OreWin
Independent Mining Consultants

SSR MINING INC.

Çöpler District Master Plan 2021

Technical Report

February 2022

Job No. 21007

IMPORTANT NOTICE

This notice is an integral component of the Çöpler District Master Plan 2021 Technical Report (CDMP21TR) and should be read in its entirety and must accompany every copy made of the Technical Report. The CDMP21TR has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The CDMP21TR has been prepared for SSR Mining Inc. (SSR Mining) by OreWin Pty Ltd (OreWin). The CDMP21TR is based on information and data supplied to OreWin by SSR Mining and other parties and where necessary OreWin has assumed that the supplied data and information are accurate and complete.

The CDMP21TR includes a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

The conclusions and estimates stated in the CDMP21TR are to the accuracy stated in the CDMP21TR only and rely on assumptions stated in the CDMP21TR. The results of further work may indicate that the conclusions, estimates and assumptions in the CDMP21TR need to be revised or reviewed.

OreWin has used its experience and industry expertise to produce the estimates and approximations in the CDMP21TR. Where OreWin has made those estimates and approximations it does not warrant the accuracy of those amounts and it should also be noted that all estimates and approximations contained in the CDMP21TR will be prone to fluctuations with time and changing industry circumstances.

The CDMP21TR should be construed in light of the methods, procedures, and techniques used to prepare the CDMP21TR. Sections or parts of the CDMP21TR should be read in context of the entire CDMP21TR and should not be removed from their original context.

The CDMP21TR is intended to be used by SSR Mining, subject to the terms and conditions of its contract with OreWin. Recognising that SSR Mining has legal and regulatory obligations, OreWin has consented to the filing of the CDMP21TR with the Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR). Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

Title Page

Project Name:	Çöpler District Master Plan 2021
Title:	Çöpler District Master Plan 2021 Technical Report
Location:	Erzincan Province, Turkey
Effective Date of Technical Report:	31 December 2021
Effective Date of Mineral Resources:	31 December 2021
Effective Date of Mineral Reserves:	31 December 2021

Qualified Persons:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director - Mining, was responsible for the overall preparation of the CDMP21TR and, the Mineral Reserve estimates, Sections 1 to 6; Sections 13; Sections 15 to 27.
- Sharron Sylvester, BSc (Geol), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director - Geology, was responsible for the preparation of the Mineral Resources, Sections 1 to 12; Section 14; Sections 23 to 27.

Signature Page

Project Name:	Çöpler District Master Plan 2021
Title:	Çöpler District Master Plan 2021 Technical Report
Location:	Erzincan Province, Turkey
Date of Signing:	23 February 2022
Effective Date of Technical Report:	31 December 2021

/s/ Bernard Peters

Bernard Peters, Technical Director – Mining, OreWin Pty Ltd, BEng (Mining), FAusIMM (201743)

/s/ Sharron Sylvester

Sharron Sylvester, Technical Director – Geology, OreWin Pty Ltd, BSc (Geol), RPGeo AIG (10125)

