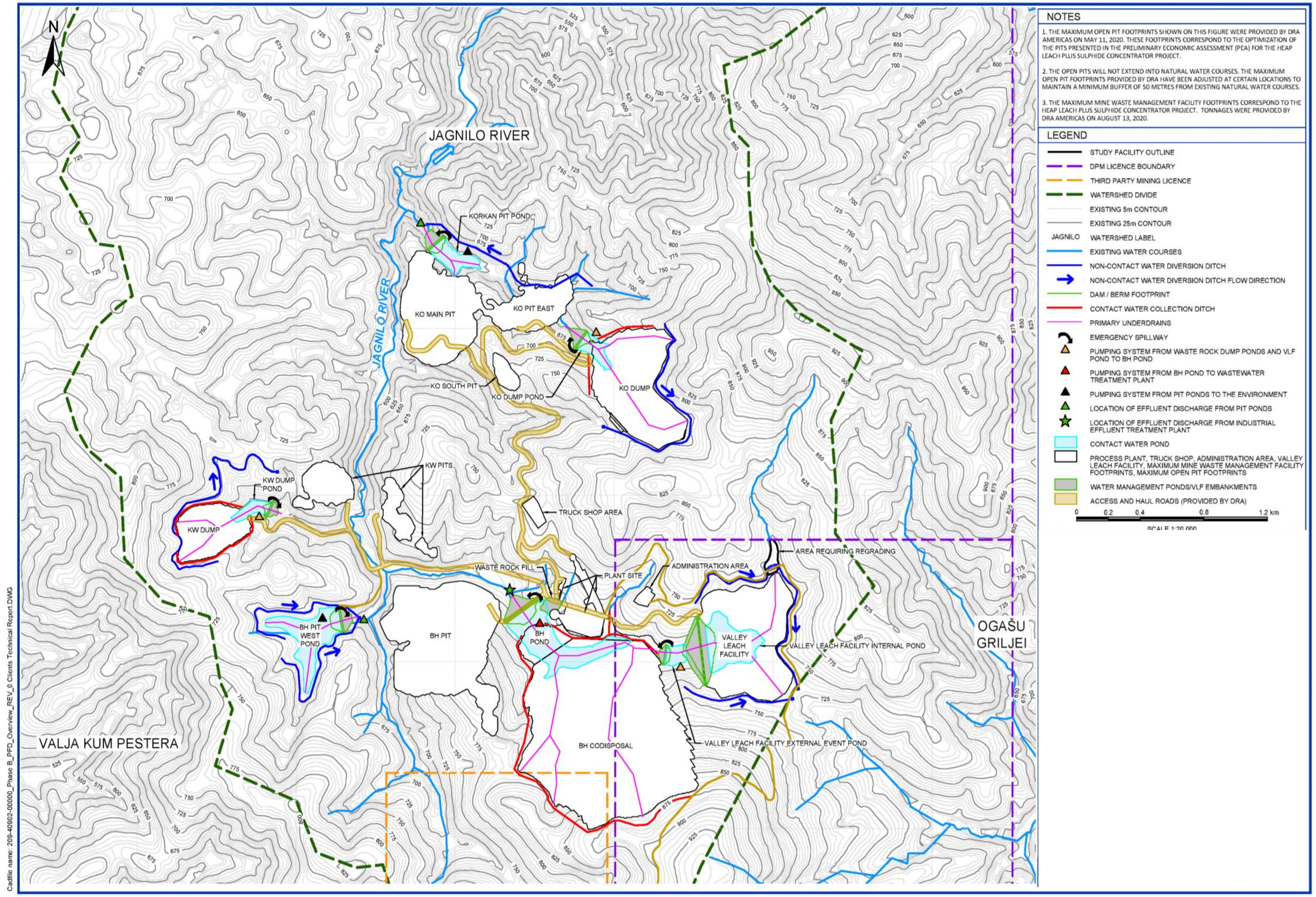
The non-contact water collected in the HLF diversion ditches will be conveyed towards the Ogasu Griljei River located in the watershed adjacent to the Jagnilo River Watershed, immediately to the east. The non-contact water collected in the rest of the diversion ditches will be conveyed towards the Jagnilo River or its tributary streams (Figure 18.18).

The water collection ditches will collect contact water from surface runoff (and toe seepage from the waste rock dumps and the co-disposal facility) to convey it towards water management ponds.

The water collected in the Korkan Dump Pond and Korkan West Dump Pond will be treated involving settling time prior to environmental discharge. The water quality is assumed to be within discharge limits. Both Korkan dump ponds provide a final monitoring point prior to discharge. In the event of unsuitable contaminants within the water, the water will be pumped to the Bigar Hill Pond. The water collected in the Bigar Hill Pond will be pumped to the Process Plant (Figure 18.17) to meet make-up water requirements for ore processing (or pumped for other water uses such as dust suppression). Excess water collected in the pond will be pumped to an industrial effluent treatment plant prior to its release to the receiving environment (to the Valja Sake Stream, a tributary of the Jagnilo River).

Surface runoff collected in the open pits together with groundwater seepage entering the pits will be pumped to two (2) sediment ponds (Figure 18.18): the Korkan Pit Pond and the Bigar Hill Pit West Pond. The purpose of the sediment ponds is to collect the pit dewatering flows and allow for settling of suspended solids prior of discharging the water to the receiving environment (to tributary streams of the Jagnilo River).

Figure 18.18 – Site Water Management General Arrangement



Source: SLR Site Water Management Design Report, 2020

18.15.1.1 Treatment of Contact Water from Waste Rock Facilities

No water quality assessment and no geochemistry characterisation of waste rock samples and tailings have been completed for the Project yet. Waste rock characterisation test work commenced during the PFS has initially indicated that 10% of the waste rock is potentially acid generating, allowing for the majority of waste rock to be placed in conventional waste rock dump facilities.

In the initial absence of ARD test work data during the initial PFS stages, the water treatment requirements were uncertain. To maintain conservatism, the PFS design allowed for ARD/ML management. Consistent with this assumption, the waste rock dumps and the potential Bigar Hill co-disposal facility were initially designed as lined facilities with geomembranes to minimise ground infiltration and protect the groundwater environment from potential contamination with contact water. This design allowed DPM to evaluate the cost associated with fully lined facilities.

Consistent with the conservative approach adopted for the PFS regarding water quality, it has been assumed that contact water collected in the water management ponds of the waste rock dumps (i.e., Korkan and Korkan West Dump Ponds) and Bigar Hill facility requires treatment prior to discharging to the receiving environment. The two Korkan ponds will provide initial settling treatment after which the water will be evaluated prior to discharge. In the event of non-compliant water results, the water will be pumped to the Bigar Hill facility where an industrial water treatment plant will be available. The implementation of an industrial effluent treatment plant is part of the Project definition at this time (refer to Section 18.15.2). Finalised geochemistry characterisation of mine waste materials and water quality studies, as inputs to the next study phase, will make it possible to assess whether discharging to the Jagnilo River would require effluent treatment to decrease concentrations below the regulated effluent standards. If treatment is required, the water quality assessment will be the basis for the key parameter determination and type of treatment required.

The proposed water management strategy includes pumping of excess water collected in the Bigar Hill Pond to an industrial effluent treatment plant, and discharge of treated effluent directly to the Valja Sake stream, a tributary of the Jagnilo River.

18.15.1.2 Treatment of Contact Water from Open Pits

Pit dewatering is considered contact water. It has been assumed at the PFS level that the water quality of mine dewatering is adequate for direct discharge to the environment and can meet regulated effluent standards if control of total suspended solids (TSS) is implemented to meet applicable regulatory limits. It is assumed that no chemical treatment of the water is required prior to discharge. Settling of solids will take place in the Korkan Pit Pond and the Bigar Hill Pit West Pond before discharging the collected water to the receiving environment (to tributary streams of the Jagnilo River). These ponds will function as sediment ponds to promote removal of sediments. If additional water treatment was required, it does not necessarily involve the pumping of water from the mine water ponds to an industrial treatment plant. There are options for in-pond treatment (e.g. batch treatment, addition of lime to control pH, etc.) that could be implemented depending on the specific requirements. Water treatment requirements will have to be verified through completion of a water quality assessment in the next phase of the Project studies.

Water quality monitoring will be implemented at the effluent discharge locations to verify compliance with applicable regulatory standards.

18.15.2 Process Water Treatment Prior to Contact Water Discharge

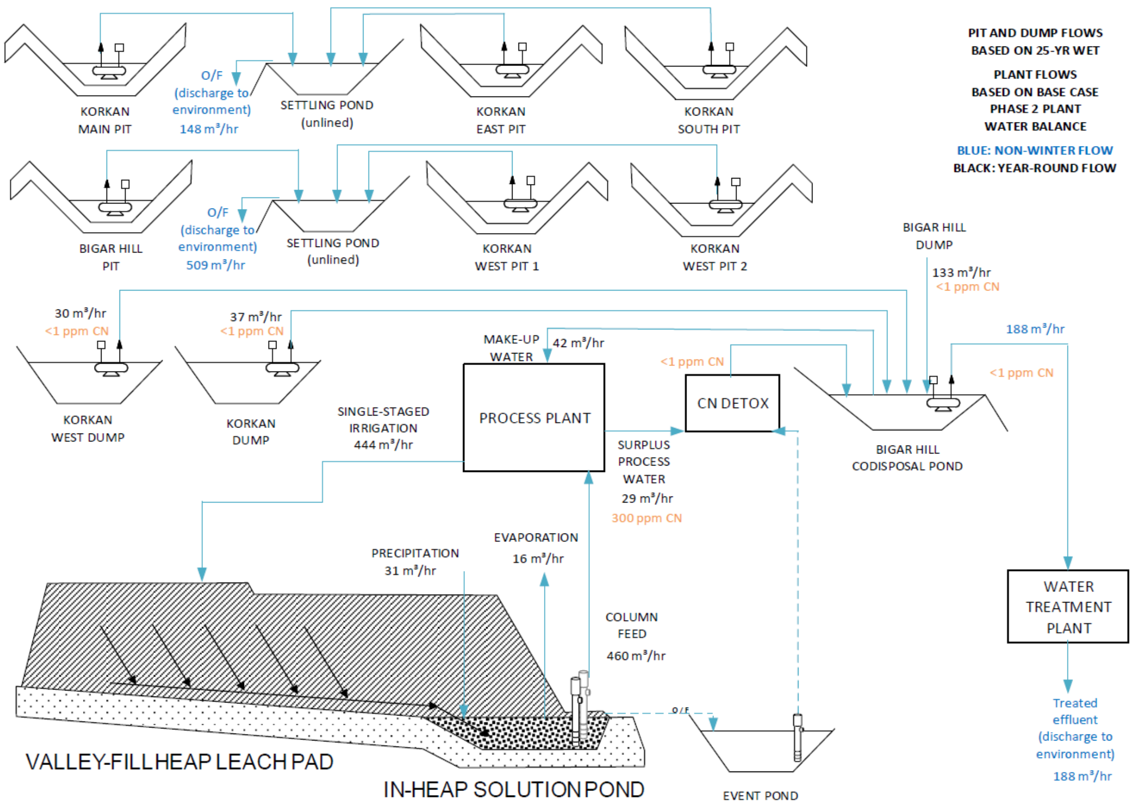
The effluent treatment plant is fed from the Bigar Hill Pond, which collects water from the Bigar Hill facility and the discharge from the cyanide detoxification plant treated water. The feed and effluent would be sampled and analysed each shift. It includes flocculant and lime preparation systems, tanks for coagulation and flocculation, a clarifier, and an effluent discharge tank, for purpose of meeting effluent guidelines for metals like copper (< 0.3 ppm).

The PFS water treatment requirements and treatment plant design were defined by DRA. The proposed treatment pathway includes a CN detoxification and an effluent treatment sector. The CN detoxification pathway makes use of Caro's acid, a mixture of sulphuric acid and concentrated hydrogen peroxide (50% strength), as an oxidizing detoxification agent, for treatment of the concentrated surplus of process water (300 ppm CN). The effluent treatment pathway makes use of coagulation, flocculation, and decantation steps for metals treatment and pH correction. Utilising the site water balance and a design factor of 20%, the effluent water treatment capacity has been determined as 227 m^3/hr, for treatment during the non-winter months.

The CN detoxification plant includes peroxide and sulphuric acid storage vessels, a static mixer and metering unit for formation of Caro's acid, and detoxification tanks where the water is treated with Caro's acid. Caro's acid is formed instantaneously within the static mixer, whereas treatment requires up to one hour within the detoxification tanks. Refer to Figure 18.20.

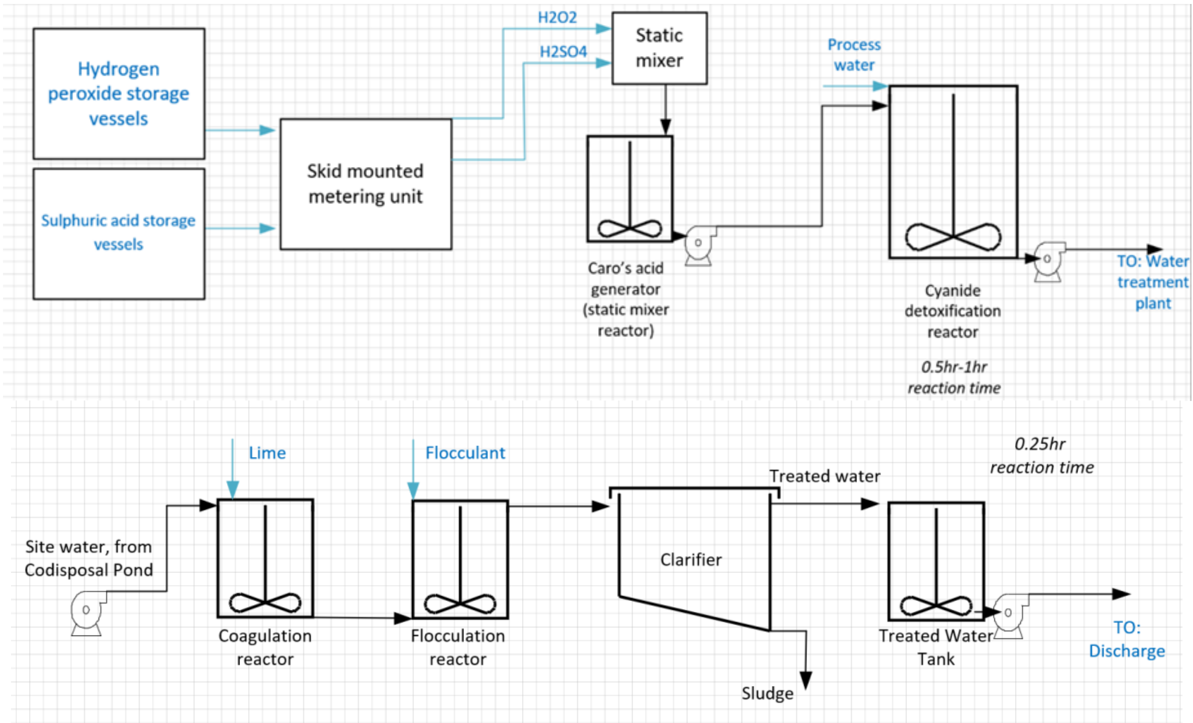
The CN detoxification plant is fed from the process plant overflow and HLF event pond. Refer to Figure 18.19. The CN detoxification plant is designed by DRA to treat water containing 300 ppm CN down to the target effluent quality of < 1 ppm CN (0.5 ppm CN WAD) in accordance with IFC EHS Guidelines for Mining. The feed and treated discharge would be sampled and analysed each shift. The plant will discharge into the Bigar Hill waste dump facility where the pond area is potentially lined for waste acid generation (ARD) purposes, as an additional safety measure. Detoxifying the process water before mixing with waste dump contact water avoids any surface waters containing > 50 ppm CN, as required by the ICMC.

Figure 18.19 – Water Treatment Flow Chart



Source: DRA, 2020

Figure 18.20 – Cyanide Detoxification and Effluent Water Treatment Plants Block Flows



Source: DRA, 2020

18.15.3 SITE-WIDE WATER BALANCE

Deterministic water balance modelling developed in Excel spreadsheets was carried out for the operating stage to:

- Estimate the available surface runoff that could be collected from the Project facility footprints taking into account the implementation of water diversion ditches;
- Verify that the process make-up water demands can be met; and
- Simulate the transfer of water between water management ponds to either store, use the water collected on site to support mining activities (process, dust suppression, etc.) or discharge excess water to the receiving environment.

Only ultimate facility footprints were considered for the PFS site-wide water balance. Snapshots for pit development sequencing were not considered in the PFS water balance simulations.

The site-wide water balance simulations were developed for average annual precipitation conditions as well as dry and wet annual precipitation conditions considering three (3) return periods: 10 years, 25 years and 100 years. The modelling results show that even during dry years (i.e., annual precipitation below average), there is a water surplus that must be discharged to the receiving environment. The analysis indicates that surface runoff collection can meet the water demand for the

Project without the need to withdraw water from natural streams or aquifers through groundwater wells, even under dry annual precipitation conditions.

18.15.4 Design Criteria for Water Management Facilities

The key design criteria of the water management components are as follows:

- The diversion ditches required to divert non-contact water are designed for closure for the 1:100 year, 24-hour rainfall event.
- The perimeter collection ditches required to convey contact water to water management ponds are designed for closure for the 1:100 year, 24-hour rainfall event.
- The water management pond used as a source of water supply for ore processing is designed with storage capacity to maintain one month of make-up water supply as a contingency.
- The water management ponds are designed with storage capacity to store surface runoff volumes during the spring freshet under the 1:25 year wet annual precipitation conditions and considering a 9-month discharge period (i.e., non-winter months).
- The water management ponds are designed with storage capacity to contain the runoff volume resulting from the 1:100 year 24-hour storm event.
- The pond emergency spillways are designed to safely convey the Inflow Design Flood while maintaining a minimum freeboard to prevent dam overtopping. The Inflow Design Flood used for sizing of the emergency spillways is one third the interval between the 1:1,000 year 24-hour storm runoff event and the 24-hour Probable Maximum Flood (PMF).

19 MARKET STUDIES AND CONTRACTS

19.1 Product Definition

The process facility will produce doré bars with a gold purity typically greater than 95% when processing Timok ores. Doré bars will be shipped regularly to a commercial refiner where their value will be verified. Sale prices will be based on London Metals Exchange market pricing and conducted under standard terms.

19.2 Market Studies

Gold and silver markets are mature global markets with numerous reputable refiners located throughout the world. Both commodities are commonly traded by banks and other market dealers. As such, a market study for gold products was not undertaken.

The terms contained within any future sales contracts are expected to be typical and consistent with standard industry practice and similar to contracts for the supply of doré elsewhere in the world.

19.3 Commodity Price Projections

DPM has adopted the following gold price projections for the PFS as depicted in Table 19.1.

Table 19.1– PFS Adopted Gold Pricing

Gold Price	Application
$1,250 per oz	Pit optimisations
$1,500 per oz	Contracts and Saleable products

Source: DPM

DPM internally compares projects at $1,250 per oz and utilises $1,250 per oz to determine a conservative Mineral Resource cut-off. $1,500 per oz was adopted as the saleable gold price based upon long term predictions and market benchmarking.

19.4 Contracts

DPM has no current contracts for project development, mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts or arrangements related to this specific Project.

Contract terms and treatment charges will be negotiated with international refining companies in proximity to the mine site. It is expected that any future contract terms will be typical of similar contracts in Serbia or the Balkan region.

Table 19.2 shows the contract terms and treatment charges that were used in the economic analysis Section 22, and in support of the Mineral Reserves Estimate. The terms were obtained by DPM via market engagement.

Table 19.2 – Proposed Contract Terms and Treatment Charges

Category	Terms
Gold Refining Charge	$0.5/troy ounce fine gold credited (average)
Gold return	100% of assayed content
Shipping Cost	$0.70/troy ounce fine gold credited (average)
Handling Charge	$500 per 100kg lot shipment from site
Initial Settlement	10 days following shipment from site

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The Project has engendered permitting efforts, environmental studies and community engagement since 2007. Initial environmental and social baseline work commenced in 2012 and continues to the present day. This section summarises the work undertaken for social and environmental studies, permitting and community engagement and highlights key risks and mitigations for these different aspects that will be more completely covered in the Project Environmental and Social Impact Assessment (ESIA) to be started in 2021, for which several baseline studies are already well advanced. This section addresses the current Project footprint and excludes Kraku Pester.

The Project is located in Serbia, and as such will be permitted to operate and regulated by Serbian authorities to Serbian standards. Mining structures are permitted under the Law on Mining and Geological Explorations whilst supporting and auxiliary structures such as roads and administrative buildings are separately permitted under the Law on Planning and Construction. There is a range of other approvals and permissions required under ministries including the Ministry of Agriculture, Forestry and Water Management and Ministry of Environmental Protection. However, the permitting system is undergoing change as part of Serbia's planned accession to the EU.

Serbia was granted candidate status in March 2012 and legislation and frameworks are being harmonised progressively to those of the EU. Therefore, the Project utilises standards that relate to EU environmental laws, such as the environmental impact assessment directive (2011/92/EU), water and waste framework directive (2008/98/EC) and industrial emissions directives (2010/75/EU). Few private-sector mining projects of this scale have passed through the entire Serbian permitting process in recent years, albeit Cukaru Peki to the east is reasonably advanced through construction works; hence there are limited precedents to inform the permitting process as it currently stands.

DPM has sought to minimise permitting risks by engaging with regulators and aligning the Project with EU requirements and good international practice such as the performance requirements of the EBRD Equator Principles and World Bank Group Environmental, Health and Safety Guidelines (International Finance Corporation World Bank Group, 2007). The permitting process, and progress through it at the current PFS stage is detailed in the sub-sections below.

20.2 Project Description

The Project is located in the District of Braničevo in Eastern Serbia; a rural, hilly area with steep valleys, characterised by seasonally grazed pastures, woodlands and isolated farms and houses. The Project site is sparsely and seasonally populated, with the closest population centre being the town of Žagubica 10 km to the southwest. There are no designated protected areas for biodiversity or cultural heritage in the Project footprint. Some of the area is underlain by limestone where a karst has developed leading to caverns, springs and sinkholes. DPM has placed the key mine facilities outside known karst-affected areas to the extent possible.

The current mine plan is for three (3) pit areas to be developed with oxide and transitional material extracted and then processed using conventional heap leach technology, whilst stockpiling any sulphide material mined for possible processing in the future. Haul roads connect the main Project infrastructure, and due to the steep topography, require a wide belt of cut and fill. Overhead electricity cables and pipelines for water management will run alongside some of the haul road routes. Figure 20.1 depicts the layout of the project infrastructure and the watercourses in the area.

Three (3) open-cast mining areas, of different sizes, are located on valley sides and hilltops and will be excavated in parallel, requiring three waste rock dumps and six (6) water management ponds with earthen dams. The main plant area will be located in the Valja Saka valley, east of the Bigar Hill pit, with a valley heap leach facility located upstream of that. A mine workshop and truck maintenance area are located north-west of the main plant, with an administration area with offices to the east. The main plant area will comprise a heap leach plant, run-of-mill and stockpile areas and a crushing and conveying line to an agglomeration plant and loading area. DPM will use heap-leach technology to extract gold from coarse-crushed oxide mineralised material using a cyanide-based lixiviant applied to the surface of the heap leach pile. Pregnant solution will be collected by a robust lined-drainage system to the heap leach plant area where gold will be chemically removed in activated carbon tanks, and the solution recharged back onto the heap leach. Excess process water will be generated in relatively small volumes, which will be treated in a CN detoxification plant before discharge to the process water pond.

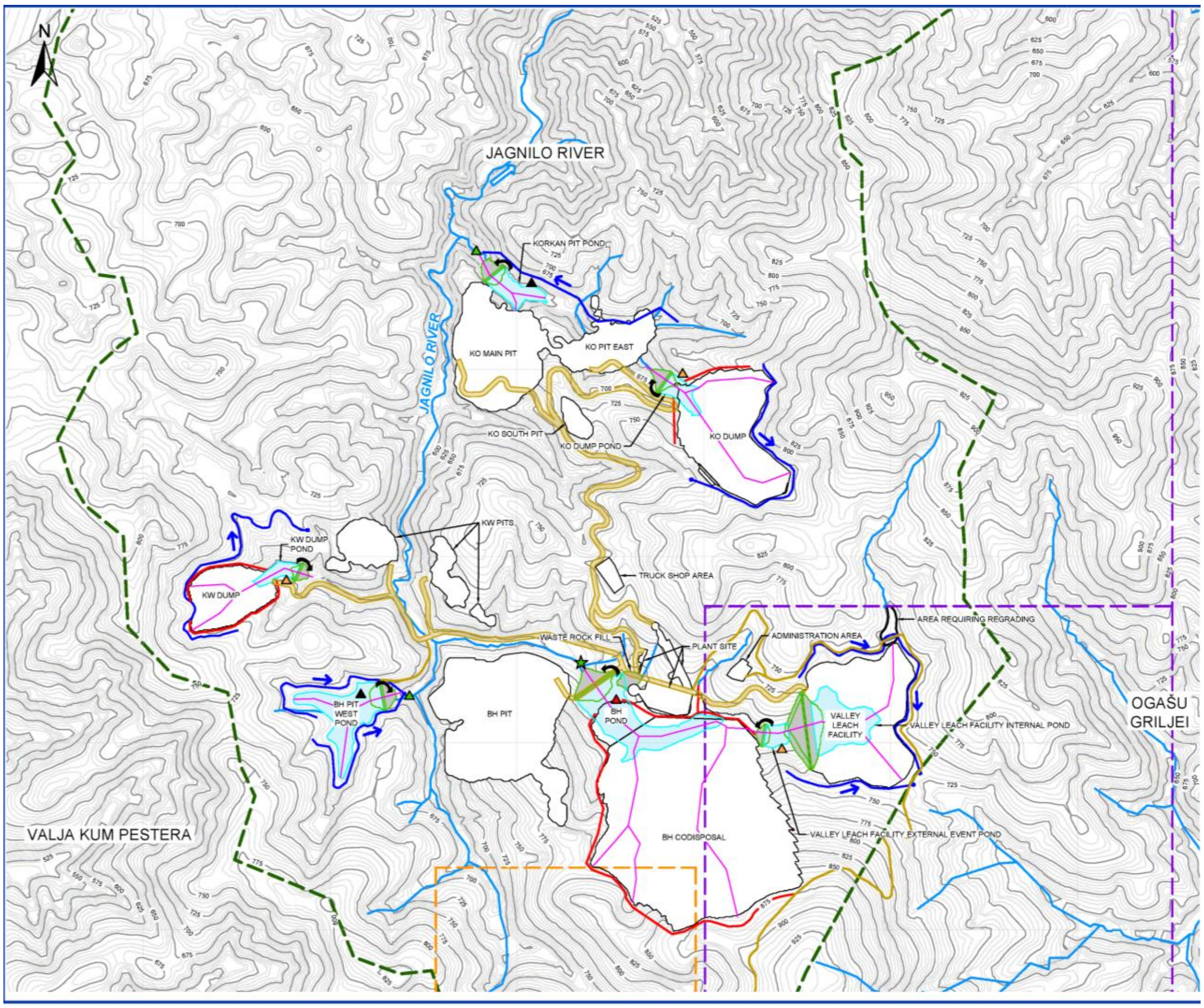
Alternative project designs considered include a processing plant and waste facilities for the deeper sulphide ore material. The conclusion was that a sulphide concentrator was not economic at current gold prices and gold recoveries indicated by test work, and therefore this phase of development is not included in this report. Future plans are to study, as a future opportunity, processing facilities for the refractory sulphide ores from the different pits, once a suitable process route has been developed. Any future development of the sulphide resource will be subject to separate pre-feasibility and feasibility studies.

Overall, the DPM approach presents several advantages compared to other approaches:

- Relatively low footprint, comprising the phased development of pits and waste rock dumps and a small plant and heap leach area; all of which will be rehabilitated at the end of the mine life.
- Low energy demand.
- Low water discharge.
- Whilst the transportation and use of cyanide in the heap leach process presents potential risks to surface and groundwater quality, DPM is a signatory to the International Cyanide Management Code (International Cyanide Management Code, n.d.) , which provides standards

of practice for protection of communities and the environment during transportation of cyanide and specific usage requirements on handling, storage, operation, disposal and decommissioning. These standards and guidance have been used in the development of the design options at this PFS stage. For example, the leach pad has been sited away from the karstic limestone zone, where infiltration to groundwater could more readily occur, and leach pad design includes a robust liner system that collects the gold solution and also prevents infiltration to groundwater.

Figure 20.1 – Timok Gold Project Layout and Water Management Structures



Source: SLR, 2020
Note: Dark blue lines are new constructed diversion channels for non-contact water. Red lines are new constructed channels to convey contact water. Thin magenta lines are sub-drains to convey contact water.

20.3 Water Management

A Mine Water Management Plan (Section 18.15) has been developed for the PFS design outlined above, which defines the hydrological parameters to support engineering design, estimates the open pit dewatering rates, and defines the site water management plan, water management closure concepts, site-wide water balance modelling, sizing of water management ponds and ditches, pump sizing, and construction material estimates. Contact and non-contact waters will be managed separately, with diversions around most of the infrastructure directed to the Jagnilo River. The diversions around the HLF area will be diverted to the adjacent Ogasu Griljei catchment to the east (Dumitrov River), see Figure 20.1.

All the pits will require dewatering, which will be done using sumps advanced as the pit progresses. In the early stages of pit development there will be little groundwater inflow as the depth to groundwater can be several to tens of metres, especially on the hilltop deposits. The sumps will also remove rainwater run-off that falls within the pit, and the water will be directed to the Bigar and Korkan Pit ponds. After settlement, the sump water will be monitored and discharged from the Bigar and Korkan Pit ponds into tributaries of the Jagnilo River. These discharges will be managed carefully to maintain appropriate social and ecological water quality and flow limits.

Process water supply will be from the Bigar Hill / process pond which is primarily fed by contact water run-off from the Bigar Hill waste rock dump and from the Korkan and Korkan West Ponds (themselves fed by waste rock dump contact water run-off). Excess water in the Bigar Hill / process pond will be treated and monitored, prior to discharge to the Valja Saka, at an industrial water treatment plant to be located by the north-western end of the Pond. Process water balance calculations and diagrams have been developed (DRA, September 2020) to model water needs in baseline conditions, in unusually dry conditions and in unusually wet conditions (24 hr 1 in 100-year storm event). Spillways on the dams will prevent over-topping and will discharge to the Jagnilo River or its tributaries, although the impoundments have been designed to accommodate extreme (24hr 1 in 100-year) rainfall events.

The design will be for no discharge / seepage to groundwater during operations, with liners, drains or seals preventing infiltration installed on the Process Plant area, the Valley Heap Leach pad and all three ponds serving the waste rock dumps. Waste rock dumps will only be lined in areas designated for rock-types identified as potentially acid generating, currently anticipated to be an area or cell within the Bigar Hill Waste Rock dump, which will drain to the lined Bigar Hill / Process Pond, where water treatment is available. There is also a lined Event Pond located at the foot of the Valley Heap Leach pad dam to collect water in the event of an emergency or extreme run-off event. If this were to fill, the Event Pond spillway will direct overflow to the Bigar Hill / Process Pond. A hydrological study of watercourses and details of planned water management and treatment will be submitted as part of the application for water conditions, a key stage in permitting requirements in Serbia. (Refer to Figure 18.20 – Mine Site Water Management Flow Schematic depicts the overall site water management.)

20.4 Alternatives

A range of alternatives has been assessed as part of the PFS, primarily in two (2) Trade-off Studies (DRA/DPM, 2020) (DRA/DPM, 2020) (DPM/DRA, 2020) (DPM/DRA, 2020) as well as alternatives and design options considered in other reports, such as the Water and Waste Management Plans and Closure Plan (SLR, November 2020) (ERM, 2020). The Project has taken environmental and social considerations into account in assessing alternatives in line with EBRD Performance Requirements and the Serbian Rulebook on the Contents of the Environmental Impact Assessment Study. These considerations have been included in the following alternatives:

- **Heap Leach Plant** – a Best Available Technology study considered 13 different process and lixiviant options that included resource usage, environmental risk profile of different process chemicals (including cyanide), social factors such as nuisance odours and number of traffic movements, the relative costs and overall efficiency of the gold recovery technology. Cyanide heap leach technology was selected as optimal, although four other options were initially considered possible. A further, more detailed analysis was made of the four most-optimal process and lixiviant options considered as alternatives to cyanide. This study concluded that the Project would not be economically viable, as a result of low gold recoveries or high operating costs or a combination of both, and cyanide heap leach remained the preferred best available technology, noting that rigorous safe management of plant and procedures that is required by signatories of the International Cyanide Management Code.
- **Sulphide Concentrator Plant -** consideration of three (3) different plant designs from a breakdown of equipment and plant components required for each, and the power consumption and cost efficiencies of each. The design of several waste management facility alternatives was also considered in detail. The overall conclusion was that a sulphide concentrator was not economic at current gold prices and gold recoveries indicated by test work.
- **Mineral Waste –** previous designs and layout for waste rock dumps in the PEA have been revised and optimised for the mine production plans, including three (3) options for the Korkan West waste rock dump, where traffic movements, noise and resource efficiency were factors. The extent to which waste rock is acid-generating / metal leaching is the subject of a detailed long-term study underway. The approach undertaken in the PFS is to designate areas or cells within the Bigar Hill waste rock dump that will be lined to prevent acid or metal leachate to groundwater or run-off to surface water. Alternatives considered included allowing for liners at all waste rock dumps, it is likely that this will not be required for a high proportion of the waste rock material and would have significant cost impacts, rendering the project uneconomical. Nevertheless, all the waste rock facility designs are such that they can be either lined or unlined, allowing for integrity of the associated mine plans or spatial layouts. Final PFS design was based

on 10% Of the waste rock dump being lined. Upon completion of the test work, the extent of PAG waste rock will be better defined, allowing a more appropriate design to be implemented.

- **Heap Leach Location and Configuration -** consideration of relative financial and environmental performance of the previous PEA heap leach location with two (2) other options; a single headwater valley facility and a multiple small pads option. Environmental factors included resource usage, environmental risk and other limitations such as underlying karst geology. The single Valley Heap Leach was considered optimal.
- **Water Management** - several aspects of the water management plan will remain flexible and reactive to actual conditions as they develop during construction and operation. These include drainage and diversion design around the waste rock dumps, and discharge and treatment options from pit ponds and the process pond. For all ponds, discharge will be contingent on water quality and seasonal flow constraints.
- **Mine Closure** – although mine closure is an ongoing process that develops as the Project progresses, early-stage closure planning has already begun and several alternatives are under consideration. These range from closure vision options on post-mining land-use (commercial, tourism, agriculture, amenity), different flooding or infilling alternatives for the pits, to more detailed technical options on the design and degree of permeability for the caps emplaced over the closed heap leach facility and waste rock dumps. The PFS closure plan depicts the option to return the site to natural state, with the option of exploring further opportunities during the next project phase.

20.5 Permitting

The Project has prepared an environmental permitting strategy (ERM, 2020) setting out the principal permits and approvals required for exploration, construction, operation and closure of the project. Key permits are summarised in Figure 20.2. The permitting system is undergoing change as part of Serbia's planned accession to the EU. To address this potential risk, the Project has been aligned with EU requirements and good international practice such as the performance requirements of the EBRD, Equator Principles and World Bank Group Environmental, Health and Safety Guidelines.

The Company currently holds exploration licences for the Project which are in the final exploration licence extension period. This allows exploration activities to continue within the exploration areas of Potaj Čuka and Bigar Istok to mid-2021 and Umka to 2024.

Following expiry of the exploration licence, the DPM will apply for a Reservation of the Exploration Field Permit. This gives a maximum three (3) year period in which to complete permitting activities required to obtain authorisation for exploitation: Approval of Exploitation of Minerals. No further exploration activities are allowed during this period.

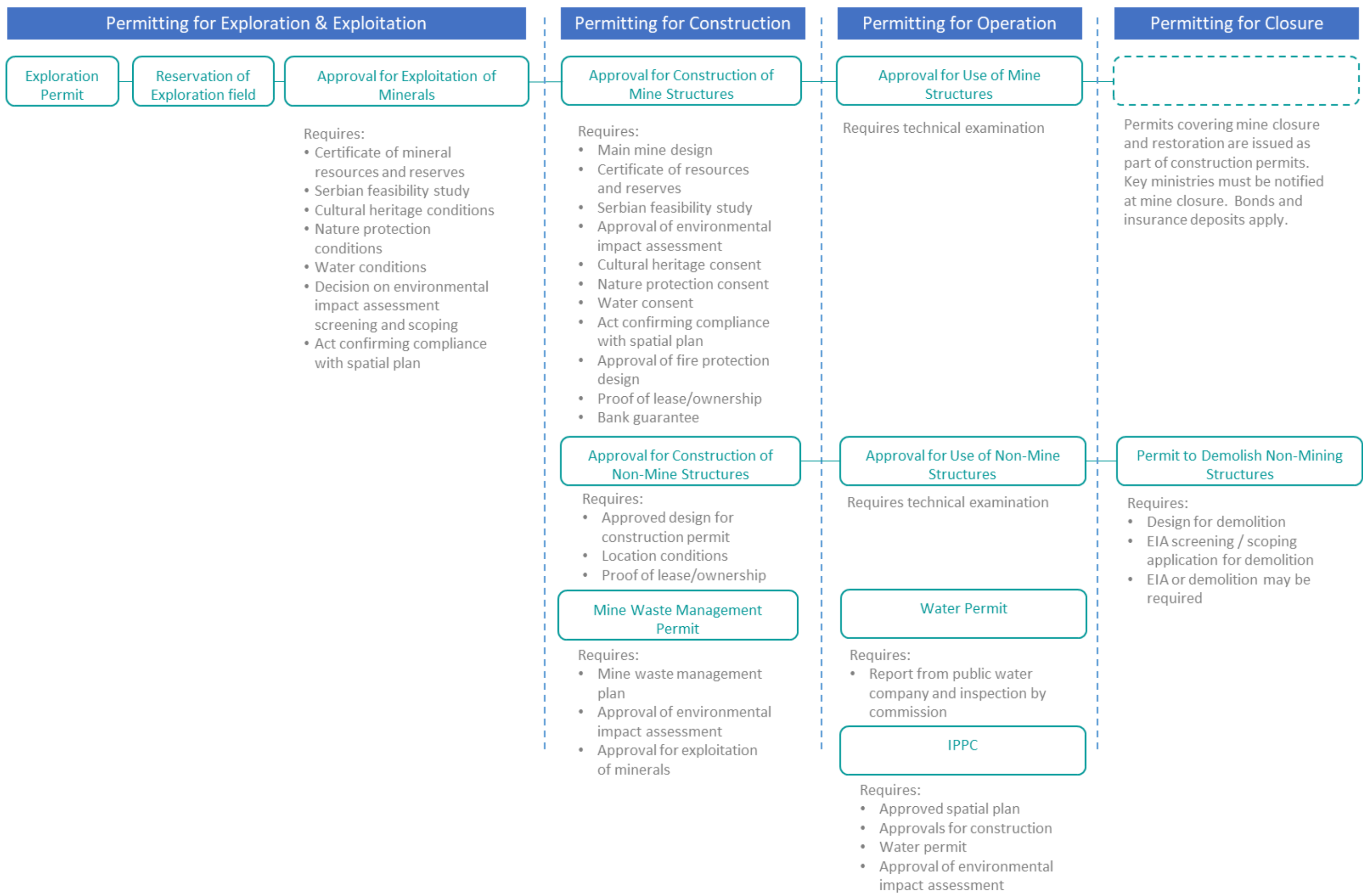
Although not part of this PFS design, the possible future expansion to extract and process sulphide material could require larger waste rock dumps and a tailings facility. These could encroach into the exploitation licence area of a neighbouring calcite ore body development immediately south of Bigar Hill, and would require a change to the existing mine licence and a potential change to the spatial plan.

Obtaining the Approval of Exploitation of Minerals is a key milestone, setting out the area and terms for exploitation. This requires various precursor permits and permissions to be obtained, including:

- Confirmation of compliance with the Spatial Plan.
- A Certificate of Mineral Resources and Reserves.
- A Serbian Feasibility Study.
- A Decision on the Environmental Impact Assessment Screening and Scoping.
- Regulator Conditions for water, cultural heritage and nature protection.

DPM has workstreams under way to prepare for these. A Spatial Plan will be required for the areas of Potaj Čuka and Bigar Istok to authorise a change in land use to mining. The process of development of a plan is lengthy, taking of the order of 18-24 months and up to 33 months in the case of a recent mining project in Serbia of similar scale. Hence this process sits on the critical path for permitting. Development of the plan is regulator-led and will be triggered and supported by DPM. DPM has requested preparation of a Spatial Plan. Design information will be disclosed during the public hearings required for this process, including information on potentially sensitive topics such as gold cyanidation and closure. DPM will work closely with the plan developer and in line with its communication strategy and stakeholder engagement plan to facilitate these processes and manage project risks. Environmental information gathered by the Project will be shared with the plan developer to help make sure that the Strategic Environmental Assessment that will accompany the spatial plan will be compatible with the Project-specific Environmental Impact Assessment.

Figure 20.2 – Principal Permits and Permissions Required



The study to support application for the Certificate of Mineral Resources and Reserves and the Serbian Feasibility Study will build on the studies developed as part of this PFS.

DPM has committed to undertake an ESIA in line with good international practice. An Environmental Impact Assessment (EIA) will also be required by the Serbian authorities. The initial phase of this assessment will involve preparation of a scoping report, to be submitted as part of the request for a decision on Environmental Impact Assessment Screening. The assessment itself will cover impact of the Project design on air, water, soils, biodiversity and soils, local people and other receptors in line with Serbian and international requirements. A range of baseline studies will be needed and given the long lead time of many of these studies, DPM has already embarked on planning and delivery of these studies. Relevant design information will be disclosed during the permitting and stakeholder engagement process.

Applications for regulator conditions on cultural heritage and nature protection are needed and will require background information and engagement with the regulators.

Water conditions will also be required. These form the first part of a three-stage process to obtain the water permit, which covers abstraction, use and discharge of water for mining projects and the process plant's operation and also the storage and discharge of hazardous and other substances that can pollute water (Law on Water, Article 122). It will define the quantity of water that can be abstracted and discharged, and the quality of the discharged water in line with the receiving water, or potential use of wastewater. Twelve (12) months of baseline data is required for this permit and planning for baseline data collection is under way.

Risks associated with potential Project delays have been identified including potential changes to regulation to align with EU law, regulator delay, public challenge to the Spatial Plan or Serbian EIA and administrative appeals. DPM continues to work closely with regulators to understand priorities, anticipate changes and to provide support as few private sector mining projects have passed through the full permitting process.

20.6 Environmental & Social Studies, Setting and Issues

The Project is located in eastern Serbia, on the mountain range between Bor to the southeast and Žagubica and Laznica to the west. Mjajdanpek and the Danube River are to the north. State roads 164 and 161 run east-west close to the Project area. There are no designated protected areas for biodiversity or cultural heritage in or around the infrastructure that make up the current Project (Korkan, Korkan West and Bigar Hill). The area is characterised by wooded valleys and seasonally grazed pastures with isolated settlements.

Key environmental and social risks are similar to those associated with other gold mining projects and include safeguarding rivers, groundwater and biodiversity and mitigating permanent effects, as well as risks associated with acquiring land. During operations, rivers will be affected by dewatering, diversions and discharges, and permanent infrastructure will overlie several kilometres of upper headwaters reach of the Jagnilo tributaries. Seasonal farming, hunting and tourist amenity will also be lost during operations, although closure planning could include the means to reinstate these after operations cease. Completion of the baseline survey program is required to fully understand and manage these impacts. The baseline program includes:

- Conducting surface and groundwater monitoring and developing groundwater models.
- Completing terrestrial and aquatic biodiversity assessments.
- Completing cultural heritage baseline data collection (caves, buried archaeology, historic farms and mills, intangible heritage, particularly for certain minority ethnic groups).
- Conducting air quality, noise and vibration monitoring.
- Determining acid-generating potential of rocks and soils to be disturbed.
- Identifying local users of water.
- Conducting a social baseline and census and socio-economic survey for landowners and those who will be economically displaced by the Project.

20.6.1 Environmental & Social Studies

Table 20.1 provides a summary of the environmental and social studies already completed for the Project and those planned in the coming 12 to 18 months. Baseline studies completed to date have provided an understanding of the key environmental and social sensitivities in the area. The schedule of future work has been planned to meet Serbian permitting requirements, align with international good practice and to help further understand or close out environmental and social risks.

Table 20.1 – Environmental and Social Studies Completed and Planned

Environment	Work Completed	Work Planned
Surface water and groundwater	Hydrological studies undertaken in 2012, 2013 and 2014 of the Bigar-Korkan area. Catchment scale groundwater model undertaken in 2020 (ERM, 2020).	12 months of surface water data collection upstream/downstream of Project and 12 months of groundwater sampling up gradient and downgradient of the Project as required under the Law on Water. Baseline work is required in the Ogasu Griljei (Dumitrov) catchment to the east.
Terrestrial ecology - biodiversity	Habitat and species surveys undertaken in 2013 of the area around Korkan, Bigar and Kraku Pester, before the Project footprint changed. Surveys included habitats and flora, mammals, birds, amphibians and reptiles. In October 2019 a Rapid Biodiversity Assessment updated the 2013 work (ERM, 2019).	Assessment to record data on presence of protected and threatened flora and fauna (e.g. trails, tracks, sighting). Dedicated seasonal surveys for flora, bats and large mammals as a minimum.

Environment	Work Completed	Work Planned
Aquatic ecology	Habitat and species surveys undertaken in 2013 of the area around Korkan, Bigar and Kraku Pester, before the project footprint changed. Surveys included phytobenthos, aquatic macrophytes, aquatic macro-invertebrates and fish, environmental DNA (eDNA) sampling and analysis to provide a list of fish and crayfish species present.	Targeted surveys to identify macroinvertebrates of conservation interest (e.g. caddisfly Helicopsyche bascescui).
Air quality	No work undertaken to date	12 months of air quality monitoring (oxides of nitrogen, nitrogen dioxide, sulphur dioxide, particulate matter (PM10 and PM2.5) and dust deposition with associated metals analysis, in line with good international practice.
Noise and vibration	No work undertaken to date	One (1) to two (2) weeks long term and short-term operator-attended monitoring.
Social baseline	No work undertaken to date	Household surveys, focus group discussions, informant interviews.
Land acquisition	Survey of land to be acquired undertaken in 2012, including details of ownership, and potential costs.	Census and socio-economic survey in line with good international practice.
Landscape	-	Observations and photographs at sensitive viewpoints.
Soils and Geology	Study by Belgrade Institute of Soil Science in 2007.	Soil sampling for land capacity (agricultural) and chemical quality (pollution) required by Law on Soils Protection.
Geochemistry	Sampling, static testing and analytical program to characterise Acid Rock Drainage / Metal Leaching (ARD/ML) capacity of the rocks to be disturbed conducted in 2020.	Complete the kinetic analysis. Undertake ARD/ML mass balance for waste rock dumps.
Heritage	Broad, high-level study of archaeology across licence area in 2012. Targeted walkover survey in October 2019 (ERM, 2020)to identify surface traces of buried remains, significant historical buildings and caves/rock shelters (documentary surveys) and identify whether further studies are needed.	Cultural Heritage walkover surveys of the full project footprint. Rural and historical railway structures that are likely to be disturbed shall be recorded archaeologically. Assess the need for further studies in areas to be disturbed that have high potential for Unidentified Archaeology and needs assessment for a LiDAR survey.