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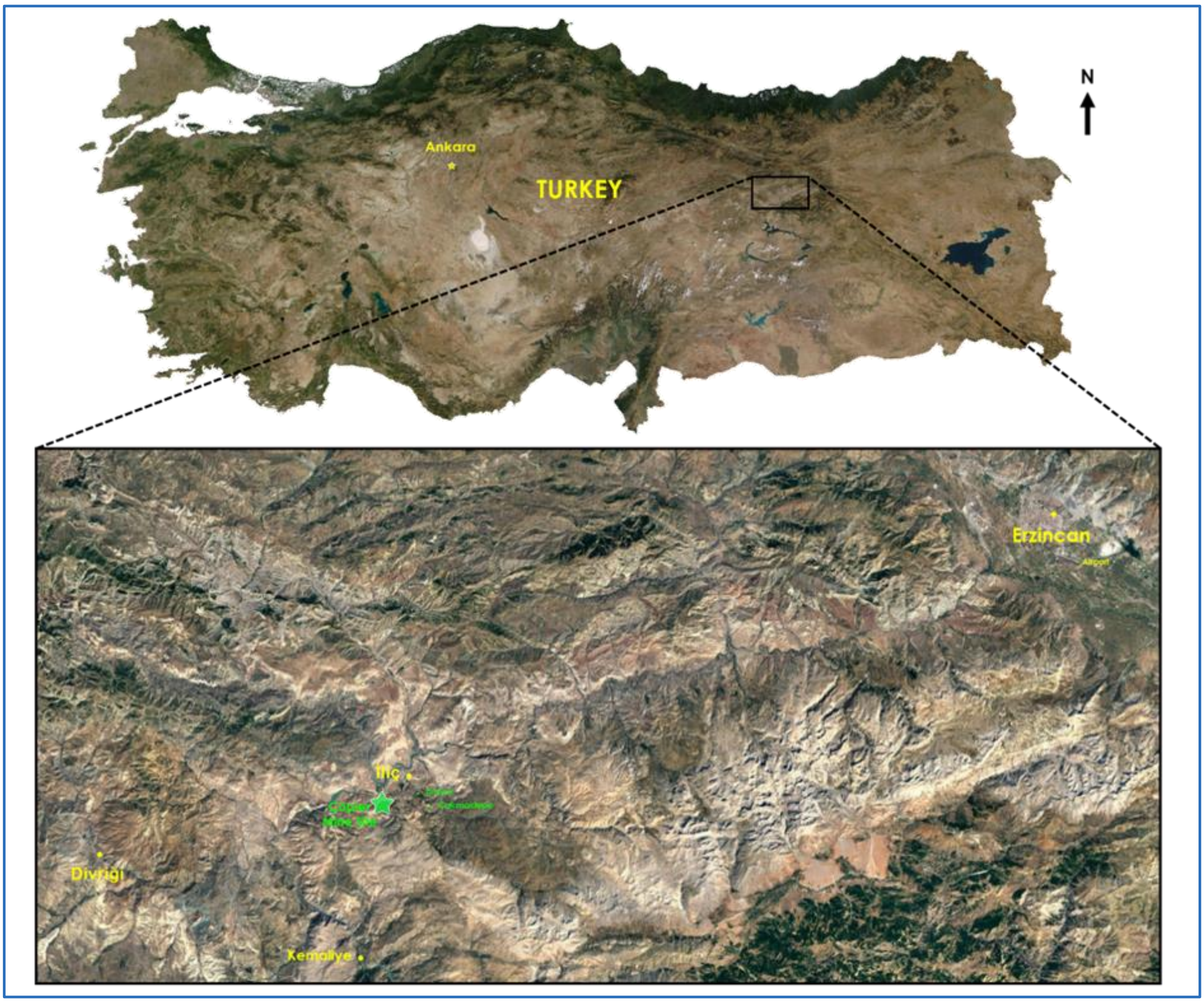
1 SUMMARY

1.1 Introduction

The Çöpler District Master Plan 2021 Technical Report (CDMP21TR) is an independent Technical Report prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The Çöpler project is located in east central Turkey, 120 km west of the city of Erzincan, in Erzincan Province, 40 km east of the iron-mining city of Divriği (one-hour drive), and 550 km east of Turkey's capital city, Ankara (Figure 1.1). The nearest urban centre, İliç, (approximate population 3,800), is located approximately 6 km north-east of the Çöpler mine.

Figure 1.1 Project Location Map



SSR, 2020

SSR is a gold mining company with four producing assets, located in the USA, Turkey, Canada, and Argentina, and with development and exploration assets in the USA, Turkey, Mexico, Peru, and Canada. SSR is listed on the NASDAQ (NASDAQ:SSRM), the Toronto Stock Exchange (TSX:SSRM), and on the Australian Stock Exchange (ASX:SSR).

The Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%. Exploration tenures surrounding the project area and mining at Çakmaktepe are subject to joint venture agreements between SSR and Lidya that have varying interest proportions. SSR controls 50% of the shares of Kartaltepe Madencilik Sanayi ve Ticaret Anonim Şirketi (Kartaltepe) and 30% of Tunçpinar Madencilik Sanayi ve Ticaret Anonim Şirketi (Tunçpinar). The remaining 50% of shares for the Kartaltepe and 70% of shares of Tuncpinar are controlled by Lidya.