20.6.2 Environmental Evaluation: Potential Environmental and Social Risks and Fatal Flaws

Environmental and social risks associated with the Project were identified through a risk review in December 2018 which has been reviewed and updated in October 2020 to consider the PFS design and increased baseline knowledge. These risks will be further investigated and assessed as part of the ESIA process. The key risks are around surface water and groundwater during operation and especially in the closure phase, loss of several kilometres of riverine habitat and consequent impact on biodiversity, dewatering and diversions affecting springs, wells and streams during operations, infrastructure placed on small areas of potentially karstic ground, and economic displacement associated with land acquisition. Table 20.2 summarises the most important interactions between Project activities and the environment.

Key aspects of the potential environmental and social risks are detailed in the sub-sections below.

Table 20.2 – Initial Assessment of Interactions between Project Activities and Environment

Item	Aspect								Receptor									
	Emissions to air	Noise/vibration	GHG emissions	Water use	Discharges	Land take	Light	Hazardous Materials	Air quality	Soils, geology	Water	Flora /fauna	Land use	Health and safety	Fishing, hunting	Livelihoods	Cultural heritage	Landscape
Construction																		
Change in land use to mining						✓				✓	✓	✓	✓	✓	✓	✓	✓	✓
Pit construction	✓	✓	✓	✓	✓		✓	✓	✓	✓	✓	✓			✓			✓
Construction of infrastructure	✓	✓	✓	✓	✓		✓	✓	✓	✓	✓	✓			✓			✓
Road & Powerline construction		✓			✓	✓		✓		✓	✓	✓	✓				✓	✓
Operation																		
Pits	✓	✓	✓						✓		✓			✓				
Heap leach process								✓			✓	✓		✓				✓
Water supply				✓							✓	✓		✓	✓			
Dewatering				✓	✓						✓	✓						
Discharges					✓						✓	✓						
Chemical/fuel storage					✓			✓	✓	✓	✓	✓						
Waste management	✓		✓		✓			✓	✓	✓	✓	✓		✓		✓		✓
Employment														✓		✓		
Major accidents	✓		✓		✓			✓	✓	✓	✓	✓		✓		✓		
Closure																✓		
Removal of structures	✓	✓						✓		✓	✓			✓				✓
Rehabilitation and restoration	✓		✓	✓	✓				✓	✓	✓	✓	✓	✓				

20.6.2.1 Watercourses and Groundwater

A network of rivers and streams run through the Project area (Jagnjilo, Bigar, Valja streži, Tisnca, Crna reka, Vrkaluca, Valja Saka, Ogešu Griljei, Dumitrov, Ogešu Krloši, Valja Mare) with the catchments of the Zlotska Reka, Mlava, Veliki Timok and Crni Timok rivers (Figure 20.1). Watercourses in this area drain to River Danube, which is internationally protected. Serbia is a signatory to the International Commission for Protection of the Danube River and projects which may affect water quality of the Danube could trigger the need for transboundary engagement between the Serbian and neighbouring governments (Romania, Bulgaria).

The waste rock dumps, the heap leach pad and to a lesser extent, the mine pits themselves, will be sited within the uppermost catchment valleys of streams in the area. Diversionary channels have been planned to redirect surface water flow around these facilities.

Discharge of hazardous substances into groundwater is prohibited (Law on Water, Article 97), and so potential risks to surface water and groundwater quality are high. Events that could affect this include sedimentation, seepage of process chemicals including cyanide or hydrocarbons, leachate or runoff from acid generating rock, reduction in baseflow due to pit dewatering, planned discharges, and pollution during flood events, which typically take place during April snowmelt. Control of these risks is one of the main design objectives in the Mine Waste and Mine Water Management Plans.

The Mine Water Management Plan (SLR, November 2020) measures to avoid or mitigate these impacts, take account of the fact that rivers in this area are heavily channelised in steep valleys, which constrained the locations available for mitigation measures such as sedimentation ponds. The main mitigation will be to separate non-contact water by diversion channels discharging further downstream, or in the case of the VHL diversions, to another catchment to the east. All contact water, such as run-off and infiltration of waste rock dumps or from the Process area are collected in ponds and ultimately directed to the main Bigar Hill Process pond, adjacent to the Process area. This pond will have an industrial water treatment plant and all discharges will be treated prior to discharge into the Valja Saka stream. Elsewhere, contact water from in-pit sumps will be directed to two (2) Pit Ponds where it will be allowed to settle and then monitored for water quality before discharge at controlled rates within ecological bounds. If treatment is required, this will either be prior to discharge (e.g. simple liming to adjust pH) or will be piped to the main Process pond if more intensive treatment is required.

Parts of the site are underlain by karstic geology which poses a risk to the Project as infiltration and spread of pollutants to groundwater is typically greater through karst. There is potential for karst under the following areas:

- The Bigar Hill pit west pond, under the dam area and deepest part of the proposed pond location.
- The Korkan West dump pond, under the dam area and deepest part of the proposed pond location.
- The Korkan pit pond under the entire area.

The potential for karst risk in these areas will be investigated as part of the next project phase with geotechnical drilling and test pits planned. Both the Korkan and Korkan West ponds will be clay lined to prevent water egress. An area within the Bigar Hill Waste Rock dump will be lined and designated for rock-types identified as potentially acid generating (see section 20.8.1).

The transportation and use of cyanide associated with the heap leach process presents potential risks to surface and groundwater quality. DPM is a signatory to the International Cyanide

Management Code, which provides standards of practice for protection of communities and the environment during transportation of cyanide and specific usage requirements on handling, storage, operation, disposal and decommissioning. The leach pad has been sited away from the karstic limestone zone, where infiltration to groundwater could more readily occur, and leach pad design includes a robust liner system that collects the gold solution and also prevents infiltration to groundwater during operation. Various options are being considered for closure, but the current closure plan proposes thoroughly rinsing the heap leach pad in the closure period and when complete, removing water from the internal pond by pumping. A low permeability cap will be installed on the HLF to prevent large-scale infiltration and so volumes of water building up within the internal pond will be low in the long term.

The Project site lies outside existing water supply protection zones. There are, however, wells and springs in the Project area, some of which may supply households and agriculture with water. Abstraction of groundwater to supply the Project and the consequent dewatering may affect community water use, agriculture and habitats. The groundwater models and water balance being developed for the Project will help identify potential impacts and further engagement with the local community is needed to fully understand potential impacts to domestic wells. The Bigar Hill Spring, to the south west of Bigar Hill, has been identified by local authorities as a potential source of municipal drinking water. Designation as a municipal drinking water source could result in additional project constraints. DPM is working with the authorities to find alternative sources.

A Catchment Scale Groundwater model (ERM, 2020) has been developed to understand the complex water regime. It was designed to simulate the average regional water conditions, and to provide reliable boundary conditions and a relationship between recharge, discharge and overall aquifer transmissivity within the Project area. The results will be used in the impact assessment and to support various Project design and permitting processes, for instance: dewatering estimates for the pits; drawdown areas; and head changes under the leach pad.

20.6.2.2 Biodiversity

The Project area is characterised by a high diversity of species and habitats, including some of conservation interest, which are typical for woodland habitats and mountainous regions of Serbia where the Project is situated. Initial baseline surveys of the biodiversity of the site and surrounding area were undertaken in 2013, including surveys of terrestrial habitat, flora, mammals, birds, amphibians and reptiles. An update survey was undertaken in autumn 2019 to validate the findings of the 2013 survey and identify any changes in the baseline, or gaps in existing data.

The dominant habitat is beech woodland, interspersed with agricultural land comprising pasture and orchards with scattered homesteads (most seasonally occupied but now often abandoned). The majority of agricultural land is disused, mainly reverting to meadow, and supports good species diversity. Much of the woodland present shows signs of harvesting for timber production; some areas are composed of mature woodland and likely support high species diversity. Several small rivers cross the site, most being tributaries to Jagnjilo. Many ephemeral riverbeds occur in valley floors around the site, likely seasonal watercourses fed by spring snow melt.

During baseline surveys, at least 163 plant species have been identified as present or likely to be present on site, of which 24 are Serbian protected species and five are Serbian strictly protected species. A single species listed on the IUCN Red List of Threatened Species was recorded, the Vulnerable (VU) dropwort (*Filipendula vulgaris Moench*).

Twenty-eight (28) terrestrial mammal species were recorded as present or likely to be present during the 2019 surveys. Species recorded included hazel dormouse (*Muscardinus avellanarius*) and red squirrel (*Sciurus vulgaris)* whilst the area has the potential to support wolf (*Canis lupus*). All mammal species recorded as present or likely to be present on site are listed on the IUCN Red List (Europe) as Least Concern (LC). Three mammals are strictly protected, and 21 protected under Serbian law. Two (2) mammal species, wolf and wild cat (*Felis silvestris*) are listed on Annex IV of the EU Habitats Directive of species of community interest needing strict protection at a European level.

At least ten (10) species of bat are identified as using the Project area. During surveys in 2019, a static bat detector was placed in the entrance to Korkan cave, a limestone cave situated 350 m north of the site footprint. Species recorded using (likely roosting within) the cave included bent-wing (*Miniopterus schreibersii*), greater horseshoe (*Rhinolophus ferrumequinum*), Mediterranean horseshoe (*Rhinolophus euryale*), common pipistrelle (*Pipistrellus pipistrellus*), Myotis sp., brown long-eared (*Plecotus auritus*), particoloured (*Vespertilio murinus*), serotines (*Eptesicus serotinus*), Nathusius pipistrelle (*Pipistrellus nathusii*) and barbastelle (*Barbastella barbastellus*). All bat species are strictly protected under Serbian law and are listed under Annex IV of the EU Habitats Directive.

The 2019 surveys identified 114 bird species recorded or likely to be present within the Project area. Two (2) bird species likely to be present on site are listed as Vulnerable (turtle dove (*Streptopelia turtur)* and great grey shrike (*Lanius excubitor*)) and four (4) Near Threatened (NT) on the IUCN European Red List). All other bird species are listed as Least Concern at a European level. Of the species recorded during baseline surveys or likely to be present, 100 are strictly protected and 13 protected.

Seven (7) reptiles and nine (9) amphibian species recorded as present or likely to be present during the 2019 surveys. All amphibian and reptile species identified as present or likely to be present are listed as Least Concern on the IUCN Red List of Threatened Species. Of the species expected to occur on site, one (1) amphibian and one (1) reptile are protected and five (5) reptiles and eight (8) amphibians are strictly protected under Serbian law. Six (6) reptiles and three (3) amphibians are on Annex IV of the Habitats Directive.

Aquatic ecology surveys were undertaken in Autumn 2020, including surveys and environmental DNA (eDNA) sampling of streams. Three (3) species of interest were identified as potentially present and sensitive to low flows; mayfly (*Baetis pavidus*), caddis fly (*Helicopsyche bacescui*) and stone crayfish (*Austropotamobius torrentium*). The protected stone crayfish was identified through the eDNA analysis at the heap leach pad sample location, and is consistent with it having been recorded previously in the area in 2013, 2017, and 2018, This species is one of the most threatened species in Europe, is data deficient on the IUCN Red List, listed in Appendix III of the Bern Convention, is protected under the EU Habitats Directive and is strictly protected in Serbia. They are sensitive to long-term drought periods and poor water quality, as well as high flow velocities. A full impact assessment and design of mitigating flow regimes, modifications to streams or other offsetting strategies will be covered in the future ESIA. eDNA analysis also identified the Serbian strictly protected Danube bleak (*Alburnus chalcoides*) fish species, and the IUCN Near Threatened fish species silver carp (*Hypophthalmichthys molitrix*).

Vertebrate eDNA analysis was conducted at a single site at the same time as the 2020 aquatic surveys. This sample identified numerous species, including the following Serbian strictly protected vertebrates: yellow-bellied toad (*Bombina variegata*), common frog (*Rana temporaria*); European robin (*Erithacus rubecula*); Eurasian nuthatch (*Sitta europaea*); Eurasian blackcap (*Sylvia atricapilla*); and Eurasian water shrew (*Neomys fodiens*).

It is likely that all species present within the Project area have a wider distribution than that of the Project footprint alone.

No protected or internationally recognised sites for biodiversity were identified within 5 km of the Project, however, a number were identified in the wider biodiversity study area. Due to their distance from the Project, it is not anticipated that any of the protected sites identified will be impacted by the Project.

The presence of IUCN Red List Vulnerable species, and species listed on Annex IV of the Habitat Directive may meet the criteria for EBRD priority biodiversity habitat and features, requiring further investigation, evaluation and management including application of the mitigation hierarchy. To inform further assessment DPM has planned further baseline studies, including surveys for bats, protected flora and aquatic ecology which will further inform the environmental impact assessment.

20.6.2.3 Noise, Vibration, Nuisance Dust, Greenhouse Gas Emissions

Air quality baseline surveys or noise surveys are yet to take place. Historically, the main source of industrial pollution in area is the Bor smelter complex, some 20 km to the southeast. The baseline environment in the region is also known to have naturally elevated concentrations of arsenic and cadmium, which can be mobilised through dust. The Project will generate noise and dust from construction and operational activities and in closure. This may affect the people living permanently or seasonally in the area, as well as the flora and fauna. A full assessment of noise, vibration and dust will be required as part of the environmental impact assessment which will also need to take account of the existing background levels of arsenic, cadmium and other historical pollution.

The Project will generate greenhouse gas emissions that will need to be measured and managed in line with international good practice.

20.6.2.4 Soils and Geology

Preliminary soil surveys were undertaken in 2007 and further surveys are planned to align with Serbian regulations. In all cases where topsoils are to be disturbed, the principle will be to conserve these in stockpiles for ultimate reuse as a rehabilitation medium. Detailed geological mapping by DPM and others has been available for the PFS, providing high resolution understanding of the lithologies and structural geology underlying the different components of the Project area.

For disturbed subsoils and bedrock, the potential for acid rock drainage and metal leaching from is currently in the process of being assessed in a comprehensive screening program using industry standard static and kinetic testing, and is discussed further in Section 20.8 Mineral Wastes.

20.6.2.5 Cultural Heritage

There are no protected areas for cultural heritage or known registered archaeological features within the Project footprint. The region is well-known to be home to some of the earliest metallurgical technology in Europe and it is possible that there are buried remains in the Project area. Physical remains of past human activity on the site exist in the form of historic buildings and the wider historic landscape. No traces of buried archaeological remains have yet been found within the development area, although sites of ancient settlement are known in the wider region. Areas covered by walkover survey undertaken in 2019 and key features identified are shown in Figure 20.3. In terms of intangible heritage, the Project area is important to the Vlach community, an ethnic community in Serbia with its own language, dress and culture. Religious customs are connected with the land, including beliefs in sprits of the woods (e.g., fairies) and celebration of sacred trees. In addition, transhumance is a fundamental element of Vlach culture, with grazing on higher ground, including that in the Project area, in the summer. The remains of this are widespread in the form of many farms, mills and other structures, most of them abandoned. As material remains of a disappearing way of life, using the definitions set out in EBRD standards, these physical remains can also be considered tangible cultural heritage resources. Most of them have seen significant modernisation/rebuilding in recent decades.

The Žagubica-Bor railway crosses the Project area. This was built by forced labour during the Second World War. This narrow-gauge line – preserved in cuttings and embankments - runs along a meandering route to the south of the development areas in Valja Saka. Relatively little is known of its history other than it was constructed during the German occupation in very difficult conditions by forced labour – many of them Hungarian Jewish prisoners based in camps at Bor. The railway has historical significance, notably to those communities whose ancestors were forced to build it. Small sections of the railway are likely to be disturbed by haul road cut and fill in the south-east part of the Project area.

There will be a further assessment of the need for exploratory field studies in those areas to be disturbed that have high potential for unidentified archaeology as part of the environmental and social impact assessment (e.g. flatter areas, close to water with direct sunlight, where human settlement is more likely to occur). A LiDAR survey could also provide valuable insights in these areas.

The environmental impact assessment will address potential impacts on both tangible and intangible heritage. Any rural and railway structures that are likely to be disturbed shall be recorded archaeologically.

20.6.2.6 Social Setting, Land Acquisition and Livelihoods

There are scattered homesteads in the vicinity of the Project, most of these are seasonally occupied though many are abandoned.

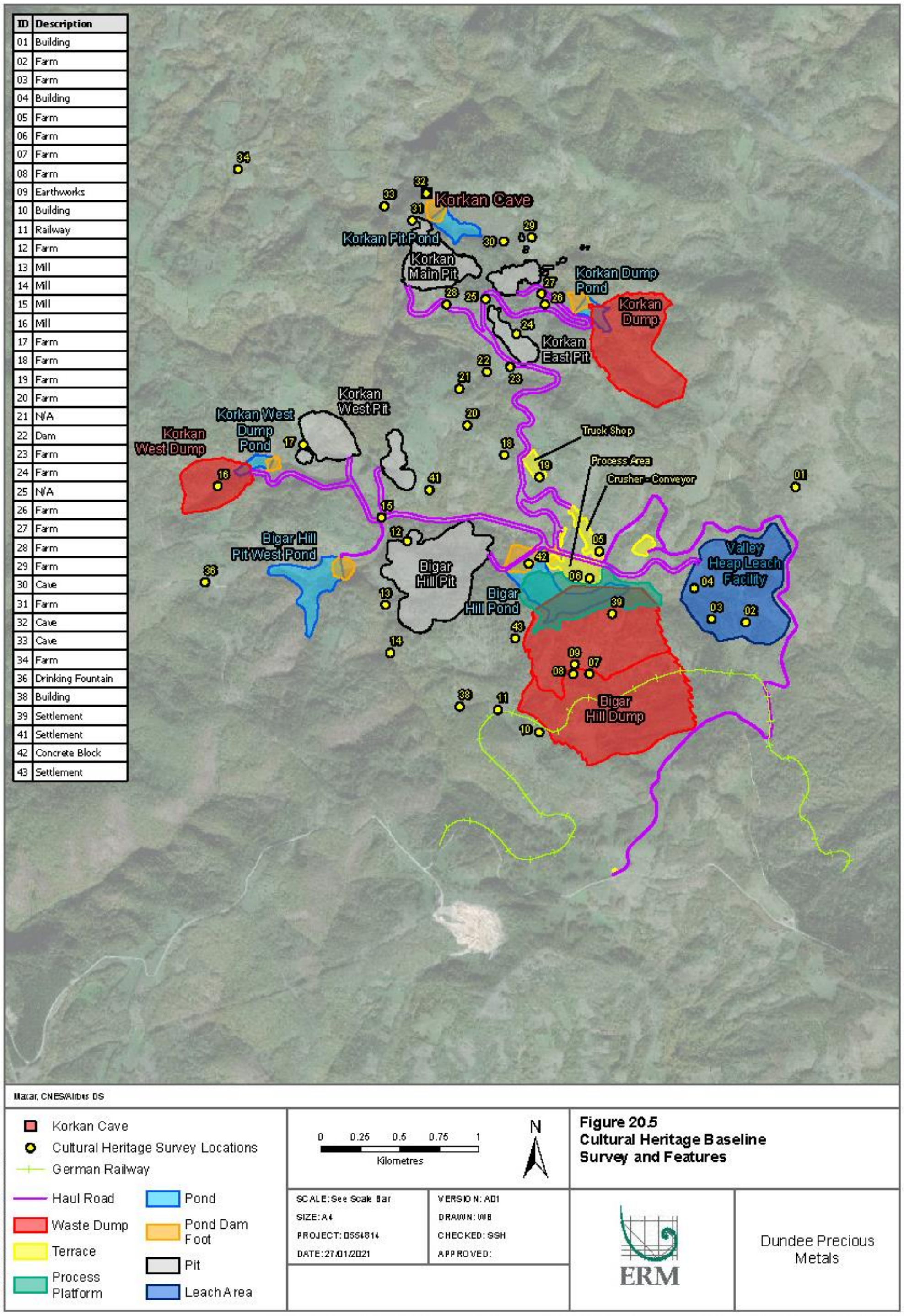
Land acquisition is required for the development of the Project, access to the site and the construction of associated facilities such as substations. Gaining access to this land requires purchasing land from private and public owners. Proof of land ownership is a prerequisite to obtaining key permits such as the Approval for Construction of Mine Structures.

Whilst habitation within the Project area is sparse and typically restricted to summer seasons, the community and relevant local government has expressed concern associated with economic displacement, particularly for those who seasonally utilise the Project area for summer season agriculture and grazing (transhumance, see Cultural Heritage).

A 2012 survey undertaken confirmed the presence of a number of small land parcels within the proposed Project footprint, of which most are used seasonally for small scale agriculture and grazing. A census and socio-economic survey of landowners is proposed to be undertaken at a later stage, to ensure that a full understanding of displacement is developed to form a detailed Livelihood Restoration Plan (LRP), outlining eligibility and entitlements for compensation and livelihood restoration. This LRP will be conducted in alignment with the EBRD guidelines, and good international practice.

At this stage, land acquisition through a voluntary willing-buyer willing-seller approach appears possible, however in the case that this does not occur, there are mechanisms under the Law on Expropriation that will allow the government to acquire immoveable properties for projects demonstrated to be within the public interest.

Figure 20.3 – Cultural Heritage Baseline Survey Areas and Features Identified in the Timok Gold Project Area



20.7 Social and Community Engagement

The Project will be subject to scrutiny by regulatory authorities and other stakeholders during the permitting process. There will be six formal hearing and consultation periods included within the spatial plan, strategic environmental assessment and environmental impact assessment processes and a range of other points where the public and other interested parties could comment. The Project will supplement these with dedicated stakeholder engagement activities.

DPM has worked to establish good relationships with the local community since 2007 and communications are managed through a Stakeholder Engagement Plan (Stakeholder Engagement Plan, 2019), Communications Strategy (Communications Strategy Timok Gold Project Serbia, 2020) and Grievance Mechanism (Grievance Mechanism). DPM has expanded its resources and conducted training in 2019 to facilitate transparent and meaningful community engagement. The Project team maintains a map of stakeholders and has earmarked vulnerable groups which will require targeted engagement.

Engagement activities include:

- Direct engagement with local community members on an individual basis.
- Engagement with community groups (schools, hunting groups).
- Engagement with regulators and local representatives.
- Corporate Social Responsibility actions including provision of flood relief and a medical vehicle.
- A telephone survey of attitudes towards mining undertaken in summer 2020.
- A forthcoming face-to-face survey of local community members.

DPM has also undertaken work to understand the likely economic impacts of the Project (Egzakta Advisory, June 2020), including contribution to direct and indirect employment, national economy through royalties and indirect impacts through the wider supply chain and on local and national businesses.

Potential issues raised so far by various stakeholders include:

- Concerns around the use of cyanide both at a governmental level and in the community, with specific questions around use of cyanide in heap leach mineral processing, associated with the storage and potential runoff of cyanide into surface water sources.
- Challenges associated with land acquisition and economic displacement. Currently a voluntary willing-buyer willing-seller approach is being pursued.
- An identified need for local employment opportunities, particularly within the Žagubica area, and across Eastern Serbia broadly where levels of unemployment are high. Unemployment is expected to rise as a result of the impact of COVID-19.

- Environmental impacts, particularly associated with air pollution, soil contamination, access to safe water sources, increased landslide and flood risks. These are based on experiences regionally and in the wider area from the long history of mining.
- Compliance with local and international permitting processes, including the requirements for independent consultation with interested and affected parties.
- The need for corporate social responsibility initiatives / donations to be provided to directly affected communities, with relevant priority investment areas clearly identified.
- The need to respect human and labour rights.

20.8 Mineral and Non-Mineral Wastes

20.8.1 Mineral Wastes

Mineral waste will be in the form of waste rock from the excavation of the pits and from some construction activities, disposed of at waste rock dumps adjacent to each of the pits.

The potential for acid rock drainage and metal leaching (ARD/ML) has previously been considered to be generally low to moderate, due in part to the oxidized nature of much of the ore, plus the presence of carbonate in some of the waste rock and ore. A geochemical characterisation study of the ARD/ML with methodologies for sampling, leaching and analysis to internationally accepted standards, such as Price et al. (1997) and the GARD Guide (INAP 2014) was started in 2020. An initial series of short term ('static') tests have been completed and longer term ('kinetic') geochemical tests will be conducted to geochemically characterise waste rock materials. The "static" tests generally comprise screening tests to determine the potential magnitude, whereas "kinetic" tests allow the development of chemical reactions to be observed, and the time-related chemical release rates of these reactions to be determined.

All the selected samples have been subject to the initial static-test screening phase. A kinetic-test phase will follow. The sampling phase, with a statistically robust set of samples, has defined a preliminary characterisation of transitional and oxide waste rock material that will be mined or exposed (including unmineralised country rock). The number of samples was designed to be sufficient to adequately represent the variability within each geological unit and waste type, with a focus on the on materials with the highest sulphur content.

Static testing included standard approaches of acid-base accounting, net acid generation, major elements by XRF and trace element by ICP-MS. Following assessment and analysis of these results, a sub-set will be selected that has the highest potential for ARD/ML and will go forward for kinetic testing.

The results of the ARD/ML study will be used both in the general principles and in detailed design for water management, especially of contact / non-contact waters, water balance and water treatment. This is also of importance to the environmental impact assessment for soils, surface water and groundwater, and the ecologies they support. Static short-term ARD tests on a representative sample distribution of waste rock types indicate acid rock drainage potential in around 10% of samples (5% of oxide/transitional and 13% of fresh samples) and further test work has been recommended only for this particular volume. Subject to further investigation and regulatory approval, it has been assumed that the majority of waste could be stored in unlined conventional dumps, based on the low ARD potential indicated.

The approach to mine waste management is set out in the statutory project Mine Waste Management Plan, developed in line with the Serbian regulations on characterisation, classification and reporting on mining waste (ORGS 53/17) and good international practice (e.g. EU Extractive Waste Directive and International Finance Corporation (IFC) Health and Safety Guidelines on Mining). Waste rock dumps will only be lined in areas designated for rock-types identified as potentially acid generating, currently anticipated to be an area or cell within the Bigar Hill Waste Rock dump, which will drain to the Bigar Hill / Process Pond, where water treatment is available. Subject to confirmation from the ongoing ARD/ML static and kinetic tests and other EU waste classification tests, the rest of the waste rock dump areas, that do not have potentially ARD/ML material, will not be lined, although surface water flow will be diverted, and runoff and leachate will be collected in lined toe-ponds. The Korkan and Korkan West toe-ponds can be pumped to the Bigar Hill / Process Pond, where water treatment is available, should the need occur. Normal operations will see the conventional unlined waste ponds providing settling treatment, sampling and monitoring opportunity prior to surface discharge to the environment.

The plan further describes three different engineered capping designs, intended to impede vertical water infiltration to three different degrees, ranging up to a low permeability sealing cap. All will support revegetation layers, and the lowest permeability option will be used at least over the valley heap leach facility. Elsewhere, cap design will depend on the material's intrinsic ARD/ML characteristics, and this will be one of the main management measures for the control of ARD/ML in the closure period.

20.8.2 Non-Mineral Wastes

Non-mineral wastes will include non-hazardous and hazardous materials such as packaging, used oil, batteries, food, medical waste and sewage. The Project will develop a waste management inventory as part of the design process and a strategy for disposal of each waste stream, following the waste hierarchy (reduce, reuse, recycle, treat, dispose) and in line with Serbian regulations and international good practice. Suitable third-party waste carriers and treatment/disposal sites will be identified and the details of the approach for storage, transportation, treatment and disposal of each waste stream will be set out in the Project waste management plan.

20.9 Mine Closure, Aftercare and Remediation

An outline Mine Closure Plan (MCP) (ERM, 2020) has been drafted as part of the PFS. The MCP includes a review of applicable legislation and permit requirements, and is designed to be in line with Serbian requirements (ORGS 27/9) and international good practice (e.g. EU Extractive Waste Directive, EU Best Available Techniques, IFC Health and Safety Guidelines on Mining, International Council on Mining and Metals (ICMM) Good Practice Guide), and DPM internal Standards (Corporate Responsibility Policy. The limited operational lifespan of the Project makes careful consideration of the closure vision and MCP particularly important. The limited operational lifespan of the Project makes careful consideration of the closure vision and MCP particularly important.

The Project's approach to closure will be to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use. The current MCP closure vision is to restore the site as far as practicable to pre-mining land use and status. Further closure post-closure options such as tourism, industry and agricultural future uses will be studied in greater detail in the feasibility stage of the project. The aim of the closure plan will be to minimise or eliminate long term active aftercare such as water treatment requirements.

The heap leach facility and waste dumps will remain after site closure as low, unsaturated tabular landforms, which will be placed over a sealed base and will ultimately be capped with low permeability covers. Appropriate stabilisation and covering options have been developed, with three capping designs offering and range of vertical impedance to rainfall, ranging up to low permeability seals. These will be tested during the operational phase and the need for different levels of sealing caps will be defined at that stage. It is already planned that the valley heap leach facility will require the lowest permeability cap. Elsewhere, cap design will depend on the material's intrinsic ARD/ML characteristics, and this will be one of the main management measures for the control of ARD/ML in the closure period. Details of closure and aftercare of the waste rock dump and heap leach are set out in the first draft of the mine waste management plan (in line with ORGS 53/17), to be submitted as part of the application for the mine waste management permit. These provisions will be incorporated into future iterations of the MCP.

Monitoring of groundwater and surface water (levels, chemistry) will be undertaken through-out the LOM to monitor the impacts of mine dewatering and develop an appropriate closure strategy to ensure that adverse impacts on hydrogeology and hydrology do not occur post-closure. Upon closure there will be a phase of further site investigation, risk assessment and regulatory liaison to identify any sensitivities or requirements relating to post-closure soil or water quality and to develop remedial action plans that may be required in this regard, including detailed scopes and costs for work programs. This will be followed by pre-demolition assessments to develop a demolition contract package (based on structural surveys, collation of structural details of all buildings) indicating what buildings, facilities and equipment will be decommissioned. At this stage, community continuity projects will assess the impact of the closure on the community, and mitigation will be developed in a program of initiatives that will be in accordance with DPM's corporate commitments to communities and local economies. Decommissioning will be undertaken by specialist contractors who will be contractually required to operate at all times in accordance with relevant legislation, DPM standards and international best practice.

Decommissioning activities will include the following:

- Decommissioning, flushing and safe rendering of the heap leach and cyanide containing facilities, followed by developed closure plans and monitoring.
- Securing open pits with rock berms, rendering them safe and stable and allowing them to flood.
- Restoring waste rock dumps and installing adequate drainage to prevent erosion and enhance physical stability.
- Dismantling site buildings – some buildings may be retained as a part of a designated end use of the site.
- Decontaminating machinery and equipment.
- Scarifying, grading and contouring roads.
- Removing chemicals, waste and explosives from the site by a licensed waste carrier and recycling and disposal in line with Serbian regulation and good international practice.
- Testing, excavating and removing any contaminated soils.
- Draining surface ponds, except for the main process pond at the toe of the Bigar Hill waste rock dump and the pond at the toe of the Korkan waste rock dump.
- Revegetating using species compatible with local habitats.

Throughout decommissioning and into the closure period, site monitoring will include a program of surface water and groundwater, air emissions and ecological monitoring. Monitoring will be conducted after closure for a defined period or until pre-defined success criteria are achieved, such as an appropriate concentration in the discharge is guaranteed (as defined by competent specialists). Following completion of the monitoring period, any monitoring wells will be sealed and abandoned.

At this conceptual stage of the MCP, the closure costs are expected to be broad, conservative cost estimates only. Refer to Section 21 of the Report for closure amounts included in the economic cost model, spread over three years at the end of the mine life. This cost covers final site remediation. The Project will re-iterate closure planning and costing exercises throughout the planning stages and operational life, with the next revision at the FS stage.

20.10 Health and Safety

An outline Occupational Health and Safety Management Plan (OHSMP) has been drafted as part of the PFS. The OHSMP includes a review of applicable legislation and is designed in line with Serbian requirements (ORSG 101/05; ORSG 94/06, 108/06, 30/10; ORSG 29/06, 72/06, 62/06; ORSG 60/06;

ORSG 62/07; ORSG 120/07, 93/08; ORSG 72/06, 84/06; ORSG 23/08; ORSG 14/09; ORSG 21/09; ORSG 23/09; ORSG 106/09, 06/10, 15/10), international best practice (e.g., IFC Environmental, Health and Safety Guidelines and ISO45001, International Cyanide Management Code) and DPM Internal Standards (Health and Safety Policy, Four Concepts and Health and Safety Model).

The OHSMP describes how occupational health and safety (H&S) will be managed during all stages of the Projects development to avoid and appropriately manage risks that could potentially affect Project personnel. Community safety, health and welfare is also considered where communities are envisaged to be affected by Project activities. The OSHMP will:

- Provide a safety and health assessment and plan prior to commencement of scheduled work activities performed by DPM and contractor workforces within DPM properties.
- Provide a preliminary assessment and review with respect to the proposed Project construction activities within an operating area, and their effect on the operations personnel, within that area.
- Provide a preliminary assessment and review with respect to the facility operating activities, and their effect on construction personnel and related activities.
- Identify Serbian legal and best practice requirements relating to H&S management, which are applicable to the Project.
- Describe how this framework will continue to develop, in line with Project progress.

During the PFS, DPM appointed a Programme Director tasked with developing the Project Design, and an Engineering, Procurement and Construction Manager. DPM has provided the appropriate levels of oversight through regular communications, project meetings and participation in risk assessment exercises. Serbian Law on Mining and Geological Exploration sets out the roles and responsibilities of a number of key personnel, during the operational phase DPM will appoint individuals to these roles to ensure ongoing appropriate management of H&S.

Serbian regulations (Article 129 of the Law on Mining and Geological Exploration) require the Company to:

- Arrange occupational H&S in accordance with the specificities and dangers that may arise.
- Organize occupational H&S related activities in accordance with this Law and regulations governing the occupational H&S.
- Provide personal protective agents and personal protective equipment for the employees.
- Provide protection against fire, damages, accidents and chemical and other accidents, and organise rescue operations.
- Organise training for workers in the field of occupational H&S and rescue operations in the case of sudden danger threatening the lives of people and safety of the facilities, in accordance with the established schedule and program throughout the year and check personnel's knowledge once a year.

The following measures will be implemented to ensure on-going management of H&S risks:

- A project wide risk assessment to develop a preliminary project risk register.
- Day to day risk assessments undertaken as part of managing construction and operational risks and changes to the work environment.

As the Project progresses an Implementation Plan will be produced which specifies the resources needed to meet the objectives and responsibilities for implementing the OHSMP, it will also establish timeframes for completion.

20.11 Environmental, Social and Health and Safety Plans and Commitments

20.11.1 ENVIRONMENTAL AND SOCIAL MANAGEMENT PLANS AND PROGRAMMES

In addition to baseline data gathering listed in Table 20.1, the Project will continue to enhance and develop the following plans and programmes:

- A Permitting Plan.
- A Stakeholder Engagement Plan & Consultation Strategy.
- An Outline Mine Closure Plan.
- Environmental Design Criteria
- An Environmental Protection Framework Plan including Monitoring Plan.
- A Change Management Procedure.
- An Occupational Health & Safety Plan.
- An Emergency Response and Contingency Plan.
- An Auditing and Continuous Improvement Plan.

20.11.2 ENVIRONMENTAL AND SOCIAL COMMITMENTS

The Project maintains a commitment register (DPM, 2020) which documents environmental design and performance commitments made during the PFS.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

21.1.1 INTRODUCTION

The Capital Cost Estimate (Capex) prepared for this Report is based on the scope of work as presented in earlier sections of this Report.

The Capex consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital and closure have been completed and are depicted separately from initial Capex requirements.

The Capex is reported in United States Dollars ($).

21.1.2 EXCHANGE RATES

The base currency for this PFS is the US Dollar (USD). For information purposes, Table 21.1 lists the currencies that could potentially be used, along with the conversion exchange rates based on an average of the previous 90 days prior to and including November 2020.

Table 21.1 – Currency Conversion Rates

Currency	Code Name	Exchange Rate	US Dollar
USD	US Dollar	1.00000	1.0000
CAD	Canadian Dollar	1.30000	0.7692
CNY	Chinese Yen	0.14725	6.7912
GBP	British Pound	0.76000	1.3158
EUR	Euro	0.86957	1.1500
AUD	Australian Dollar	1.39300	0.7179
ZAR	South African Rand	0.06049	16.5306
RSD	Serbian Dinar	0.01001	99.9121

21.1.3 SCOPE OF THE ESTIMATE

The Capex includes the material, equipment, labour and freight required for the mine, process facilities, infrastructure and services necessary to support the operation.

The Capex also includes for the estimates developed and provided by external sources such as SLR for the heap leach facility, and waste rock management facility, water management and closure related to heap leach and waste rock facilities.

The Capex prepared for this PFS is based on a Class 4 type estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11 with a custom accuracy band of +30% -15%. Although some individual elements of the Capex may not achieve the target level of accuracy, the overall estimate falls within the parameters of the intended accuracy.

The reference period for the cost estimate is 4th Quarter 2020.

21.1.4 Capital Cost Summary

Table 21.2 presents a summary of the initial Capex. Sustaining Capex is distributed over the LOM, separately indicated from the initial Capex. All costs relate to the area within DRA's scope unless otherwise noted. Owner's costs, its contingencies and risk amounts have been included in this Capex.

The purpose of this section is to outline the methodology by which the Capex was developed for the Project. The scope of the Capex includes the open pit mine, heap leach and ore handling, processing plant and support on-site infrastructure.

Table 21.2 – Initial Capex Summary by Major Area

Description	Cost ($ Millions)
Pre-stripping	8.6
Mining (mine fleet, haul and access roads)	36.7
Processing (heap leach, processing plant)	52.7
Infrastructure	17.2
Waste rock facilities and Other	16.7
Total Direct Costs	**131.9**
Construction Indirect & owner's costs	42.1
EPCM	15.8
Total indirect costs	**57.8**
Contingency	*21.0*
Total Initial Capital	**211.0**
Life of mine	
Sustaining capital expenditures	24.4
Closure and rehabilitation costs	23.3

Figures may not add due to rounding.

21.1.5 SUSTAINING CAPITAL

Sustaining capital comprises the replacement of mobile equipment, expansion of waste facilities and systems as shown in Table 21.3.

Table 21.3 – Sustaining Capital Cost Estimate

Area	Sustaining Capital ($ US '000)
3710-Heap Leach Pad	8,717
1311-Haul Road -Plant to Korkan Pit	1,630
1315-Haul Road to Korkan Dump	813
1316-Haul Road to Korkan West Dump	608
1317-Haul Road to Korkan Cut1	777
1318-Haul Road to Korkan Cut2	585
1319-Haul Road to Korkan West 2	273
1325-Overhead Powerline	910
1134-Korkan West Waste Dump	451
1136-Korkan Waste Dump	1,095
1200-Open Pit Mining Equipment	1,975
29100-Contractor Indirects for Sustaining Capital	1,781
21100-Closure-Phase IV-HLF	3,681
21200-Closure-Phase IV-Bigar Hill Dump Waste	12,507
21300-Closure-Phase IV Korkan West Dump Waste	971
21400-Closure-Phase IV-Korkan Dump Waste	2,983
21500-Closure-Plant & Haul Roads	3,146
31000-Contingency for Sustaining Capital	4,822
GRAND TOTAL	**47,727**

Figures may not add due to rounding.

21.1.5.1 Waste Facilities

Initial results, as described in Section 18, indicated the need for 5 Mt waste rock containment facility for Potentially Acid Generating Rock (PAG). A lined facility is anticipated within the Bigar Hill waste dump to accommodate such waste. This facility was not specifically designed as PAG test work data was received towards end of the PFS. A conservative allowance is included in the Capex for a lined waste facility suitably sized for approximately 4.8 Mt (10%) of waste rock. The associated lined facility Capex cost was approximated using the first phase construction of Bigar Hill fully lined facility. A breakdown of the costs associated with the lined waste facilities are provided below.

21.1.6 Closure and Rehabilitation Costs

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. SRL advised the quantities associated with the Project closure and the prices were included in this estimate as Sustaining Capex.

The following methodology for disposal and rehabilitation is proposed for various sections of the Project:

- Process plant and infrastructure: To be deconstructed to ground or foundation level. Any remaining foundations left at ground level will be covered with a 100 mm thick layer of topsoil, which will be fertilised and seeded.
- Access and haul roads: Based on stakeholder discussions, these are likely to remain in place for the use of future forestry and farm related activities. In situations where this is not the case, roads would be deep ripped, culverts removed, re-contoured, and covered with a 100 mm thick layer of topsoil, which will be fertilised and seeded. A cost for the recontouring has been allowed for.
  - Mine infrastructure: As for the process plant and infrastructure.
  - Mine pits: Rehabilitation costs are included in the mining costs at $0.09/t.
  - Waste dumps and HLF closure covers have been designed and quantified.
  - ROM pad and any temporary stockpiles: As for mine pits.

21.1.7 Estimate Structure

The Capex has been structured into the following major categories:

- Direct Costs;
- Indirect Costs;
- Contingencies;
- Owner's Costs.

21.1.7.1 Direct Costs

Direct costs are those expenditures that include supply of equipment and materials and construction labour at site.

21.1.7.2 Indirect Costs

Indirect costs are those expenditures covering temporary construction facilities, transport of equipment and materials to site plus engineering, procurement, and construction management (EPCM) services together with the supervision of and commissioning of the works.

21.1.8 Estimation Methodology

The Capex has been assembled on an electronic spreadsheet using the following general methods of calculation.

21.1.8.1 Quantity Development

Quantities were developed based on the design quantity takeoffs and supplemented by estimates or allowances.

The following engineering documents were produced in order to complete the estimate of quantities:

- Site Plot Plans;
- Equipment List;
- Electrical Equipment List;
- Process Flow Diagrams;
- Preliminary P&IDs;
- Layouts Drawings;
- Electrical Single Line Diagrams;
- Sketches.

21.1.9 Productivity and Construction Man-Hours

The base unit man-hours are those for standard North American construction (Gulf units). Productivity factors were applied to the base unit man-hour for the mechanical equipment, Electrical, Instrumentation and controls (EC&I) commodities. The productivity factors were based on the quality of workers, work hours per week, and complexity of the work. Other factors considered were weather, travel and the turnover of workers. The productivity factors are based on DRA database benchmarking for similar projects.

For this Project, a productivity factor of 100% has been applied to these norms to reflect the estimated hours considered applicable in this region of Serbia. In other words, the norms are multiplied by a factor of 2 to give actual construction hours.

Please note that any productivity losses due to labour market, schedule requirements (fast track), boom period or industrial climate have not been taken into consideration.

21.1.10 Direct Costs, Estimate and Scope Basis

The following sub-sections summarise the estimation derivation and basis adopted for the Capex.

21.1.10.1 Bulk Earthworks

Earthwork rates were selected following adjudication of tenders and clarifications with the contractors.

Large waste fill volumes related to pond embankments will be provided from suitable waste rock following mine pre-stripping activities. To affect material supply during the construction period, suitable pre-stripping will commence earlier than required.

It is assumed that the cost to establish a quarry operation for the extraction, crushing, screening and stockpiling of suitable backfill materials was not included in these rates as the materials will be provided to Contractors free of charge by mining operations.

21.1.10.2 Detail Earthworks and Concrete

Detail earthworks and concrete rates were provided by local contractors with recent experience in the region during a formal tendering process. Concrete rates were selected following an evaluation of the tenders and clarifications with the contractors.

The rates are assumed to include cost of operating a concrete batch plant, established at a nearby location and include the necessary washing and screening plants for sand and aggregates for which production unit rates were provided.

21.1.10.3 Structural Steel

Structural steel fabrication, detailing and erection rates were received from local supplier and local contractor. Unit rates were priced per tonne of steel, including shop fabrications, with separate rates for erection on site.

21.1.10.4 Mechanical Equipment

Generally, offshore supplied equipment was quoted FOB port of embarkation. There was a separate allowance for freight and insurance during transportation and import duties.

Mechanical and electrical equipment were divided into the following categories:

- Major equipment requiring budget quotations;
- Minor equipment requiring budget quotations;
- Equipment priced from in-house data;
- Allowances.

21.1.10.5 Mechanical (Platework and Tanks)

Budget prices were obtained for a sample range of carbon steel, alloy steel and lined tanks to establish unit prices per tonne.

Prices for mechanical bulks including chutes, ducting and insulation were based on locally available plate purchase, detailing, fabrication and installation.

21.1.10.6 Piping

The cost for process plant piping was factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience.

Overland piping was quantified with material take-offs and priced using rates from suppliers and fabricators of piping systems.

21.1.10.7 Electrical

The electrical equipment list and the single line diagrams were used to form the basis for the electrical equipment estimate.

Power and control cables were factored using DRA's historical units.

21.1.10.8 Instrumentation and Controls

The cost for process plant Instrumentation & Controls were factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience.

21.1.10.9 Plant Bulk Earthworks, Drainage, Storage Ponds and Access Roads

Plant bulk earthworks are based on necessary cut to fill and embankment construction suitable for plant foundation and road construction. Drainage consists of earthen side drains and under road concrete culverts. Water storage ponds include for HDPE liners. Roads are constructed to regional standards and not paved. It is presumed suitable materials are available from borrow pits reasonably adjacent to the Project site.

21.1.11 INDIRECT COSTS

21.1.11.1 Contractor Indirects

Contractor's indirect costs include all contractors' overheads such as contractual requirements (safety, sureties, insurance, etc.), the site establishment and the removal thereof, and company and head office overheads.

The contractor field indirect costs also include:

- Construction temporary facilities which include:
  - Offices, mess halls, lunchrooms, bathrooms, first aid, showers, laundry;
  - Warehouses & yards, shelters, etc.
  - Power generation
  - Aggregate & concrete batch plants;
  - Water systems;
  - Temporary power;
  - Maintenance & clean up;
  - Personnel transportation.
- Temporary services:
  - Survey;
  - Inspection;
  - Quality Controls;
  - Medical Services;
  - Security;
  - Heating;
  - Fuel supply;
  - Fuel stations;
  - Water;
  - Sewage and Waste Disposal;
  - Third Party Consultants;
  - Warehousing.
- Construction Equipment
  - Cranes;
  - Vehicles;
  - Mobile Equipment;
  - Specialty Equipment.

It is important to note that contractor costs to construct the Project are all included in the Direct Costs. Only the costs associated to manage the contract are included in Indirect Costs.

Unit prices submitted by contractors are "all-in" rates, which include contractor's construction equipment, operators, insurance, overhead and profit.