The key features of the CDMP21TR are:

- Updated Mineral Resources on the Ardich deposit.
- Updated Mineral Reserves on the Çöpler deposits.
- A new Mineral Reserve on the Ardich deposit.
- Initial Assessment on the Çöpler deposits analysing copper processing facilities.

The Mineral Reserves are supported by feasibility study level work on the currently operated pits at the Çöpler and Çakmaktepe deposits, the brownfield Ardich deposit, and the oxide heap leach facility and sulfide plant in the Reserve Case.

The Mineral Resource for the Ardich deposit has benefited from additional drilling and an updated model in 2021. The mining of Ardich requires development of a new open pit that is approximately 6 km east of the current Çöpler pit and 1 km north of the Çakmaktepe pits.

The CDMP21TR also includes an Initial Assessment on a proposed copper concentrator and a copper recovery circuit added to the existing sulfide plant. The Initial Assessment Case analyses inclusion of this previously-unexploited revenue stream and reflects the increased capital costs and infrastructure required to leverage value from the copper mineralisation. The Initial Assessment Case is a whole-of-project analysis that represents a significant change from the Reserve Case economics analysis results and production.

The Initial Assessment Case is a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

A plan showing facility location and the boundary of the Reserve Case is shown in Figure 1.2. The Initial Assessment Case boundary is the same as the Reserve Case boundary.

The key production and economic analysis from the CDMP21TR are shown in Table 1.1.

The economic analysis uses long-term metal price assumptions of $1,600/oz gold, $21.00/oz silver, and $3. 40/lb copper. These prices are based on a review of consensus price forecasts from financial institutions and similar studies recently published.

The Reserve Case production includes 22.6 Mt at 1.69 g/t Au oxide ore processed by heap leaching and 52.9 Mt at 2.33 g/t Au processed in the sulfide plant. Total production is 75.4 Mt at 2.14 g/t Au. Total gold production is 4.4 Moz. Mining at the Çöpler pit is completed in 2029 and at Ardich in 2034. Oxide heap leach stacking is completed in in 2034, while sulfide processing will continue from stockpiles until 2042.

The Reserve Case results include:

- After-tax NPV at a 5% real discount rate is $1.73 billion.
- Mine life of 21 years.

An IRR is not reported as the operation is cash positive in each year of the mine plan until closure. The Reserve Case average all-in sustaining cost (AISC) is $966/oz gold.

The Initial Assessment Case production is oxide of 41.8 Mt at 1.26 g/t Au, 59.7 Mt at 2.45 g/t Au of sulfide, and an additional 24.9 Mt at 0.50 g/t Au and 0.20% Cu amenable to concentrator treatment for a total of 126.4 Mt at 1.67 g/t Au. The gold production in the Initial Assessment Case is 5.4 Moz and 164 Mlb of copper. Copper is produced from all three processing streams. The impact of including the copper concentrator as a processing facility is to expand the Çöpler pit, which ceases mining in 2043. Additional production in the Initial Assessment Case comes from feed of 1.8 Mtpa to the copper concentrator and from additional sulfide and oxide processing feed that is exposed when the pit gets deeper. Total capital including contingency of 25% for the copper concentrator and the copper recovery circuit in the sulfide plant is $218M. The capital costs have an accuracy of ±50%.

The Initial Assessment Case results include:

- After-tax NPV at a 5% discount rate of $2.00 billion.
- Mine life of 22 years.