21.1.11.2 Capital Spares and Inventory

Capital Spares and Inventory include:

- Capital Spares

  Capital Spares for major mechanical equipment were quoted by vendors/suppliers. For minor mechanical equipment, an allowance for spare parts is based on 5% of equipment purchase price.

- Operational Spares

  Two-year Operational Spares for major mechanical equipment was quoted by vendors/suppliers. For minor mechanical equipment, an allowance for operational spare parts is based on 4% of equipment purchase price.

- Critical Spares

  Critical Spares for major mechanical equipment was quoted by vendors/suppliers. For minor mechanical equipment, an allowance for critical spare parts is based on 3% of equipment purchase price.

- Commissioning Spares

  Commissioning Spares for major mechanical equipment was quoted by vendors/suppliers. For minor mechanical equipment, an allowance for critical spare parts is based on 3 % of equipment purchase price.

- Initial Fills

  An allowance for one month of initial first fills was based on quantities developed by the Process Group and a unit rate was applied.

The estimate includes the cost of the required material quantities for the first start-up fill of the following:

- Chemical reagents;
- Cement;
- Quicklime;
- Water consumption;
- Fuels;
- Lubricants;
- Oils;
- Others.

21.1.11.3 Freight and Logistics

The cost for transport including insurance and demurrage cost was factored as a percentage of the overseas equipment cost based on DRA's historical data and experience. In this case, a factor of 8% was applied to the offshore equipment costs.

The cost for in-land transport was factored as a percentage of the equipment cost based on DRA's historical data and experience. In this case, a factor of 5% was applied to the offshore equipment and bulk materials cost depending on the country of origin.

The freight and logistics costs include for brokerage and agent fees, warehouse services, and import. It is assumed that there will be no requirement for air freighted items to site.

21.1.12 OWNER'S COSTS

The Owner's Costs was compiled by DRA with assistance from DPM.

The Owner's Costs include, but are not limited to:

- Land acquisition and rights of way;
- Definition drilling, assaying and related reports and models;
- Owner's Project administration team (allowance);
- Owner's pre-production and mine development;
- Health safety and security;
- Closure and rehabilitation;
- Sunk costs (covered in the economic evaluation as supplied)
- Process rights, royalties, license fees, technology fees and the like (addressed in the financial model);
- Project financing and interest charges (covered in the economic evaluation);
- Training and recruiting of plant operating personnel (Operational Readiness);
- Working capital (covered in the economic evaluation) (Operational Readiness);
- Start-up and commissioning costs (client commissioning team);
- All production costs prior to commercial production;
- All costs associated with the permitting process;
- Builder's risk insurance;
- Cost of this or any other study;
- Community relations;
- Emergency response;
- Environmental / permitting / government relations;
- Finance and general administration;

- Human resources;
- Information technology;
- Insurances;
- Legal;
- Power (free issue fuel and electrical for construction);
- Site security;
- Site maintenance and mobile equipment;
- Import duties;
- Taxes (addressed as part of economic evaluation);
- VAT (covered in the economic evaluation);
- Clinic;
- Escalation (covered in the economic evaluation).

21.1.13 CONTINGENCY

Contingency was included in this Capex to cover items which are included in the scope of work, but which cannot be adequately defined at this time due to lack of accurate detailed design information.

Contingency covers uncertainty in the estimated quantities and unit prices for labour, equipment and materials contained within the scope of work.

Contingency, as defined herein, is not intended to cover items such items as labour disputes, change in scope, or price escalation.

A probabilistic contingency analysis was conducted using Monte Carlo simulation @risk software. P50 value was selected.

21.1.14 QUALIFICATIONS

All estimates are developed within a frame of reference defined by assumptions and exclusions. It is grouped under qualifications. Assumptions and exclusions are noted below.

21.1.14.1 Major Assumptions

The following are assumptions on which the Capex is based:

- The Capex reflects an Engineering, Procurement and Construction Management (EPCM) type execution wherein an EPCM contractor will provide the design, procurement and construction activities for all aspects of the Project. All sub-contracts would be managed by the EPCM contractor;
- The Capex assumes minimal requirement or limitation with respect to local content in terms of labour, materials, equipment and economic impact;

- The Capex assumes no restriction to site at any time during execution of the Project;
- The Capex is based on no delays in execution from time of contract award to the selected EPCM contractor as a result of either of the following:
  - Owner's financing charges;
  - Owner's permitting delays.
- The Capex assumes no delays as a consequence of labour disputes;
- The Capex assumes no underground obstructions for all excavation activities to be performed during the construction;
- It is assumed that suitable borrow pits for building materials are in close proximity and where there is a requirement for select fill it can be produced with a minimum of screening and water conditioning.
- The Capex is based on the milestone schedule presented in the Report.

21.1.14.2 Exclusions

Unless specifically included in the Owner's Cost, the Capex excludes allowances for the following:

- The estimate reflects the mechanical equipment list accompanying this Report.
- Escalation during construction;
- Interest during construction;
- Schedule delays exceeding 2 weeks and associated costs;
- Scope changes;
- Unidentified ground conditions;
- Extraordinary climatic events;
- Force majeure;
- Labour disputes;
- Insurance, bonding, permits and legal costs;
- Receipt of information beyond the control of EPCM contractors;
- Schedule recovery or acceleration;
- Cost of financing;
- Property taxes, corporate and mining taxes, duties;
- Sunk costs;
- Research and exploration drilling;
- Salvage values;

- VAT or any like tax is not included as part of the CAPEX estimate presented;

There will be no Construction camp and catering in this Capex. It is assumed that local contractors can find local accommodation.

21.2 Operating Cost Estimate

This section describes the basis of estimate and approach taken in calculating the operating costs for the Project.

The Operating Cost Estimate (Opex) is presented in United States Dollars (USD). DRA developed these operating costs in conjunction with DPM, with specific inputs provided by external consultants.

The following are examples of cost items specifically excluded from the Opex:

- Value Added Tax (VAT);
- Project financing and interest charges.

21.2.1 Opex Summary

The Opex summary is depicted in Table 21.4.

Table 21.4 - OPEX Summary

Description	Annual Average ($ M)	LOM Totals ($ M)	Average/t ($/t)
Mining	19.0	152.6	7.89
Processing	12.4	99.1	5.16
G&A	12.4	33.8	1.76
Water Treatment	0.5	5.2	0.27
Total	**44.2**	**289.6**	**15.07**

Figures may not add due to rounding.

21.2.2 Mining Opex

Mine Opex were developed on a first principles basis following the tonnage and haulage requirements as defined by the mine plan in Section 16 of the Report.

21.2.2.1 Mine Cost Summary

The Mine Opex summary is shown in Table 21.5.

Table 21.5 – Mine Opex

Description	LOM ($)	Cost ($/a)	Cost ($/t)	Total Cost (%)
Equipment	95,595,000	11,949,000	4.94	62.64
Manpower	29,047,000	3,631,000	1.50	19.03
Explosives	26,975,000	3,372,000	1.39	17.67
Technical Services Equipment	1,000,000	125,000	0.05	0.66
Total Opex	**152,617,000**	**19,077,000**	**7.89**	**100.00**

Figures may not add due to rounding.

21.2.2.2 Mine Equipment Costs

Equipment unit operating costs were estimated using DRA database costs for each piece of equipment. Those costs were developed by DRA over time from vendor data and mine operation data. Unit costs for each type of equipment is provided in Table 21.6. These unit costs were then multiplied by the equipment hours necessary for each type per annum to obtain annual equipment costs. Contractor mine equipment costs are included in the unit mining costs.

Table 21.6 – Mine Equipment Unit Operating Cost

Equipment Type	Capacity	Fuel, Oil, Lube ($/h)	Tires ($/h)	Parts ($/h)	Total ($/h)
Truck	54t	47.33	9.41	27.76	84.5
Shovel	6.8 m^3	80	0	68.18	148.18
Loader		109.7	18.32	97.84	225.86
Drill		71.15	0	65.5	136.65
Grader		34.53	2.92	34.3	71.75
Track Dozer		52.25	0	52.34	104.59
Grade Control Drill		$0.436/t			
Water Truck		57.55	10.8	14.2	82.55
Support Backhoe		4.9	0	4.74	9.64

Equipment Type	Capacity	Fuel, Oil, Lube	Tires	Parts	Total
		($/h)	($/h)	($/h)	($/h)
Pump Truck		5.57	0.11	0.64	6.32
Light Plant		3.6	0.01	0.26	3.87
Pick-Up Trucks		5.57	0.11	0.64	6.32
Man Bus		7.4	1.01	0.8	9.21
Lowboy and Tractor		5.99	1.69	10.76	18.44
Lube/Fuel Truck		7.62	0	1.25	8.87
Tire Manipulator		2.43	0.49	0.71	3.63
Mechanic Truck		6.23	0	1.51	7.74

21.2.2.3 Explosives Cost

Explosives costs were estimated for the drilling pattern given in Section 16 of the Report. It is assumed a local explosives contractor will supply and provide all necessary equipment, manpower and explosives and supplies at an all-inclusive cost of $1.50 /kg of explosives or $0.41/t mined.

21.2.3 PROCESS PLANT OPEX

The process plant Opex represents the estimated annual Opex for the heap leach over the LOM. The heap leach plant is expected to operate for a period of 8 years, with 2 Mt processed in Year 1, 2.5 Mtpa from Year 2 to 7, and 2.2 Mt in Year 8. The Opex covers crushing and conveying, agglomeration, stacking, heap leach pad, leaching and irrigation, and the ADR facility.

The Opex is divided into six (6) categories: manpower, power, reagents, assay laboratory, maintenance, and mobile equipment. The breakdown of these costs is summarised in Table 21.7.

Table 21.7 – Process Opex

Description	LOM ($ '000)	Cost ($ '000 a)	Cost ($/t)	Total Cost (%)
Manpower	15,920	1,990	0.83	16.07
Power Electrical Power	12,831	1,604	0.67	12.95
Reagents	52,987	6,623	2.76	53.48
Assay Lab	976	12	0.05	0.99
Maintenance and Consumables	7,789	973	0.41	7.86
Mobile Equipment	8,566	1,070	0.45	8.65
Total Opex	**99,069**	**12,384**	**5.16**	**100.00**

Figures may not add due to rounding.

21.2.3.1 Manpower

The labour cost estimate includes 54 shift and office personnel working directly at the crushing area, stacking area, heap leach pad, and ADR facilities. The plant administration includes one (1) superintendent, one (1) plant metallurgist, and one (1) operations metallurgist. The labour rates used for the process labour estimate were provided by DPM. The shift schedule is defined as 7 days a week, 12 hours a day, with 4 employees required for full-time coverage or 2 for day-time coverage.

Table 21.8 summarises the labour breakdown for the process plant.

Table 21.8 – Process Plant Manpower

Description	Number
Administration	4
Operations	45
Maintenance	12
Assay Laboratory	11
Total	**72**

21.2.3.2 Power Costs

Electrical power is required to operating equipment in the processing plant such as conveyors, crushers, pumps, furnaces, services, etc. The unit electricity rate of $0.0878/kWh was provided by DPM. The total annual cost for the process electrical power is estimated as $1.65 M for 2.5 Mt processed.

21.2.3.3 Reagents

The reagent cost considers reagent consumption required for heap leaching and ADR facilities (including natural gas). Reagent consumption is specified in the Process Design Criteria. The reagent supply prices were based on quotations from reputable suppliers, and include sodium cyanide, quicklime, anti-scalant, cement, activated carbon, hydrochloric acid, sodium hydroxide, and gold room reagents. Natural gas consumption was estimated using a rate of $0.041/L for Serbia.

21.2.3.4 Maintenance

Maintenance includes major wear parts, heap leach pad construction, and general maintenance. Wear rates for liner and screens replacement frequencies were estimated based on vendor-supplied data and DRA experience. A general maintenance allowance was estimated as 3% of direct mechanical equipment costs for the plant. This allowance was included to address any miscellaneous consumables and wear items within the plant.

21.2.3.5 Assay Laboratory

Assay laboratory includes analysis costs for process and environmental testing, as well as allowance for equipment and supplies. Assay types, quantities, and percent of tests performed at internal and external laboratories were determined based on DRA experience.

21.2.3.6 Mobile Equipment

Mobile equipment includes fuel and maintenance costs for the mobile equipment required for the process facilities. The equipment includes light duty trucks, a bobcat, a front-end loader, a forklift, a flat bed truck with a crane, and other small equipment.

21.2.4 Water Treatment Opex

DRA derived a conceptual water treatment plant with associated Opex as shown in Table 21.9.

Table 21.9 – Water Treatment Plant Opex

Description	Consumption	Cost ($ '000/a)	Cost ($/m³ water treated)
Power	5,216,257 kWh/a	**$458**	***$0.36***
Reagents		***$153***	***$0.12***
Sulphuric Acid	4,032 g/t	$26	
Peroxide	900 g/t	$42	
Lime	157 g/t	$51	
Flocculant	5 g/t	$33	
TOTAL		**$611**	**$0.49**

Figures may not add due to rounding.

21.2.5 General & Administration Opex

The G&A cost for the PFS stage was compiled using internal DPM data and external benchmarked projects. The G&A costs are depicted in Table 21.10.

Table 21.10 – G&A Opex Cost Breakdown

Item	Annual Cost ($ '000 /a)	Unit Cost ($/tonne feed 2.5 Mtpa)
Manpower	2,074	0.82
General Services	1,442	0.59
Site Services	457	0.18
Total G&A	**3,973**	**1.59**

22 ECONOMIC ANALYSIS

22.1 Basis of Evaluation

DRA has prepared the assessment of the Project on the basis of a discounted cash-flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback, and other measures of the Project's economic viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the PFS was to determine the potential economic viability of developing the Project, which consists of three open-pit deposits and heap leach processing to produce doré.

The cash flow arising from the PFS has been forecast, enabling a computation of the NPV to be made. The sensitivity of the NPV and IRR to changes in the base case assumptions is also examined.

22.2 Macro Economic Assumptions

22.2.1 Metal Prices

Gold sales provide Project revenues. No other product is being considered as part of the financial analysis.

The Project has been evaluated using a constant metal price of $1,500/oz Au as described in Section 19.

Gold price sensitives are tabulated at $1,250, $1,500, $1,750, and $2,000/oz.

22.2.2 Expenses and Net Returns

Section 19 describes the relevant markets associated with the sale of gold. The adopted sales terms are repeated in Table 22.1.

Table 22.1 – Doré Sales Terms for Financial Analysis

Category	Terms
Gold Refining Charge	$0.5/troy ounce fine gold credited (average)
Gold return	100% of assayed content
Shipping Cost	$0.70/troy ounce fine gold credited (average)
Handling Charge	$500 per 100kg lot shipment from site

22.2.3 Royalties

The Government of Serbia imposes a 5% royalty on the Net Sales Revenue.

No formal local resident royalty or profit share agreements are anticipated for this project.

22.2.4 TAXES

The corporate tax rate in Serbia is 15%.

The Serbian government provides tax incentives related to new country investments with the following covenants:

- Direct investment is more than 1 billion Serbian Dinars (Approximately $10m).
- Create new employee positions for more than 100 Serbian citizens.

Corporate tax reduction incentives ranging from 90%-100% can be achieved for a period of 10 years. A 91% tax incentive has been adopted for this PFS, remaining consistent with previous study work performed (CSA Global, 2019).

Serbian VAT is imposed at a rate of 20%. Although this tax is not applicable to corporations, the cashflow consideration must be considered. VAT will be applied as follows:

- Gold sales revenue is exempt from VAT receipts.
- All Opex costs except salaries will attract VAT payment and refunds. It is assumed that VAT will be reconciled on a monthly basis and have not been specifically included in the Financial Model.

Tax associated with losses may be carried forward for a period of 5 years.

22.2.5 DEPRECIATION

Depreciation using a straight-line balance method at 10% per annum will be applied to mining fleet, mechanical equipment and mobile plant.

No depreciation on property assets will be applied.

Depreciation is included to determine Gross and Net Profits indications and has no bearing on the free cashflow NPV calculations.

22.3 Base Case Cash Flow Analysis and Economic Results

22.3.1 RESULTS

A summary of the Cashflow output is depicted in Table 22.2.

Table 22.2 – Economic Analysis Results Summary

Description	Unit	Value
Gold Price	$/oz	1,500
Production Profile		
Total tonnes of ore mined and processed	kt	19,214
Total tonnes waste mined	kt	48,345
Strip ratio	waste/ore	2.52
Head grade	g/t Au	1.07
Peak tonnes per day ore mined	t	7,610
Weighted average gold recovery – Oxide Ore	%	84.95%
Weighted Average gold recovery – Transitional Ore	%	71.49%
Total gold ounces recovered to doré	oz	547,034
Average annual gold production to doré	oz	68,379
peak annual gold production to doré	oz	87,166
Mine life	year	8
Unit Operating Costs		
LOM average cash cost	$/oz Au	606
AISC(1)	$/oz Au	693
Project Economics		
Royalties	%	5.0
Average annual EBITDA	$ M	64
Pre-tax NPV 5% / After-tax NPV 5%	$ M	139 / 135
Pre-tax IRR / After-tax IRR	%	20.9 / 20.6
Undiscounted operating pre-tax cash flow/after-tax cash flow	$ M	230 / 226
After-tax payback period	Annum	3

(1) All In Sustaining Costs

This report contains certain non-GAAP (Generally Accepted Accounting Principles) measures such as cash cost and ASIC. All-in sustaining cost per ounce for the Project represents mining, processing, site general and administrative costs (“G&A”), water treatment costs, royalties, treatment and refining charges and sustaining capital, divided by payable gold ounces, and excludes corporate G&A. Such measures have non-standardized meaning under GAAP and may not be comparable to similar measures used by other issuers. See DPM’s latest Management’s Discussion and Analysis available on DPM’s website (www.dundeeprecious.com) and on SEDAR (www.sedar.com) for additional general information about non-GAAP measures reported by DPM.

22.3.2 CASHFLOW

The scheduled cashflow is depicted in Table 22.3. The peak funding requirement is anticipated in year Y-1 prior to commencement of commercial gold production. The cashflow is followed by NPV and IRR calculations based in the year Y-2, upon commencement of major capital expenditure.

Table 22.3 – Annual Cash Flow Schedule

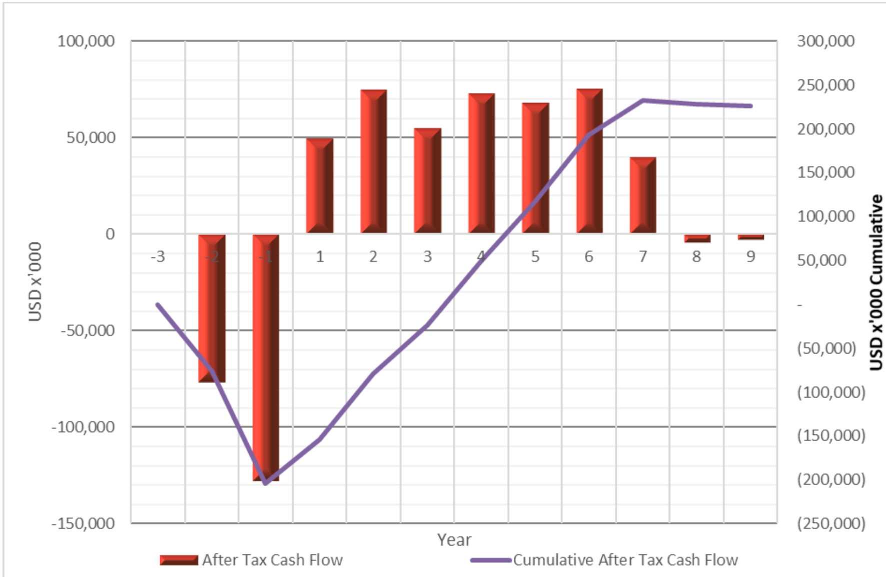
		Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
MINE PRODUCTION													
Ore Tonnage from Pit	kt	19,214		74	2,739	3,486	2,375	2,537	3,267	3,102	1,634		
Grade	g/t	1.07		0.9	1.2	1.0	0.8	1.4	1.1	1.0	1.0		
Gold	oz	661,682		2,143	103,239	111,636	61,639	111,425	117,922	101,005	52,672		
OXIDE ORE PRODUCTION													
Tonnage to Leach Pad	kt	15,836			2,000	2,500	2,272	2,374	2,244	2,028	1,198	1,220	
Grade	g/t	1.08			1.23	1.20	0.96	1.32	1.23	1.03	0.96	0.24	
Gold Stacked	oz	548,986			79,260	96,581	69,897	100,420	88,996	67,324	37,133	9,375	
Recovery	%	84.9%			84.3%	84.2%	84.9%	85.2%	85.0%	85.5%	86.1%	84.9%	
Oxide Ore Gold recovered	**oz**	**466,206**			**66,815**	**81,295**	**59,329**	**85,599**	**75,655**	**57,565**	**31,984**	**7,964**	
TRANSITIONAL ORE PRODUCTION													
Tonnage to Leach Pad	kt	3,378					228	126	256	472	1,302	994	
Grade	g/t	1.04					1.59	0.55	1.66	2.37	0.82	0.47	
Gold Stacked	oz	112,696					11,645	2,208	13,636	35,945	34,360	14,903	
Recovery	%	71.7%					71.4%	71.0%	71.2%	71.7%	72.2%	71.5%	
Trans Ore Gold recovered	**oz**	**80,827**					**8,314**	**1,567**	**9,707**	**25,765**	**24,819**	**10,655**	
TOTAL ORE PRODUCTION													
Tonnage to Leach Pad	kt	19,214			2,000	2,500	2,500	2,500	2,500	2,500	2,500	2,214	
Grade	g/t	1.07			1.23	1.20	1.01	1.28	1.28	1.28	0.89	0.34	
Gold	oz	661,682			79,260	96,581	81,541	102,628	102,632	103,269	71,493	24,278	
Recovery	%	82.67%			84%	84%	83%	85%	83%	81%	79%	77%	
Total Gold recovered	**oz**	**547,033**			**66,815**	**81,295**	**67,642**	**87,166**	**85,362**	**83,330**	**56,804**	**18,618**	
CASHFLOW													
Gold price	$/oz	**1500**											
REVENUE	000 $	**820,550**			100,223	121,943	101,464	130,749	128,043	124,995	85,206	27,928	
OPEX	**000 $**	**331,410**		**518**	**41,986**	**45,855**	**44,854**	**46,486**	**46,726**	**45,957**	**34,595**	**21,929**	**2,505**
	$/t	**17.2**			**21.0**	**18.3**	**17.9**	**18.6**	**18.7**	**18.4**	**13.8**	**9.9**	
Mining	000 $	**151,617**			**21,443**	**22,626**	**22,579**	**22,760**	**23,088**	**22,390**	**12,731**	**4,000**	
	$/t ore	7.89			7.8	6.5	9.5	9.0	7.1	7.2	7.8		
Processing	000 $	**99,069**			**10,950**	**12,529**	**12,618**	**12,578**	**12,629**	**12,713**	**13,036**	**12,016**	
	$/t ore	5.16			5.48	5.01	5.05	5.03	5.05	5.09	5.21	5.43	
G&A	000 $	**33,774**			**3,973**	**3,973**	**3,973**	**3,973**	**3,973**	**3,973**	**3,973**	**3,973**	**1,987**
	$/t	1.76			1.99	1.59	1.59	1.59	1.59	1.59	1.59	1.79	
Water Treatment	000 $	**5,179**		**518**	**518**	**518**	**518**	**518**	**518**	**518**	**518**	**518**	**518**
	$/t	0.27			0.26	0.21	0.21	0.21	0.21	0.21	0.21	0.23	
Dore Treatment & Refining Cost	000 $	**743**			91	111	92	119	116	113	77	25	
Royalty (5% NSR to Serbian Govt.)	000 $	**41,028**			5,011	6,097	5,073	6,537	6,402	6,250	4,260	1,396	
CAPEX & Sustaining	**000 $**	**258,744**	**76,458**	**126,714**	**7,845**	**824**	**1,139**	**10,360**	**12,116**	**2,593**	**10,348**	**10,348**	
Mining Equipment	000 $	**50,191**	13,130	12,622		824	1,139	10,360	12,116				
Pre-Stripping	000 $	**8,643**		**8,643**									
Process Plant	000 $	**94,200**	32,382	59,488	2,329								
Indirect costs	000 $	**61,102**	21,351	35,301	4,450								
Contingency	000 $	**21,320**	9,594	10,660	1,066					Closure	Closure	Closure	
Closure (incl contingency)	000 $	**23,288**								2,593	10,348	10,348	
EBITDA	000 $	**489,140**	-	(518)	58,237	76,088	56,609	84,263	81,317	79,039	50,611	5,999	(2,505)
Depreciation	000 $	**175,067**	3,094	7,691	20,237	20,237	20,237	20,434	20,434	20,434	22,681	19,587	
EBIT Gross Profit	000 $	**314,074**	(3,094)	(8,208)	38,000	55,851	36,372	63,830	60,883	58,605	27,929	(13,588)	(2,505)
Loss Carried Forward - Applied	000 $	**11,303**				27,395						-13,588	-2,505
Profit tax	000 $	**4,240**			513	384	491	862	822	791	377		
Net profit	000 $	**309,834**	(3,094)	(8,208)	37,487	55,467	35,881	62,968	60,061	57,814	27,552	(13,588)	(2,505)
Before Tax Cash Flow	000 $	**230,396**	(76,458)	(127,232)	50,392	75,264	55,470	73,904	69,201	76,446	40,263	(4,349)	(2,505)
Cumulative Before Tax Cash Flow	000 $	**230,396**	(76,458)	(203,690)	(153,298)	(78,034)	(22,564)	51,340	120,541	196,986	237,249	232,901	230,396
After Tax Cash Flow	000 $	**226,156**	(76,458)	(127,232)	49,879	74,880	54,979	73,042	68,379	75,654	39,886	(4,349)	(2,505)
Cumulative After-Tax Cash Flow	000 $	**226,156**	(76,458)	(203,690)	(153,811)	(78,931)	(23,952)	49,090	117,469	193,124	233,009	228,661	226,156

Table 22.4 – Sensitivity Summary

Pre-Tax Financial ratios		Value
IRR		20.90%
NPV	Undiscounted	230,396,000
NPV	5.0%	138,582,000
NPV	7.5%	104,559,000
NPV	10.0%	76,463,000
NPV	15.0%	33,842,000
Payback period, from start of production		2
After Tax Financial ratios		
IRR		20.60%
NPV	0.0%	226,156,000
NPV	5.0%	135,432,000
NPV	7.5%	101,823,000
NPV	10.0%	74,073,000
NPV	15.0%	27,820,000
Payback period, from start of production		3

An after tax cashflow graph is depicted in Figure 22.1.

Figure 22.1 – After Tax Cashflow Graph



22.4 Sensitivity Analysis

The sensitivity of Project returns in key value drivers was tested over a range of ±25% for metal prices, operating costs, and capital expenditure. For this project, sensitivity to grade is the same as for metal price. The sensitivity results are presented in Figure 22.2 and Table 22.5. A gold price/NPV comparison table is presented in Table 22.6.

Figure 22.2 – After-Tax IRR Sensitivities

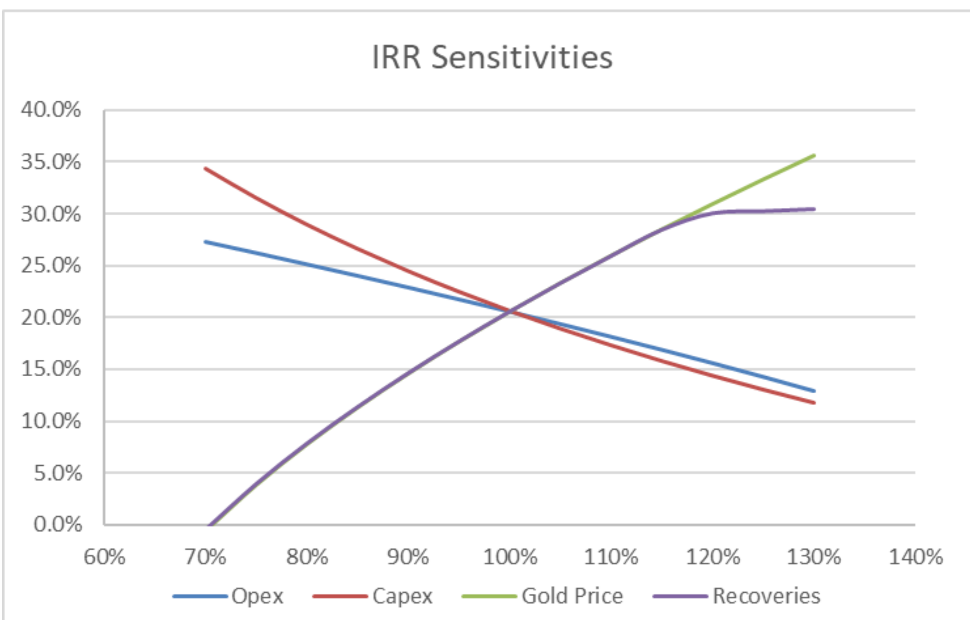


Table 22.5 – Cashflow Sensitivity Table

After-Tax NPV @ 7.5%				
Change	**Capex**	**Opex**	**Gold Price**	**Recoveries**
-30%	165,039,000	164,632,000	-47,916,000	-47,774,000
-25%	154,503,000	154,164,000	-22,953,000	-22,834,000
-20%	143,967,000	143,696,000	2,006,000	2,101,000
-15%	133,431,000	133,227,000	26,960,000	27,032,000
-10%	122,895,000	122,759,000	51,914,000	51,962,000
-5%	112,359,000	112,291,000	76,868,000	76,892,000
0%	101,823,000	101,823,000	101,823,000	101,823,000
5%	91,286,000	91,354,000	126,777,000	126,753,000
10%	80,750,000	80,886,000	151,731,000	151,683,000
15%	70,214,000	70,418,000	176,685,000	176,613,000
20%	59,678,000	59,949,000	201,639,000	192,187,000
25%	49,142,000	49,481,000	226,593,000	195,360,000
30%	38,606,000	39,013,000	251,547,000	198,532,000

Table 22.6 – Gold Price After Tax NPV Sensitivity Table

Description	$1,250/oz Totals ($ M)	$1,500/oz Totals ($ M)	$1,750/oz Totals ($ M)	$2,000/oz Totals ($ M)
IRR	10.2%	20.6%	29.4%	37.1%
After-Tax Cash Flow				
Undiscounted	98	226	354	482
NPV at 5%	40	135	231	326
NPV at 7.5%	19	102	185	268
NPV at 10%	1	74	147	219
NPV at 15%	-21	28	77	126

23 ADJACENT PROPERTIES

This section is not applicable as there are no relevant Adjacent Properties to describe.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Schedule

The PFS Project execution schedule is integrated and includes EPCM activities on a durational basis. The pre-feasibility study team members from DRA, SLR, ERM, and DPM provided input to the Project schedule. This section describes the basis of the Project schedule, discussing the overall project timelines, construction timing, objectives and strategies, contractor mobilizations, dependencies, and constraints.

The schedule is based on the following source documents:

- DRA – engineering schedule; process plant and site infrastructure construction schedule; procurement and contracting plans.
- DRA – pre-production mine development and pit development schedules.
- SLR – heap leach pad and pond construction schedule.
- ERM – permitting, Social, Environmental.

The schedule is organised by discipline, activity/package code, and area for Engineering, Procurement, and Contracts, by WBS.

The Project critical path runs through the procurement, fabrication, delivery, and installation activities of the modular crushing and screen package. The procurement process starts three months after EPCM award. To expedite the crushing and screening package, a phased delivery approach has been incorporated into the schedule.

The Project construction start date is assumed to be immediately upon award of the construction related contracts with EP being significantly progressed. Construction activities related to the initial phase are scheduled for 24 consecutive months from this date. Ore production from the Bigar Hill pit will occur 21 months after Project sanction. Production at Korkan West and Korkan are scheduled to occur later on during the operational phase.

24.1.1 Execution Strategy

24.1.1.1 Project Type

The Project is considered a greenfield site with minimal infrastructure and service connections available.

A single EP+CM contractor (or Owner managed CM) will be responsible for the delivery of the Project. The Project team will consist of team members from the EPCM Contractor, heap leach specialist and selected members of the DPM team. DPM may self-perform the mining and construction activities. All equipment Purchase Orders (PO) and construction contracts will be established between the package proponent and DPM directly.

24.1.1.2 Logistics and Sequencing

Process facility construction is intended to commence with the crushing and screening area, before advancing to the absorption-desorption-recovery (ADR). Concrete foundations, steel, and cladding (where applicable) will be installed first, followed by mechanical equipment, piping, electrical, and instrumentation.

Mine development will commence with Bigar Hill Pit cut and associated haul road, clearing and grubbing. Suitable stripped waste material from Bigar Hill will be used for the construction of the necessary haul roads to access the Korkan West pit. The haul road designs will be such that owner operated mining can assist with their construction to optimise capital input.

Heap leach and waste facility construction will commence early to ensure completion prior to process plant readiness. The placement of ore is required to effectively test the ADR facility. Heap leach and waste facilities require the construction of contact water containment embankments and other related earthworks that require fill material. The early establishment of the mining fleet is anticipated to commence with pre-stripping activities to supply suitable waste material for heap and waste facility construction. A balance of material exercise was undertaken to ensure that required pre-stripping on Bigar Hill and Korkan West are sequenced with construction requirements. Pre-stripping is planned to commence 12 months prior to plant commissioning.

24.1.2 KEY MILESTONES

24.1.2.1 Summary

A duration summary of key project milestones is depicted in Table 24.1. A preliminary project schedule is depicted in Figure 24.1.

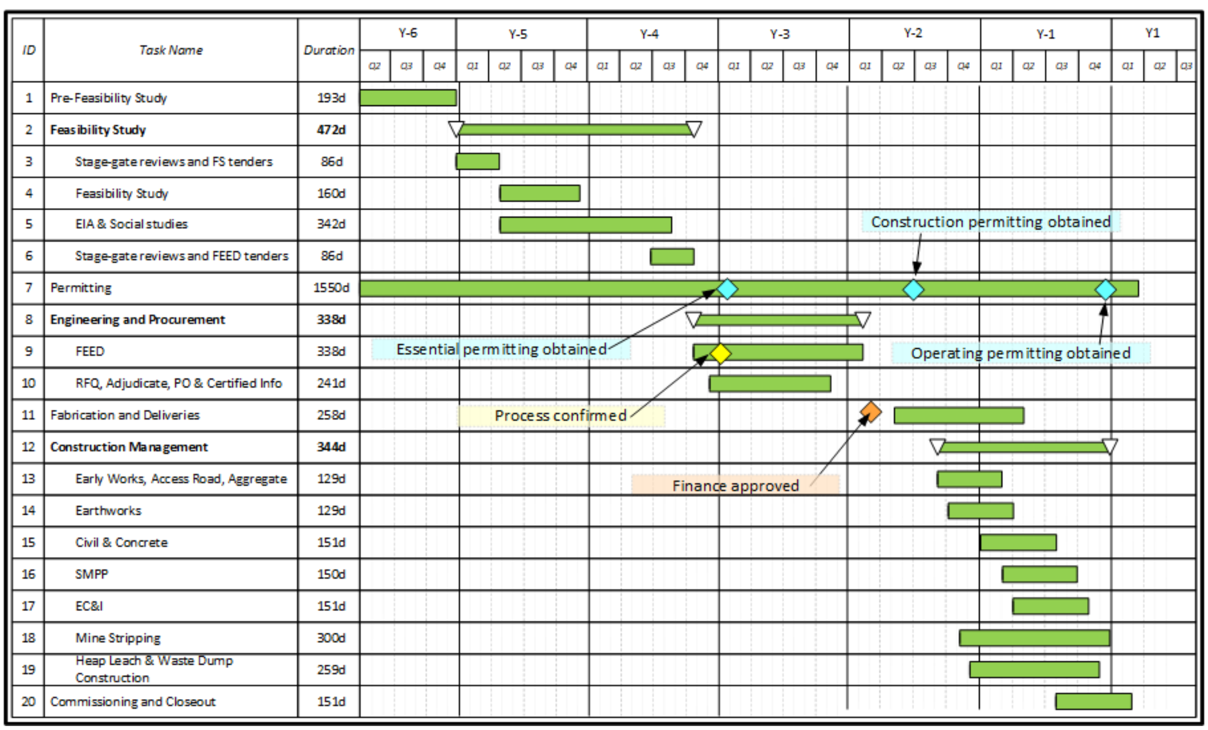
24.1.2.2 Permitting

The progression of the Project into execution requires the approval of certain permits as highlighted in Section 20. A detailed permitting strategy and schedule has been devised using mean timeline durations. Delays in permitting approval will delay to commencement of the execution process. Following initial construction approvals for the project, further operations related permit applications will continue past the completion of construction.

Table 24.1 – Key Project Milestones

Milestone	Months After Partial Project Approval
Partial Project Approval to proceed (FS Commencement Approval)	0
Award of Detailed Engineering	17
Critical Permits Received	20
Engineering 30% Complete	23
Engineering 90% Complete	33
Long-Lead POs and Contracts Awarded	34
Start Construction – (CM or DPM)	36
Start Construction – Process Facilities	40
Start Construction – Heap Leach (Phase 1)	44
Connection of Electrical	41
C1 Complete	47
C2 Complete	48
First Production	50

Figure 24.1 – Overall Project Schedule



24.1.3 CRITICAL PATH

The completion of the Engineering & Procurement designs followed by purchasing and installation of the ADR processing plant represents the longest durational path in schedule through to completion of construction. The preparation of the heap leach pad, crushing of first ore in preparation of the ADR plant commissioning and operations is the next critical path items. The early establishment and availability of the mining fleet to provide the necessary bulk earthworks material is also critical.

The critical path includes the following activities:

- Obtaining essential permits
- Project sanction and availably of finance;
- Purchase and delivery of mining fleet;
- Freeze design criteria;
- Complete EP design and procurement;
- Procurement Package – ADR Plant;
- SMPP installation – Processing Facility;
- Heap Leach Facility overliner material placement;
- Commissioning.

This critical path exposes the associated schedule risk. During the EP phase, schedule risk will again be assessed, and a management plan will be implemented to mitigate it as far as is practical. Mitigation strategies already implemented include prioritisation of the procurement package, sound contractual terms with fabricators, and partial delivery modules.

24.1.4 NEAR CRITICAL PATH

Risks associated with near critical path items require the same level of planning and scrutiny to manage delay risks.

The Project Execution Schedule also contains a series of near-critical activities (less than 10 days float) involving the design, supply/fabrication, and delivery of the:

- Crushing Equipment (Primary and Secondary);
- Main Substation;
- Power Transformers.

The above series of activities is less than a week off the critical path, and this highlights the importance of proactive management of the various items and early order placement or fabrication slot booking. Power transformer and substation will be critical for the commissioning of the plant and may limit opportunity for early progression of commissioning. Detailed strategy around energizing must be explored as part of the next project phase to ensure sufficient buffering to realize early equipment commissioning.

24.1.4.1 Schedule Contingency and Risk

No schedule contingency has been determined or allowed for in the study execution schedule itself. The schedule has been positioned to allow for a one (1) month late start and an additional month late completion, allowing some variation within the overall DPM Project schedule.

24.2 Project Risk Review

A risk review process was undertaken as part of the PFS update following the technical design definition of the process.

Key environmental and social risks are discussed in Section 20 of the Report. Further key project risks and recommended mitigations are summarised in Table 24.2. Recommendations are made to address remaining uncertainty/risk in the next phase of the project.

Table 24.2 - Project Risk Identification and Recommendations

Risk	Impact	Recommendations for next project phases
Site Closure Monitoring	Site monitoring and obligations post closure - unclear legislative requirements or guidelines - long term obligations.	Perform full EIA for baseline monitoring data to be captured. Engage formally with regulators utilising prepared closure planning.
Geotechnical and Topographical Design Risk	Significant quantities of bulk earthworks and mine waste stripping are required for the project. The current project economics are dependant on accuracy of topographical mapping to define the bulk earthworks quantities. Accuracy will be key to ensure accurate costs are included in the final financial analysis.	A geotechnical drilling program associated with current infrastructure locations have been drafted. Final location layouts
Process Recovery Risk – Clay Content	The role of clay within the ore body can impact the effectiveness of heap leach facility. An agglomeration circuit has been allowed for with an estimated reagent addition quantity defined. The extent of clay content requires better definition to ensure the economics accurately reflect anticipated clay agglomeration.	Mapping of clay content from borehole assay data and sequencing of clay extraction is required to model periods of high clay content to ensure it can be addressed operationally.
Waste Rock Characterisation and Treatment	Acid Generating potential and heavy metal leaching characterisation is currently not sufficiently defined to determine waste facility design requirements (lining, treatment, co-disposal).	Complete long-term waste characterisation testwork prior to FS design phase. Complete regulatory compliance framework for various waste types to ensure efficient waste management design and operation on site.
Seismic Classification of Facilities	A conservative application of dam design guidelines have been incorporated in current designs in the absence of site-specific seismic studies.	Conduct site specific seismic studies to allow accurate facility classification and design parameters for the FS design phase.

Risk	Impact	Recommendations for next project phases
Construction/Execution Risk Local Contractors	Initial Serbian market enquiries produced mixed responses. Supplier vetting and development may be required to mitigate execution related risk.	Perform formal supplier vetting and tendering process during FS, including document and regulatory compliance assessment.
Global Pandemic Management	The impact of COVID-19 during the PFS was prevalent in the cancelling of site visits and vetting of local suppliers.	Ensure a travel plan is devised to allow FS phase personnel to travel to Serbia to ensure better site and supplier definition is obtained.

25 INTERPRETATION AND CONCLUSIONS

25.1 Risk Evaluation

Risk and opportunity management is an important element of the Project's operational strategy and implementation. It is used to avoid losses, anticipate problems and realize gains or benefits.

External risks are elements such as the political situation in the Project area, the price of minerals, the exchange rate and government legislation. These external risks are generally applicable to all projects. A negative variation of these elements compared to the assumptions made in the economic model would affect the profitability of the Project and the MRE. These external risks are to some extent independent of the Project and are much more difficult to predict and mitigate.

As with all mining projects, there are technical risks that could affect the feasibility and economic results of the Project. There are several risks and uncertainties identifiable with all projects and generally cover the aspects of understanding mineralisation controls, MRE, metallurgical testwork and process design, mine planning and Mineral Reserve estimates, infrastructure development, acquisition of surface rights, cost estimation, financial analysis, environmental studies and permit authorisation. These risks are common to most mining projects, many of which can be mitigated by adequate engineering studies, design, planning and management.

25.2 Mineral Resource Estimates

A relative risk to the MRE is related to the UC/LUC estimate approach used which is quite new in the mining sector. UC is a non-linear estimator which was developed to resolve common problems found with kriging. It was developed by Matheron in 1974, refined by Riviorard in 1994 and the LUC approach was added in 2006 by Abzalov. It is a recoverable resource estimation approach; in that it estimates the portion of the deposit that is technically recoverable when one applies a cut-off to selective mining units (SMU's) to be mined at production stage. However, it is unable to predict a spatial location of the recoverable mineralisation within the panel although the localisation step does attempt to solve this. The risk related to the appropriateness of the selected estimation approach, along with the use of small SMU size 5×5×5 m of can be mitigated by running a trade-off study where the MRE is conducted using different block sizes and comparing the results to the economics of the project.

Another route to mitigate this risk is the addition of infill drilling in the resource pit area to reach an average drill hole spacing of 20m and run the resource estimate directly with a linear estimator approach (OK) at the Feasibility stage of development of the Project.

Risk within the current MRE can also be attributed to the weathering domains. The transtitional domain in particular, built using a range of TS%, AuCN/Au and visual logging values, inherently contains varaibiltiy due to the grouping of different datasets. This varaibility will likely influence local metallurgical recoveries within seperate regions of the Timok Gold Project and should be investigaged further in preperation for the Feasability study.

Furthermore, limited close spaced data (sub 40m x 40m spacing) has been collected to date. Infill drilling may result in a change to the geometeries and volumes of mineralisation and weathering domains.

25.3 Mineral Processing and Metallurgy

25.3.1 HEAP LEACH PLANT

Testwork on the Timok deposit is considered representative of the ore to be processed according to the PFS mine schedule. Testing was conducted on ores distributed throughout the planned pit shells, with respect to both depth and spatial coverage.

The tests provide suitable data coverage for the oxide and transitional geological rock types, namely from Bigar Hill, Korkan, and oxide ore from Korkan West deposit.

The sulphide mineralization type is not included in the mine plan due to the poor leach performance. Other methods for gold recovery were considered for the sulphide Mineral Resources including flotation for production of concentrate and oxidation of the concentrate through the Albion process followed by cyanidation of the oxidation product and flotation tails, CIL adsorption of the gold, and production of the doré bars as a final product.

The majority of the ore samples tested have head grades which are considered representative of the Mineral Reserves in the PFS mine schedule, and within an acceptable grade range of the design criteria.

The column leach test results were used to determine the ore crush size, metallurgical gold recoveries and reagent consumption levels.

There is an opportunity in evaluation if the coarser crush size (P_{100} of 38 mm) will be acceptable for all types of ore with regards to the leach recovery. The coarser crush size promises potential benefits in simplification of the crushing area flowsheet and reduction of the operating costs.

Insufficient CWi and Ai tests were conducted on the oxide and transitional ore samples to design the crushing plant flowsheet and size the relevant plant and equipment, in addition to predicting the operational cost associated with crushing-plant wear parts. The PFS crushing circuit design is presently based on DRA design and operational experience. Additional testing of ore composites and variability samples is required to develop a sufficient data set.

Based on industry benchmarking, DRA experience, and outcome from the process design reviews the discount of 2% was applied to the lab recovery for the Oxide ore, and 5 % for the lab results for the Transitional ore.

Expected discounted recovery numbers for Oxide ore lays in the range of 83.4-85.1%, and 68.8 - 81.0% for Transitional ore type.

Currently, no information on the oxide and transitional ore regarding 'preg robbing' or 'preg borrowing' behaviour is available. Providing for the expected clay loaded areas within the deposit, this work should be budgeted for the feasibility study (cyanide shake tests).

Present knowledge of the ore body suggests that agglomeration will be required intermittently on some clay loaded ore domains. It should be noted that an agglomeration circuit has been included on an as required basis, but a greater understanding on the amount and locations of clay zones in the deposit is required.

The knowledge of the clay speciation presently is limited to the clay types within the deposit and no clay speciation data is available. No load permeability tests nor agglomeration tests results required for definition of agglomerates strengths and the required dosage of lime and cement are available to date. This work should be planned for the feasibility study testwork programme.

With regards to the ore stacking, all types of ore can be stacked together based on the current results of the leach tests. Agglomerated ore should be physically separated from non-agglomerated ore on the leach pad where practical.

A review of the column leach test sodium cyanide consumption, at the Timok design solution application volume, gave a cyanide consumption level of 0.40 kg/t. A laboratory-to-field factor of 30% was applied to the column leach test results, which gave an expected field sodium cyanide consumption of 0.12 kg/t.