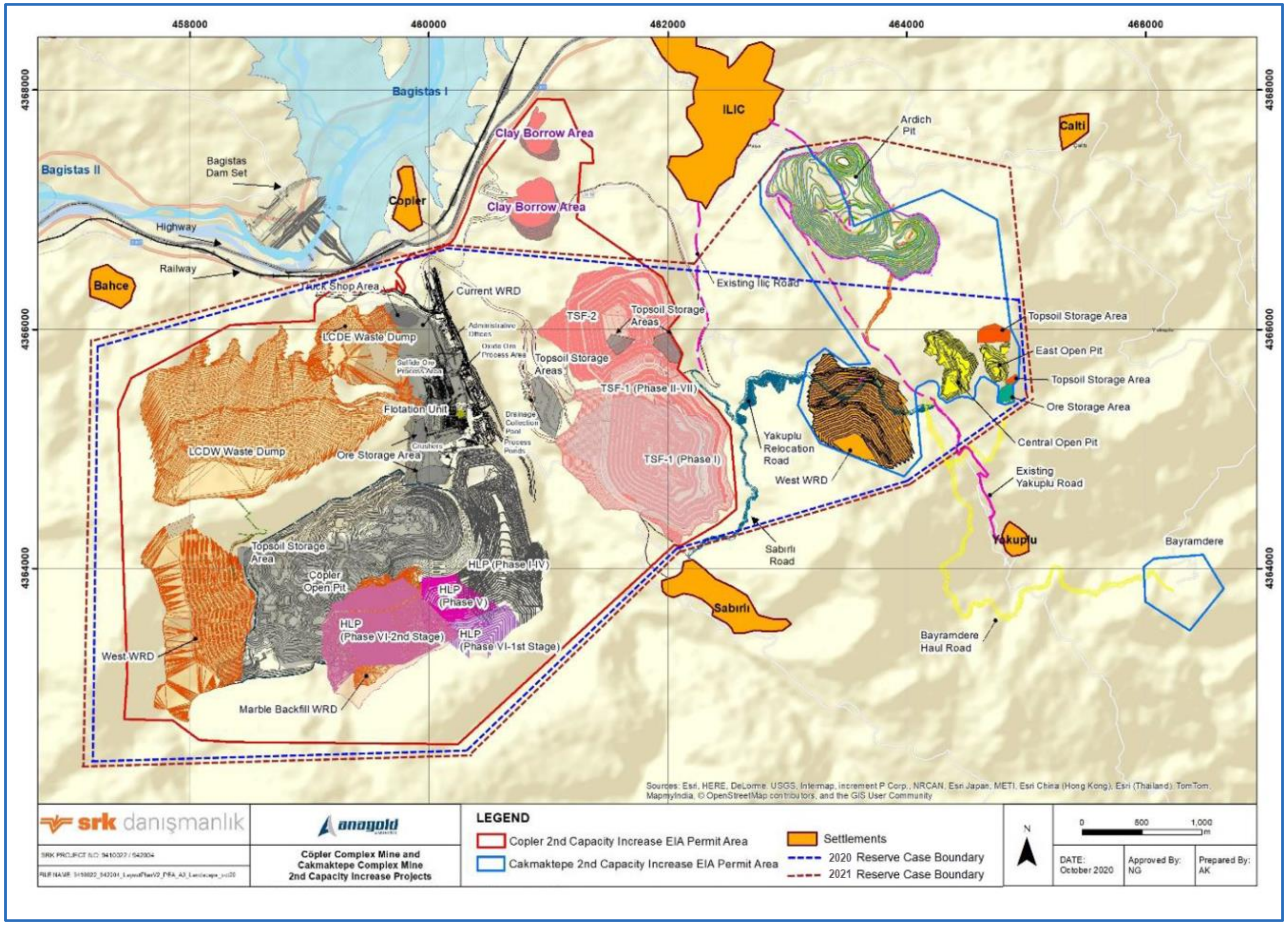
The initial Assessment Case shows an average AISC of $924/oz gold.

The Initial Assessment Case is a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

The Initial Assessment Case is described in Section 24 of the CDMP21TR.

Key results of the Reserve Case and Initial Assessment Case economic analyses are shown in Table 1.1.

Figure 1.2 CDMP21TR Reserve Case and Initial Assessment Case Boundaries



Anagold, 2022

Table 1.1 CDMP21TR Results Summary

Item	Unit	Reserve Case	Initial Assessment Case
Oxide Processed			
Heap Leach Quantity	kt	22,557	41,792
Au Feed Grade	g/t	1.69	1.26
Sulfide Processed			
Quantity Milled	kt	52,892	59,654
Au Feed Grade	g/t	2.33	2.45
Cu Concentrator Processed			
Quantity Milled	kt	–	24,939
Au Feed Grade	g/t	–	0.50
Cu Feed Grade	%	–	0.20
Total Gold Produced			
Oxide – Gold	koz	765	1,068
Sulfide – Gold	koz	3,604	4,078
Cu Concentrator – Gold	koz	–	222
Total – Gold	koz	4,369	5,368
Total Copper Production	Mlb	0.02	164
5-Year Annual Average			
Average Gold Produced	kozpa	278	300
Free Cash Flow	$Mpa	158	165
Total Cash Costs (CC)	$/oz gold	880	761
All-in Sustaining Costs (AISC)	$/oz gold	1,071	938
Key Financial Results			
LOM Total Cash Costs (CC)	$/oz gold	803	783
LOM All-in Sustaining Costs (AISC)	$/oz gold	966	924
Site Operating Costs	$/t treated	45.91	43.79
After-Tax NPV5%	$M	1,732	2,004
Mine Life	years	21	22

5-Year annual average is for the period 1 January 2022 through 31 December 2026
LOM is life-of-mine

The after-tax net present value (NPV) sensitivity to metal price variation is shown in Table 1.2 for gold prices from $1,000–$2,000/oz.

Table 1.2 Gold Price Sensitivity

After-Tax NPV5% ($M)	Long-Term Gold Price ($/oz)					
Case	1,000	1,200	1,350	**1,600**	1,750	2,000
Reserve Case	769	1,115	1,370	**1,732**	1,939	2.252
Initial Assessment Case	859	1,294	1.579	**2,004**	2,259	2,642

1.2 Mineral and Surface Rights

Anagold holds the exclusive right to engage in mining activities within the Çöpler project area. Anagold holds six granted licences covering a combined area of approximately 16,600 ha. Mineral title is held in the name of Anagold. Kartaltepe holds eight licences covering approximately 9,200 ha. The total near-mine tenement package is approximately 25,800 ha. Anagold currently holds sufficient surface rights to allow continued operation of the mining operation in the Reserve Case.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Çöpler project is serviced by road and rail networks. The mine is accessed from the main paved highway between Erzincan and Kemaliye. The project area is in the Eastern Anatolia geographical district of Turkey. Mining operations are conducted year-round. The climate is typically continental with cold wet, winters and hot dry, summers.

1.4 History

The Çöpler region has been subject to gold and silver mining dating back at least to Roman times. The Turkish Geological Survey (MTA) carried out regional exploration work in the early-1960s that was predominately confined to geological mapping. In 1964, a local Turkish company started mining for manganese, continuing through until closing in 1973. Unimangan Manganez San A.Ş. (Unimangan) acquired the property in January 1979 and re-started manganese production, continuing until 1992.

In 1998, Anatolia Minerals Development Ltd (Anatolia) identified several porphyry-style gold–copper prospects in east central Turkey and applied for exploration licences for these prospects. During this work, Anatolia identified a prospect in the Çöpler basin. This prospect and the supporting work were the basis for a joint venture agreement for exploration with Rio Tinto and Anatolia and in January 2004, Anatolia acquired the interests of Rio Tinto and Unimangan.

In August 2009, a joint venture agreement between Anatolia and Lidya was executed.