To ensure proper pH control, the expected quick lime addition maximum is 0.32 kg/t (equivalent to 0.30 kg/t of pure calcium oxide).

Cement addition will be required for the agglomeration circuit only, and a consumption rate is presently understood at 10 kg/t based on DRA's experience due to the absence of the testwork data. The cement consumption rate should be revised during the next stage of the Project upon receipt of the testwork results and the necessity to agglomerate.

Preparation of a geometallurgical model of the Timok deposit is recommended to optimise reagent consumption and gold recovery for the different ore types.

Irrigation rates, durations, and application flux rates (m^3 of solution / tonne of ore) were determined from the testwork and DRA experience in leach kinetics modelling and heap leach plant design.

Based on the results of the leach kinetics modelling and leach cycle costs analysis a single-stage irrigation schedule with the leach cycle of 90 days for both oxide and transitional ore presents a suitable process setup.

25.3.2 SULPHIDE MATERIAL

Based on the current understanding of the sulphide resource, the metallurgical responses observed in the testwork, and current metal prices, an economic processing route for the sulphide material has yet to be determined and the further work is required. It is recommended that work continue to better understand the sulphide resource and investigate the economics of alternate processing options.

25.4 Mining Method

This Report for the Timok Mineral Reserve is based on an 8-year life of mine open pit which includes 19.2 Mt of Probable Mineral Reserves at an average grade of 1.07 g/t Au with an average stripping ratio of 2.5:1. The ore material is contained within three (3) areas (Bigar Hill, Korkan and Korkan West). The mine will operate year-round, seven (7) days per week, 24 hours per day (two [2] twelve [12] hour shifts). The total Mineral Reserves are composed of 82% of oxide ore and 18% of transitional ore.

Over the life of mine, an average of 2.7 Mtpa of ore will be mined from the open pits, with 2.5 Mtpa hauled directly to the crusher/heap leach pad and 0.2 M tonnes per year to stockpiles for future reclaim to the crusher in Year 8.

The mine will be operated with a fleet of 60 T rigid haulage trucks and 6.8 m^3 excavators.

25.5 Project Infrastructure

The infrastructure section described the geographical location and configuration of the Project elements that will be necessary to extract the minerals from the Timok ore body.

The design and impact of the various ore body pits and associated waste rock facilities was discussed along with water containment and treatment assumptions applied. A partially lined Bigar Hill facility has been included in the PFS capital budget to allow for containment and treatment of ARD runoff. Alternative storage of potentially acid generating mined sulphide ore was discussed.

The use of cyanide in the heap leach facility is mentioned along with containment and treatment measures that comply with the ICMC guidelines for DPM to remain a compliant signatory.

Project support infrastructure such as roads, administration, security and effluent treatment is addressed along with budget allocations within the capital expenditure for the construction of the mentioned infrastructure.

The site water balance and treatment requirements are outlined along with treatment and monitoring requirements to minimise risk associated with cyanide contamination and ecological impact. The water management system has been conservatively designed to contain storm related events and minimise the risk of cyanide contamination during abnormal events.

The PFS level of design is well advanced but is still based on experienced assumptions and regional data until locally conducted test work results can be made available. Geochemical, hydrogeological, and geotechnical investigations are underway to conclude the suitability of the project element locations and underlying soil and ground water conditions.

25.6 Environmental Studies, Permitting and Social or Community Impact

The Project has undertaken permitting efforts, environmental studies and community engagement since 2007. Environmental and social baseline work commenced in 2012 and continues to the present day, with a more extensive baseline collection programme planned.

Environmental and social factors have been considered in the assessment of the various alternatives, trade-off and options studies in this PFS design. Key environmental and social mitigation features include:

- Full design compliance with the International Cyanide Management Code, to which DPM is a signatory.
- Use of liners where appropriate to manage potential acid and metal leaching risks to groundwater and surface water.
- Siting of project infrastructure away from karstic or potentially karstic limestone, to the extent possible.
- Active water management with contact / non-contact water separation, pit dewatering separation, testing and treatment to the required permit limits prior to discharge and managed discharge flow rates.
- Active management of health and safety including taking account of the requirements of the International Cyanide Management Code.
- Closure planning already underway and informing PFS design options.

Overall, the DPM approach presents several advantages compared to other approaches:

- Relatively low footprint, comprising the phased development of pits and waste rock dumps, a small plant and heap leach area; all of which will be rehabilitated at the end of the mine life.
- Low energy demand.
- Low water discharge.

25.6.1 Key Risks

Key environmental and social risks are similar to those associated with other gold mining projects and as a result measures are required to safeguard rivers, groundwater, biodiversity, the local community and local heritage and mitigate permanent effects.

During operations, rivers will be affected by dewatering, diversions and discharges, and permanent infrastructure will overlie several kilometres of upper headwaters reach of the Jagnilo tributaries.

Discharge of hazardous substances into groundwater is prohibited, and so potential risks to surface water and groundwater quality are high. Events that could affect this include sedimentation, seepage of process chemicals including cyanide or hydrocarbons, leachate or runoff from acid generating rock, reduction in baseflow due to pit dewatering, planned discharges, and pollution during extreme flood events. Control of these risks is one of the main design objectives in the Mine Waste and Water Management Plans.

One of the key areas of uncertainty at this point in the PFS design is in the extent to which waste rock are acid generating / metal leaching. Watercourses in this area drain to River Danube, which is internationally protected. The Mine Water Management Plan includes measures to avoid or mitigate these key risks. Further, DPM is a signatory to the International Cyanide Management Code, which provides standards of practice for protection of communities and the environment during cyanide management.

Parts of the site are underlain by karstic geology which poses a risk to the Project as infiltration and spread of pollutants to groundwater is typically greater through karst. The leach pad has been sited away from the karstic limestone zone and the design of the leach pad and other key project infrastructure includes a robust liner system that prevents infiltration to groundwater during operation.

The Project will result in loss of habitat, including several kilometres of riverine habitat, which will have an impact on biodiversity. Abstraction of groundwater to supply the Project and the consequent dewatering may affect community water use, agriculture and habitats. Stone crayfish are present in the Project area and are sensitive to low flows, high water velocities and poor water quality. The Project may affect these and other protected species. The groundwater models and water balance being developed for the Project will help identify potential impacts and further engagement with the local community is needed to fully understand potential impacts to domestic wells. The Bigar Hill Spring, to the south west of Bigar Hill, has been identified by local authorities as a potential source of municipal drinking water. DPM is working with the authorities to find alternative water sources. These potential impacts and associated mitigation measures will be addressed through the ESIA and engagement with regulators.

There are also risks associated with acquiring land, for which careful planning will be required. Acquisition through a voluntary willing-buyer willing-seller approach appears possible. Resorting to involuntary resettlement is permitted under Serbian Law but would have an effect on stakeholder relations.

Seasonal farming, hunting and tourist amenity will also be lost during operations and certain cultural heritage buildings and features will be affected by the Project. This will be addressed through the Livelihood Restoration Plan and ESIA. Closure planning could include the means to address some of these changes after operations cease.

There are risks to the projects associated with permitting delays. Such delays be caused by potential changes to Serbian regulations to align with EU Law, regulator delay, public challenge to the Spatial Plan or EIA and administrative appeals. Similar risks have been experienced by other private sector mining projects permitted in Serbia.

The Project's social licence to operate is of critical importance. DPM has built a good relationship with the local community over many years. The use of cyanide lixiviant, short life of mine and expectations around employment may be key issues amongst stakeholders. The key risks and mitigations for these different aspects that will be more completely covered in the Project ESIA and Livelihood Restoration Plan to be undertaken in the near future, for which several baseline studies are already well advanced. A Project-specific Environmental and Social Management Plan (ESMP) will be developed in line with DPM's overarching environmental management system, to allow these mitigating procedures to be implemented as the Project goes forward to construction.

26 RECOMMENDATIONS

26.1 Geology and Mineral Resources

The QA/QC procedures are comprehensive and are suitable to monitor assay contamination, accuracy and precision. The author of this report recommends the following:

- Although the failure limits used for the standards are adequate, DPM should move to using standard deviations to obtain acceptable limits. Any standard result that varies from the expected value by more than three (3) standard deviations, or any two (2) consecutive standards differing more than two standard deviations would constitute a failure.
- Although Ag is not a significant contributor to the project, for completeness, DPM should include Ag CRM with expected grades ≥ 10 x LDL as well as analyse blanks and field duplicates for Ag.
- DPM should send a suite of bulk density samples for umpire analysis since the previous testwork was unavailable for review. A minimum of 30 umpires by rock type for each prospect would be adequate considering the low levels of variability within the dataset.
- Perfom additional diamond twins within the Bigar Hill Minzone 10, targeting those RC holes that are correctly orientated to test narrow vein mineralisation perpendicular to mineralisation.
- There is no apparent relationship between dimaond drill core or RC recovery and gold content at Timok. However, mineralised sections, beneath the overburden model with < 70% recovery should be considered for re-drill and sections with <50% should be designated automatic re-drill. This was generally the accepted practice during the 2008-2013 RC drilling campaigns however there are a low number of intervals that have not been re-drilled. A reccomendation would be to review these intervals and re-drill them in preperation for future Mineral Resource updates.

For the FS stage, it is recommended to complete infill drilling in the resource pit areas to gain more confidence on the continuity of grade and mineralisation. The objective should be to reach a drill spacing of about 20 to 25 m to confirm mineralisation and grade continuity, targeting a Measured Mineral Resource category in areas dues to be mined early on in the LOM sequence. Linear estimates are applicable at the FS stage and as such, the update of the MRE should be conducted at the SMU size using a linear estimator such as OK.

As the Project progresses, a more comprehensive Geometallurgical program is required to properly constrain weathering domains based on analytical data. Such data should include:

- More systematic AuCN soluble assay data, with focus on sampling mineralised zones within the PFS pit volumes and adjacent areas.
- Systematic Sulphur speciation assay values which will allow more rigorous domaining of weathering domains and input into the generation of recovery curves, which will better honour local variability.

- Review of the 5 m composite length for Au soluble assay data. A shorter composite, perhaps 2 m in length, would provide better resolution and more reliable domain statistics.

26.2 Mining

In next phase of the Project, DRA recommends the development a geometallurgical model to estimate expected process recovery for each block of the block model and use this information to optimise mine schedule and Project NPV.

DRA also recommends looking at the possibility of in-pit waste stockpiling in mined-out portions of the pits to minimise ex-pit waste stockpile size and related cost and minimise environmental footprint.

Owner operated mining fleet vs. contractor mining presents further opportunity for value addition during the next phase of the Project.

26.3 Hydrogeology

Develop a numerical groundwater model of the pits in order to determine the transient inflow rate during both the initial stages of dewatering and as the pit depth is increased. This model would allow modelling of the impact on dewatering rates of annual changes in groundwater level as well as providing an input into any water quality modelling undertaken.

26.4 Mineral Processing and Metallurgy

26.4.1 Heap Leach Plant

The following recommendations for the FS heap leach testwork program were developed as a result of the test work gap analysis conducted by DRA and agreed with the DPM. The programme is based on 2 ore types (oxide and transitional), 4 subsamples of each ore type plus 4 samples of agglomerated ore. Estimated cost of the test work programme expected approximately 650,000:

- Master composite samples needed to be tested to support the PFS; ore blends based upon PFS mine ore schedule. A sampling campaign should be completed to generate sufficient quantity of representative samples for composites and variability testing.
- Very limited information is available on clay distribution within the deposit including tonnage, clay speciation, mineralogy and physical properties. A review of the block model is recommended to quantify the degree of clay content, and its distribution throughout the various deposits. Physical characterisation of the clay in the drill core available and the ones to be produced for the FS is required, including:
  - Clay mineralogy;
  - Clay content;
  - Process properties – particle size distribution, natural moisture, ability to swell.

- Defining the material handling properties is important for the heap, stockpiles, storage bins, conveyors, chutes and hoppers design. It is recommended to engage a recognized material handling testwork contractor to determine ore bulk density (loose/packed), compressibility, angle of repose, chute angle, cohesive strength, wall friction. This work should be carried out on all ore representing feedstock to the process plant.
- Limited comminution results are available for oxide and transitional ore types. Further testing is required during the FS stage including: Bond CWi and Ai tests to estimate crushing power required and crusher liners wear rate. It is also recommended to undertake UCS tests to determine suitability for using toothed roll crushers that can be better able to deal with high clay ores.
- There is limited knowledge on the material load solution percolation properties. No standard load permeability, nor aggregate strength and stability tests results are available to date. To address this, load permeability tests on samples of the crushed and agglomerated ore are required, as well as tests to determine agglomerate strength, and stability testing for the agglomerated ore samples.

  Further ore agglomeration tests are recommended to determine the cement and lime dosages, and the impact of agglomeration with cyanide on Au and Ag leach recovery if required. As above.
- Evaluation of the coarser crush size (P_{100} of 38 mm) impact on the leach recovery is recommended. A coarser crush size has potential benefits in simpler flowsheet for the crushing circuit as well as reduced operating costs.
- Additional leach testing data required on leach performance of the composite and variability samples of crushed and agglomerated ore produced for the FS stage of the Project. Coarse bottle roll and column leach tests recommended on samples of the oxide and transitional ore, and the agglomerates, to determine Au and Ag recoveries, Hg leach behavior and distribution within the leach products, reagents consumptions, carbon loadings, leach kinetics.
- The impact of cold climate on gold and silver leach kinetics has not been determined during the testwork to date. It is recommended to complete the column tests on samples of the crushed (and agglomerated ore) at cold temperatures (in refrigerated containers or similar) to evaluate the effect of cold temperatures on the metals extraction rate.
- Providing for the expected clay loaded areas within the deposit, a series of the cyanide shake tests should be planned and completed during the FS.
- Development of the ore stacking plan is recommended for the FS stage of the Project.
- Further environmental testing is recommended to support the facility permitting process. The scope of the environmentaltesting should be recommended by the Project environmental consultants.

26.4.2 SULPHIDE MATERIAL

It is recommended that work continue to better understand the sulphide resource and investigate the economics of alternate processing options.

A number of recommendations were developed for the further study of the sulphide material:

- Sample representativity and geo-metallurgical sample selection work should be completed by the DPM geologist. The work will confirm FS samples representativity, summarise samples selection procedure and origin and relevancy to the ore tonnage, deposit block model, ore lithology, and RED/OX state.
- A geometallurgical model for the material classified as sulphides should be developed. The main objective should be to determine the amounts and association of the cyanide soluble gold in the sulphide deposit.
- Further metallurgical testwork is required to determine an optimal process treatment route for sulphide material. Estimated cost of the sulphide material testwork programme is expected to be approximately $1,100,000.

26.5 Project Infrastructure

26.5.1 MINING

The mining operations are dependant on the efficient movement of mineral containing materials around the mine site. Further opportunity exists to optimise haul road designs during the next project phase:

- The current haul roads were designed to access pit ramps and balance bulk earthworks during haul road construction. Opportunity exists to further optimise haul road design through the finalisation of mine fleet sizing (mine contracting); and
- Maximizing excavation earthworks during construction that could be undertaken by the early establishment of the mine fleet or mining contractor.

Plant access road routing has been completed during the PFS phase. Further optimisation is possible with an alternative routing of the first 2-kilometer section off the Road 105 intersection.

26.5.2 PROCESS

The heap leach crushing circuit relies on a truck load out bin (150t) facility to move ore to the heap. Finalisation of the mine fleet truck size may reduce the load out bin size, resulting in potential savings.

26.5.3 HEAP LEACH FACILITY

The design of the HLF is based on the limited existing information, reasonable engineering judgement, and a focus on flexibility for the FS design and operations. Additional work recommended for the next level of design will be costed as part of the FS stage of the Project. The following work is recommended to be completed prior to or in conjunction with advancing the HLF to the next level of design:

- Develop and perform a geotechnical investigation under the supervision of the Design Engineer(s) to characterise the foundation conditions, confirm that karst is not present within the general HLF and develop site specific parameters (i.e., density, shear strength, consolidation, etc.) to be used in the design. The geotechnical investigation should include geophysics, boreholes and test pits with representative samples collected for laboratory testing.
- Perform a hydrogeological investigation within the general HLF footprint area under the supervision of the Design Engineer(s) to confirm the depths to groundwater and identify groundwater conditions that could impact the design.
- Perform a field reconnaissance under the supervision of the Design Engineer(s) as part of a geologic hazard assessment to confirm the site conditions within the general HLF footprint area and note any potential geologic hazards.
- Develop and perform geotechnical investigations and laboratory testing in coordination with the Design Engineer(s) to identify, characterise and quantify suitable borrow materials for the HLF for materials specified in the design such as low permeability soil liner, overliner, bedding material, etc. The laboratory testing program should also include the material shear strength and interface shear strength values to be used in the stability analysis.
- Perform additional ore characterisation studies in coordination with the HLF Design Engineer(s) to characterise the ore gradation, percolation, maximum ore height, general properties, and mineralogy. The ore characterisation should also confirm if part or all of the ore is going to be agglomerated, and the HLF design adjusted to account for potential impacts to design elements such as the internal pond, solution recovery, and slope stability.
- Perform a mineralogical and geochemical assessment and characterisation of the leach ore to assess potential environmental issues. The design of the liner requirements should be reviewed to confirm the appropriate containment and risk mitigation strategies.
- The HLF construction staging and material requirements should be reviewed within the context of the mine plan schedule to confirm that suitable earthwork material would be available for construction.
- Perform a site-specific water balance for the HLF and Process plant circuit that considers average monthly climatological conditions, in addition to extreme events (i.e., wet and dry seasons), to validate the HLF Internal Solution and External Event Pond sizing, and include critical stages in the water balance such as start-up and rinsing at closure.
- Collect site specific daily precipitation data (and evaporation data if possible) to compare against the values used in the current design and confirm the design storm event.

- Develop a more detailed liner sub-grade grading plan for the HLF that optimises the cut and fill balance (i.e., regrading).
- Perform a risk assessment and develop a scope of work to address data gaps, reduce uncertainty, and address high level risks.
- Perform a site specific Probabilistic Seismic Hazard Assessment (PSHA) to determine the appropriate PGA to be used in the pseudo-static stability analysis in the FS design.
- Upon receipt of the updated information recommended above, perform trade-off studies to confirm the decisions made in the PFS that include: Internal solution and / or External event ponds for the HLF.
- Review and confirm the HLF closure approach, including such items as an infiltration model to assess the effectiveness of the Enhanced Low Permeability Cover design, potential requirements to monitor the solution levels within the HLF and pump excess solution, and impacts of elevated solution levels on the post mining HLF stability.
- The Project definition and mine waste tonnages considered in SLR's PFS design are based on the tonnages produced by the initial DRA-led pit optimisations performed on the 2018 block model. Subsequent completion of the 2020 Mineral Resource Estimate update and new mine plan determination suggest opportunity to reduce the size of the PFS designed waste facilities during the next project phase.

26.5.4 Environmental and Social Management Plan

Non-contact water management within the mountainous region remains challenging. The eastern upstream catchment area of the current heap leach facility design requires diversion of non-contact water to a different catchment area. The affected area is deemed small and no permitting complications are expected. The permitting application process is to be finalised to limit further infrastructure investment in storm water management.

Ecological and Environmental Impact Assessment studies must be concluded to obtain any related restrictions on water discharge or habitat avoidances within the area.

26.5.5 Waste Rock Dumps

The design of the mine waste facilities is based on limited existing information, reasonable engineering judgement, and a focus on flexibility for the FS design and operations. Additional work recommended for the next level of design will be costed as part of the FS stage of the Project. The following work is recommended to be completed prior to or in conjunction with advancing the facilities to the next level of design:

- Develop and perform a geotechnical investigation under the supervision of the Design Engineer(s) to characterise the foundation conditions, confirm that karst is not present within all mine waste facility footprint areas, and develop site specific parameters (i.e., density, shear strength, consolidation, etc.) to be used in the design. The geotechnical investigation should include geophysics, boreholes and test pits with representative samples collected for laboratory testing.

- Perform a hydrogeological investigation within all mine waste facility footprint areas under the supervision of the Design Engineer(s) to confirm the depths to groundwater and identify groundwater conditions that could impact the design.
- Perform a field reconnaissance under the supervision of the Design Engineer(s) as part of a geologic hazard assessment to confirm the site conditions within all mine waste facility footprint areas and note any potential geologic hazards.
- Develop and perform geotechnical investigations and laboratory testing in coordination with the Design Engineer(s) to identify, characterise and quantify suitable borrow materials for all mine waste facilities for materials specified in the design such as low permeability soil liner, overliner, bedding material, etc. The laboratory testing program should also include the material shear strength and interface shear strength values to be used in the stability analysis.
- In the event of a sulphide concentrator proceeding in the future that involves tailings disposal, perform physical tailings characterisation laboratory testing in coordination with the tailings Design Engineer(s), including grain size testing, Atterberg limits, standard proctor compaction, shear strength testing, liquefaction potential, and permeability testing.
- Perform a mineralogical and geochemical assessment and characterisation of mine waste materials (i.e., tailings, waste rock, overburden, etc.) to assess potential environmental issues. The design of the mine waste facilities and liner requirements should be reviewed to confirm the appropriate containment and risk mitigation strategies.
- The mine waste facilities construction staging and material requirements should be reviewed within the context of the mine plan schedule to confirm that suitable earthwork material would be available for construction.
- Review the mine plan and scheduling to determine if filtered tailings and/or waste rock could be placed in the pits to reduce the volume and corresponding footprint of the surface storage footprints. The in-pit storage option would also need to consider the geochemistry of the mine waste (tailings and waste rock) material and the hydrogeologic conditions around the mine pits.
- Develop a liner sub-grade grading plan for the mine waste facilities that optimises the cut and fill balance (i.e., regrading).
- Confirm the availability of land needed for all elements of the Project. Careful consideration should be given to the southwest corner of the Bigar Hill co-disposal facility which lies outside of DPM licensed area.
- Perform a risk assessment and develop a scope of work to address data gaps, reduce uncertainty, and address high level risks.

- Perform a site specific Probabilistic Seismic Hazard Assessment (PSHA) to determine the appropriate PGA to be used in the pseudo-static stability analysis in the FS design.
- With receipt of the updated information recommended above, perform trade-off studies to confirm the decisions made in the PFS that include the following:
  - Potential filtered tailings co-disposal in the Bigar Hill waste rock dump (currently excluded from future project phase work);
  - Waste rock management dump locations;
  - In pit waste storage.
- The Project definition and mine waste tonnages considered in SLR's PFS design are based on the tonnages produced by the DRA-led pit optimisations performed on the 2018 block model. DRA subsequently updated the block model during the PFS, so the tonnages presented in the PFS and in the SLR design report will differ. SLR has not reviewed or verified the difference in design during the PFS as updated tonnages were finalised toward the end of the PFS period. SLR recommends that the tailings and waste rock tonnages be reviewed in subsequent stages of the Project, and the corresponding facility design be adjusted accordingly.

Beyond the technical data gaps noted above, it is recommended that the Project initiate the following environmental and social governance programs and tailings management system best practises:

- Engage external stakeholders and elicit feedback regarding tailings management strategies and the evaluation of different alternatives.
- Begin developing a risk-based tailings management system framework, including corporate commitment, responsible parties, review processes, and risk management.
- Conduct a review of the effort needed for the Project to comply with the 2020 Global Industry Standard on Tailings Management.

26.5.5.1 Haul Road Recommendations

Upon completion of the mining scheduling work during the PFS, the mine fleet was finalised with smaller 60-ton trucks that require a reduced haul road width of 16 m compared to current 18 m. An estimated bulk earthworks quantity reduction of 10% is expected with such a width reduction and was applied in the capital estimate. Additionally, engineered fill is indicated as more expensive compared to excavation cut work.

The haul roads must be further optimised during the next Project phase to optimise the bulk earthworks toward increasing cut quantities and reducing fill quantities.

26.5.6 Geotechnical and Geochemical

Project facilities have been placed to optimise operations and avoid problematic karst lithologies. Specific geotechnical drilling and test pits must be undertaken to determine ground condition suitability for the various elements. Borehole and test pit locations have been identified. Completion of this investigative work prior to the next Project phase will ensure designs can progress.

ARD test work was undertaken during the PFS with final results still pending. Test work conclusion and interpretation is required prior to proceeding with the next project phase. Potential ARD waste quantities and severity require confirmation, along with appropriate treatment methodologies to ensure the need for a lined waste facility is confirmed.

The ARD potential of the mined sulphide ore also requires confirmation to determine the correct treatment method and storage facility. At the current sulphide ore is estimated at 2.7 Mt. In the absence of a lined waste facility for ARD, the sulphide ore may be added to the HLF, which in turn will require expansion to accommodate the increased ore placement.

26.5.7 Water Management

The design of the water management facilities is based on limited existing information on geochemistry and water quality, reasonable engineering judgement, and a focus on flexibility for the FS design and operations. Additional work recommended for the next level of design will be costed as part of the Feasibility stage of the project. The following work is recommended to be completed prior to advancing the facilities to the next level of design:

- Perform a geochemical assessment and characterisation of tailings and waste rock to evaluate potential environmental issues. The design of the liner system requirements should be reviewed to confirm the appropriate containment and risk mitigation strategies.
- Collect water quality baseline data in the Jagnilo River and its tributary streams.
- Conduct preliminary (or concept level) assimilative capacity study for the Jagnilo River to estimate the maximum analyte concentrations that can be discharged to the receiver without exceeding applicable water quality objectives.
- Identify specific effluent water treatment requirements based on the geochemical characterisation and the preliminary assimilative capacity study.
- Obtain or develop Intensity-Duration-Frequency rainfall data representative of the Project site location to define: i) design rainfall intensities required to calculate design peak flows, and ii) design rainfall volumes required to calculate design pond volumes for runoff resulting from short duration storm events.
- Implement collection of local meteorological data at an on-site meteorological station. Regular maintenance activities should be implemented. Regular checking of collection data should be

performed in order to register reliable and complete records and implement timely corrective actions in case of equipment malfunctions.

- Resume collection of streamflow data in the Jagnilo River system (Bigar and Jasikovo monitoring locations) and expand the program by adding other stations. Initiate collection of streamflow data in the Ogasu Griljei River. Validation of data records should be performed.
- Develop a numerical groundwater model of the pits in order to determine the transient inflow rate during both the initial stages of dewatering and as the pit depth is increased. This model would allow modelling of the impact on dewatering rates of annual changes in groundwater level as well as providing an input into any water quality modelling undertaken.
- Refine the sizing of ditches and ponds at the next stage of Project engineering when new data becomes available (e.g. Intensity-Duration-Frequency rainfall data, streamflow data, and underground inflows to the pits from groundwater numerical modelling).
- Perform a risk assessment and develop a scope of work to address data gaps and intolerable risks.
- Develop a stochastic flow (water balance) model to simulate the water management strategy with greater detail incorporating decision making functions and considering probabilistic climatic scenarios. Consider the mine plan for pit development sequencing to account for variation of pit dewatering rates through the mine life.
- Conduct flood routing to refine the calculation of design peak flows used for sizing of ditches and emergency spillways.

Beyond the technical data gaps identified in this Report, it is recommended that the Project initiates environmental and social governance programs. Specific recommendations related to water management include:

- Project climate change vulnerability assessment (especially as it pertains to the sizing and design of water management structures, and closure concepts within this report).
- Commence a water stewardship program and engage stakeholders for a holistic watershed perspective.

Note that early engagement of stakeholders can reduce permitting or operating challenges for the Project in the future and is consistent with international industry best practise.

26.6 Environmental Studies, Permitting and Social or Community Impact

The impacts of the Project will be identified definitively during the ESIA, and plans to mitigate, manage or offset will be formulated in the ESMP and permitting process. Ultimately, many impacts will be addressed in the closure period where the larger spatial impacts will be rehabilitated.

It is critical that both technical design of plant, and safety management procedures for the cyanide heap leach facility and process plant are implemented in the construction phase. These must be maintained in the operations phase, supported by corrective auditing, in line with the International Cyanide Management Code, to which DPM is signatory and against which the Project will be audited. Careful stakeholder communications planning will be required to maintain good community relations and reduce the risk of objections during the permitting process.

One of the key areas of uncertainty at this point in the PFS design is in the extent to which waste rock are acid generating / metal leaching. There is a study underway which will inform the proportion of waste rock with this characteristic, and this will be a critical consideration in the FS stage design and distribution of liners, specification of the water treatment plant, design and scheduling of waste rock dumps, capping and rehabilitation requirements. It is therefore recommended that the acid rock drainage study be completed in full, prior to reassessing these designs elements at the FS stage.

Front-loading and initiating permitting processes that have a long lead time, such as the Spatial Plan will be important to maintaining progress on the overall permitting timeline. Ongoing engagement with regulators will be critical, particularly given the limited precedent for permitting private sector mines in Serbia. The Project has already made a commitment to alignment with EU and international requirements.

Maintaining the Project's social licence to operate will be of key importance, building upon the existing good relationship with the local community. This will be managed through the Stakeholder Engagement Plan, Communications Strategy and Grievance Mechanism.

Outline scopes for ongoing and future baseline work required for the ESIA have been presented in Table 20.1. It is essential that these are completed so that impacts are properly identified, their significance assessed, and mitigation and management solutions identified, such as discharge flow velocity and seasonality. The ESIA process will develop an ESMP to map out these mitigating procedures and designs to be implemented as the project goes forward to construction.

A commitment to a Project-specific ESMP, with identified plans, procedures and appropriate resources, will encourage the implementation of the ESMP and compliance with permit conditions. Ongoing revision of the Closure Plan throughout FS, Construction, Commissioning and Operation phases, with appropriate and active stakeholder (community and regulator) participation will give the best chance for long-term closure goals to be met, and a positive legacy made from the relatively short-lived Timok Gold Mine.

26.7 Proposed Work Program

To ensure the potential viability of the mineral resources, the following activities should be undertaken in the next phase of the Project. These activities as well as their estimated costs are shown in Table 26.1.

Table 26.1 – Estimated Budget for Next Phase

Activities	Estimated Budget ($ USD)
FS Resource Drilling and Assays	2,100,000
FS Geotechnical Drilling and Testwork	500,000
Metallurgical Testwork	650,000
Environmental Studies, including ARD/ML	490,000
Total	***3,740,000***

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28 CERTIFICATE OF QUALIFIED PERSON

20 Queen Street West / 29th Floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800 8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Timok Project Pre-Feasibility Study, Zagubica, Serbia*" which is effective as of February 23, 2021 and issued on March 30, 2021 (the Technical Report) prepared for Dundee Precious Metals Inc. (DPM or the Company).

I, *Philippus (Philip) de Weerdt, Pr.Eng, PMP, MBA.*, do hereby certify:

1. I am a Project Manager with DRA Americas with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada.
2. I am a graduate from University of Johannesburg, Johannesburg, South Africa in 2006 with a B. Eng Mechanical, and graduate of the University of Pretoria, Pretoria in 2016 with a Masters in Business Administration.
3. I am a registered member of the Engineering Council of South Africa (#20150058).
4. I am a register Project Management Professional (PMP).
5. I have worked continuously as a Mechanical Engineer and Project Manager since my graduation.
6. My relevant work experience includes:
    - Design and construction execution of mining related processing facilities and infrastructure in Western and Central Africa.
    - Project management and oversight of design, estimating and execution of mining related processing facilities and infrastructure in Western Africa.
    - Participation and author of several NI 43-101 Technical Reports.
7. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
8. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
9. I am responsible for the preparation of Sections 2 to 5, 18, 18.15.2, 19, 21, 23, and 24, with the exception of Sections 18.14, 18.15, 21.2.2 and 21.2.3. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.

10. I personally did not visit the property that is the subject to the Technical Report.
11. I have had no prior involvement with the property that is the subject of the Technical Report.
12. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
13. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

"Original signed and sealed"

Philip de Weerdt, Pr.Eng, PMP, MBA
Project Manager
DRA Americas Inc.

DUNDEE SOFIA OFFICE

26 Bacho Kiro Street
Sofia 1000
BULGARIA

Tel: +359 2 930 15 00
Fax: +359 2 930 15 95

www.dundeeprecious.com

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Timok Project Pre-Feasibility Study, Zagubica, Serbia*" which is effective as of February 23, 2021 and issued on March 30, 2021 (the "Technical Report") prepared for Dundee Precious Metals Inc. ("DPM" or the "Company").

I, *Ross Overall, C.Sci, MIMMM, FGS*, do hereby certify:

1. I hold the position of Corporate Mineral Resources Manager, of Dundee Precious Metals Inc. the parent company of Dundee Precious Metals Chelopech, 26 Bacho Kiro, Sofia 1000, Bulgaria.
2. I graduated from Camborne School of Mines, UK in 2004 and hold a level 2.1 BSc (Hons) degree in Applied Geology.
3. I am a Chartered Professional Member of the institute of Materials, Minerals and Mining (CSci, MIMMM, Membership Number IOM/112/000538).
4. My relevant work experience includes underground and open pit mining, resource geology, geostatistics, exploration and grade control. My work experience includes 17 years in the mining industry and have participated and authored several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am not independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 6,7,8,9,10,11 and 12. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I have visited the project site that is the subject of this Technical Report on a regular basis since 2012. I last personally visited the Timok property that is the subject of the Technical Report, between the dates of December 4th and December 7th, 2019.
9. I have had prior involvement with the property that is the subject of this Technical Report as Corporate Mineral Resources Manager with Dundee Precious Metals Inc. I work in Bulgaria providing technical supervision for all of Dundee Precious Metal's operating mines, development and exploration projects.

DUNDEE SOFIA OFFICE

26 Bacho Kiro Street
Sofia 1000
BULGARIA

Tel: +359 2 930 15 00
Fax: +359 2 930 15 95

www.dundeeprecious.com

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

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

Dated this 30th day of March 2021

Ross

Ross Overall BSc (Hons), CSci, MIMMM, FGS.
Corporate Mineral Resources Manager
Dundee Precious Metals Inc.

20 Queen Street West / 29th Floor / Toronto / Ontario / Canada / Quebec / Canada / M5H 3R3
T +1 416 800 8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Timok Project Pre-Feasibility Study, Zagubica, Serbia*" which is effective as of February 23, 2021 and issued on March 30, 2021 (the Technical Report) prepared for Dundee Precious Metals Inc. (DPM or the Company).

I, *Schadrac Ibrango, P.Geo., Ph.D., MBA.,* do hereby certify:

1. I am a Principal Geologist Consultant with DRA Americas Inc, with an office at 555 René-Lévesque Blvd. West, 6th Floor, Montreal, Canada;
2. I am a graduate from University of Ouagadougou (Burkina-Faso) with a Master Degree in Geology in 1998, a Ph.D. in Engineering of Darmstadt University of Technology (Germany) in 2005 and an executive MBA from Université du Québec à Montréal (Canada) in 2016;
3. I am a registered member of the Ordre des Géologues du Québec (OGQ), membership # 1102 and Professional Engineers & Geoscientists of Newfoundland and Labrador, membership # 07633;

   I have worked continuously as a geologist for more than 20 years in the mining industry since my graduation from university;
4. I have worked on similar projects to the Timok Project in Canada, South America and in Africa; My experience for the purpose of the Technical Report includes:
   a) Hands-on experience in exploration and mining for gold deposits;
   b) Generation of gold and precious metals exploration projects in Canada and Africa;
   c) Management of gold and precious metals exploration projects in Canada and Africa;
   d) Gold projects review as independent QP in Africa and South America;
   e) Participation as QP in the preparation of a Ni43-101 technical reports for gold project in in Africa and Canada;
   f) Design, implementation and supervision and implementation of drilling programs;
   g) Participation in the preparation of parts of several other NI 43-101 compliant Technical Reports.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
6. I have not visited the property that is the subject to this Technical Report;

7. I have participated in the preparation of this Technical Report and am responsible for Section 14 as and parts of Sections 1 and 25 to 27;
8. I have not had prior involvement with the property that is the subject of the Technical Report;
9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101;
10. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report;
11. I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice.
12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of March 2021

"Original Signed and sealed"
Schadrac Ibrango, P.Geo., Ph.D., MBA

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
T +1 514 288-5211 / **E** info@draglobal.com / https://www.draglobal.com/

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I, Daniel M. Gagnon*, P. Eng.,* do hereby certify:

1. I am Vice President Mining, Geology and Met-Chem Operations, with DRA Americas Inc located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 1995 with a bachelor degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #118521).
4. I have worked as a Mining Engineer for a total of 25 years continuously since my graduation.
5. My relevant work experience includes:
    - Design, scheduling, cost estimation and Mineral Reserve estimation for several open pit studies similar to Timok in Canada, the US, South America, West Africa and Morocco.
    - Technical assistance in mine design and scheduling for mine operations in Canada, the US and Morocco.
    - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Sections 15 and 16, with the exception for Sections 16.3 and 16.4. I am also responsible for the relevant portions of Sections 1, 21.2, and 25 to 27 of the Technical Report.

9. I did not visit the property that is the subject to the Technical Report.

10. I have had no prior involvement with the property that is the subject of the Technical Report.

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

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

Dated this 30 day of March 2021

"Original Signed and sealed"
Daniel M. Gagnon, P. Eng.
VP Mining, Geology and Met-Chem Operations
DRA Americas Inc.

CSA Global
Mining Industry Consultants

CSA Global (UK) Ltd
(UK Office)

Suite 2, First Floor, Springfield House
Horsham, West Sussex
RH12 2RG
UNITED KINGDOM

T +44 1403 255 969
F +44 1403 240 896
E info@csaglobal.com
www.csaglobal.com

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Timok Project Pre-Feasibility Study, Zagubica, Serbia*" which is effective as of February 23, 2021 and issued on March 30, 2021 (the Technical Report) prepared for Dundee Precious Metals Inc. (DPM or the Company).

I, Galen White, FAusIMM, do hereby certify:

1. I am a Principal Consultant with CSA Global (UK) Limited located at Suite 2, First Floor, Springfield House, Springfield Road, Horsham, West Sussex, RH12 2HD, UK.
2. I hold a B.Sc. (Hons) degree in Geology from the University of Portsmouth, UK (1996)
3. I am a registered Fellow in good standing of the Australasian Institute of Mining and Metallurgy (AusIMM, #226041).
4. My relevant work experience includes over 25 years continuous experience in the exploration, evaluation and mining of gold deposits in Europe, Australia and Africa. I have conducted Exploration activities, held roles in Mineral Resource Evaluation and Production in open-pit and underground gold deposits and have reviewed and audited Mineral Resource Estimates over the past 15 years spent in consulting. I have participated in, and been the author of several NI43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 14.18, and the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I have not visited the property that is the subject to the Technical Report.
9. I have had prior involvement with the property that is the subject of the Technical Report, being contribution to the assessment of Mineral Resources at the property in 2018.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

Galen White, FAusIMM

Principal Consultant

CSA Global (UK) Limited

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
T +1 514 288-5211 / **E** info@draglobal.com / https://www.draglobal.com/

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I, *Claude Bisaillon, P. Eng.,* do hereby certify:

1. I am Senior Geological Engineer with Met-Chem, a division of DRA Americas Inc. with an office at suite 600, 555 René-Lévesque Blvd. West, Montreal, Quebec, Canada;
2. I am a graduate from Concordia University in Montreal in 1991 with a B.Sc. in geology and from the Université Laval in Quebec City in 1996 with a B.Ing. in geological engineering;
3. I am a registered member of "*Ordre des Ingénieurs du Québec*" (#116407). I am also a registered engineer in the province of British Columbia (#25669). I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum;
4. I have worked as an engineer continuously since graduation from University in 1996.
5. My relevant work experience includes:
   - Over 23 years of consulting in the field of Mineral Resource estimation, orebody modelling, mineral resource auditing and geotechnical engineering in Canada, the USA, Asia, and South America.
   - Participated and/or supervised in several NI 43-101 Technical Reports.
   - QP Review, audits, due diligence, interpretation of geoscientific data for several projects.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Section 16.3. I am also responsible for the relevant portions of Sections 1, and 25 to 27 of the Technical Report.

9. I did not visit the property that is the subject to the Technical Report.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

"Original Signed and sealed"

Claude Bisaillon, P. Eng.
Senior Geological Engineer
DRA Americas Inc.

20 Queen Street West / 29th Floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800 8797 / **E** info@draglobal.com / https://www.draglobal.com/

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I, *Volodymyr Liskovych, Ph.D., P. Eng.*, do hereby certify:

1. I am a Principal Process Engineer with DRA Americas with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada.
2. I am a graduate from Zaporizhzhia State Engineering Academy, Zaporizhzhia, Ukraine in 1996 with a Metallurgical Engineer Degree, and a graduate from National Metallurgical Academy of Ukraine, Dnipro, Ukraine with the PhD degree in Metallurgical Engineering in 2001.
3. I am a registered member of the Professional Engineers of Ontario (#100157409).
4. I have worked continuously as a Metallurgical Engineer for more than *24* years since my graduation from Zaporizhzhia State Engineering Academy.
5. My relevant work experience includes:
    - Review and report on mineral processing and metallurgical operations and projects around the world for due diligence and regulatory requirements;
    - Engineering study (PEA, PFS, FS, and Detailed Engineering) project work on many minerals processing and metallurgical and hydrometallurgical projects around the world, and in North America;
    - Operational experience in operations management and operational support positions in metallurgical and hydrometallurgical operations in Ukraine, Canada, and Brazil.
    - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for the preparation of Sections 13 and 17. I am also responsible for the relevant portions of Sections 1, 21.2, and 25 to 27 of the Technical Report.
9. I did not visit the property that is the subject to the Technical Report.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

"Original Signed and sealed"
Volodymyr Liskovych, PhD, P. Eng.
Principal Process Engineer
DRA Americas Inc.

CERTIFICATE OF QUALIFIED PERSON

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I, Luis Vasquez, M.Sc., P.Eng., do hereby certify:

1. I am a Senior Hydrotechnical Engineer with SLR Consulting (Canada) Ltd. at Suite 501, 55 University Ave., Toronto, ON M5J 2H7.
2. I am a graduate of Universidad de Los Andes, Bogotá, Colombia, in 1998 with a B.Sc. degree in Civil Engineering and a Master of Science – Water Resources Engineering in 1999. I have 20 years of experience in the field of water resources engineering.
3. I am a registered member of the Professional Engineers Ontario.
4. My relevant work experience includes:
    - Experience in the field of water management,
    - Design an operational support experience on various projects and mine sites,
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 16.4 and 18.15, with the exception of Section 18.15.2. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I did not visit the property that is the subject to the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

(Signed and Sealed) Luis Vasquez

“Original Signed and sealed”
Luis Vasquez, M.Sc., P. Eng.
Senior Hydrotechnical Engineer
SLR Consulting (Canada) Limited

SLR Consulting (Canada) Ltd.
501-55 University Avenue, Toronto ON M5J 2H7

CERTIFICATE OF QUALIFIED PERSON

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I, David Ritchie, M.Eng., P.Eng., do hereby certify:

1. I am a Managing Principal and Engineering Operations Manager with SLR Consulting (Canada) Ltd. at Suite 501, 55 University Ave., Toronto, ON M5J 2H7.
2. I am a graduate from Ryerson Polytechnic University, Toronto, Canada (Bachelor of Engineering - Civil in 1995) and University of Western Ontario (Master of Engineering - Geotechnical in 2000). I have over 25 years of experience in the field of tailings management and geotechnical engineering.
3. I am a registered member in a good standing of the Professional Engineers Ontario.
4. My relevant work experience includes:
    - Experience in the field of tailings and waste management,
    - Design an operational support experience on various projects and mine sites, and
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 18.14. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I did not visit the property that is the subject to the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

(Signed and Sealed) David Ritchie

“Original Signed and sealed”

David Ritchie, M.Eng., P. Eng.
Managing Principal, Engineering Operations Manager
SLR Consulting (Canada) Ltd,

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Timok Project Pre-Feasibility Study, Zagubica, Serbia*" which is effective as of February 23, 2021 and issued on March 30, 2021 (the Technical Report) prepared for Dundee Precious Metals Inc. (DPM or the Company).

I, Kevin Leahy, BSc (Hons), PhD, CGeol, SiLC do hereby certify:

1. I am a geologist and Technical Director with Environmental Resources Management Ltd located at 2nd Floor, Exchequer Court, 33 St Mary Axe, London, UK. EC3A 8AA
2. I am a graduate from University of Leeds, UK, where I obtained a degree in Geological Sciences in 1992, and a PhD on the subject of diamond exploration in 1996.
3. I am a registered Fellow of The Geological Society, Burlington House, Piccadilly, London, UK, and have been a Chartered Geologist there since 2005. I am also a Registered Suitably Qualified Person and Specialist in Land Condition in the UK Land Forum National Quality Mark Scheme since 2017.
4. My relevant work experience includes geological exploration, environmental impact assessment, mine audit and land contamination remediation and closure planning on numerous mines, processing plants and smelters over my 25-year career:
    - Geological exploration for minerals in South Africa, Equatorial Guinea, Sweden and Canada, as well as a structural geologist for numerous hydrocarbon exploration projects. In the last five years jointly developed a hydro-geochemical exploration tool, including several projects in the Timok Belt, Serbia as well in the US and Australia.
    - Environmental Impact Assessment on dozens of EIA projects in Europe, Africa and Asia, including three in Serbia, on a variety of mineral targets and deposit types. My role in EIA projects is usually as topic lead on soils and geology, also contributing to surface and groundwater and early closure planning.
    - Mine audits on several sites in Europe, Asia and South America both for transaction due diligence and for compliance with environmental standards: internal, national and international.
    - Closure planning and land contamination projects on several mine, smelter and processing sites in Europe.
    - Participation in several NI 43-101 Technical Reports, including for projects in Serbia.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 20 with the exception of Sections 1 to 19 and 21

to 27. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.

8. I have not personally visited the property that is the subject to the Technical Report due to COVID restrictions. However, several members of my team have visited the site for water, social, biodiversity and cultural heritage baseline surveys in 2019 and 2020.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

"Original Signed and sealed"

Kevin Leahy, BSc (Hons), PhD, CGeol, SiLC
Technical Director
Environmental Resource Management Ltd

20 Queen Street West / 29th Floor / Toronto / Ontario / Canada / Quebec / Canada / M5H 3R3
T +1 416 800 8797 / E info@draglobal.com / https://www.draglobal.com/

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I, *Alexander Duggan, P. Eng.*, do hereby certify:

1. I am a Civil Engineer and Estimator Consultant located at 8045 Wyandotte Street, East, Windsor, N8S 1T2, Canada.
2. I graduated with a Bachelor of Science degree in Civil Engineering from the University of Aston, Birmingham, UK, IN 1982. In addition, I have obtained a Master of Science in Planning from the University of Salford, UK in 1984.
3. I am a current member of the Professional Engineers Ontario (PEO No. 100103898),
4. I have worked as an Estimator in the mining and heavy industries for more than 33 years.
5. My relevant work experience includes:
   -
   - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Section 22. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
9. I did not visit the property that is the subject to the Technical Report.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information

and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30 day of March 2021

LICENSED PROFESSIONAL ENGINEER
A. O. DUGGAN
100103898
PROVINCE OF ONTARIO

Alexander Duggan, P. Eng.
Estimator Consultant