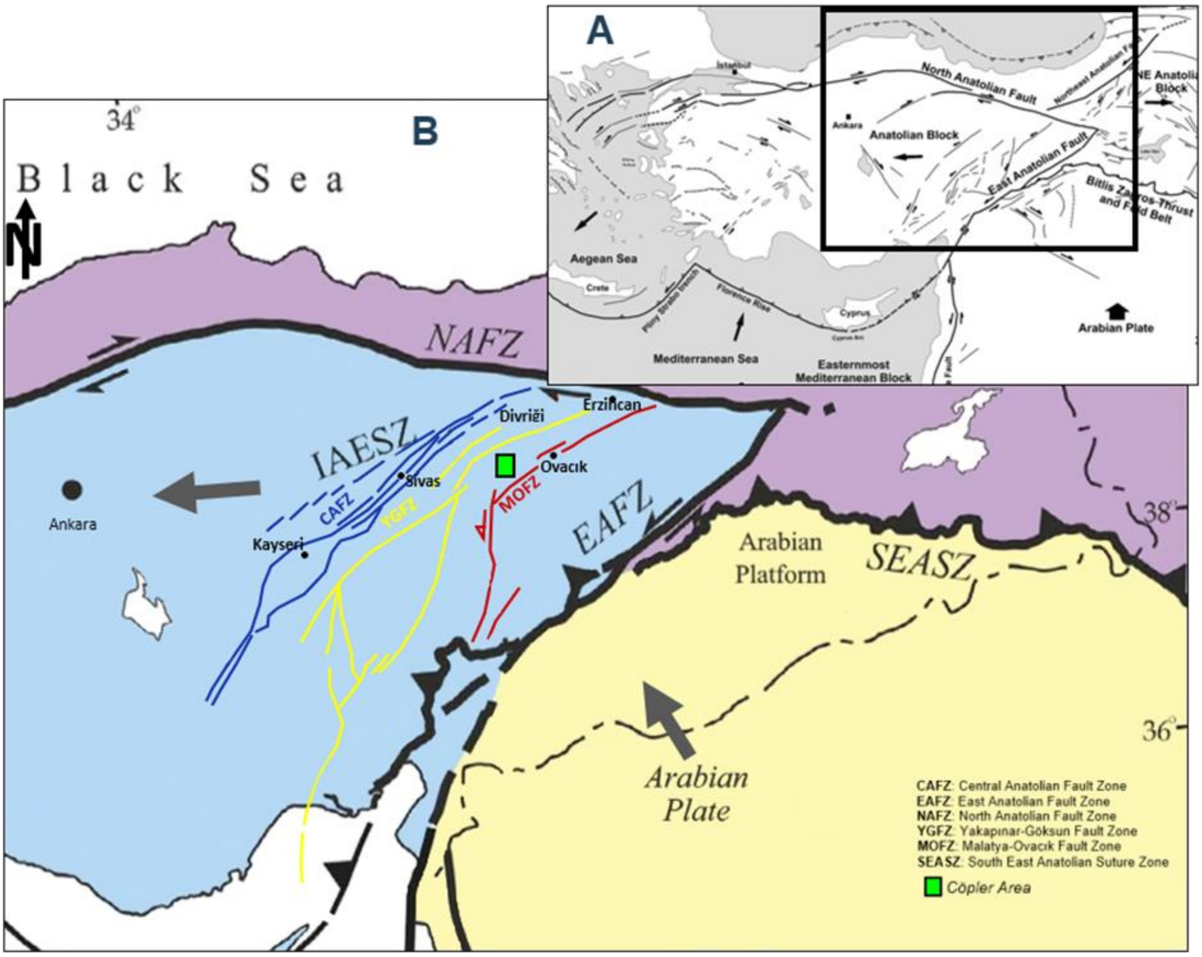
In February 2011, Anatolia merged with Avoca Resources Limited, an Australian company, to become Alacer Gold Corp. (Alacer). In September 2020, Alacer merged with SSR.

Technical Reports have been prepared on the project, in accordance with NI 43-101 Standards for Disclosure for Mineral Projects, since 2003. The previous Technical Report on the project, issued in 2020, described a Reserve Case plus a Preliminary Economic Assessment (PEA) on the Ardich project.

1.5 Geological Setting and Mineralisation

The project is located near the northern margin of a complex collision zone that lies between the Pontide Belt / North Anatolian Fault, the Arabian Plate, and the East Anatolian Fault, which bounds several major plates. The region underwent crustal thickening related to the closure of a single ocean, or possibly several oceanic and micro-continental realms, in the late Cretaceous to early Tertiary period. Figure 1.3 illustrates the broad structural setting of the Anatolia region of Turkey. The Çöpler project area is located between Divriği and Ovacık.

Figure 1.3 Structural Setting of Anatolia



Anagold, 2020

The gold, silver, and copper mineralisation of economic interest at Çöpler occurs in a porphyry-related epithermal environment, with most of the gold mineralisation concentrated in six distinct areas in the deposit: Main, Main West, Main East, Manganese, Marble, and West. The mineralisation is considered to be related to fluids associated with diorite intrusions at depth and generally manifests as three closely related mineralisation styles across the six areas:

- Low-Grade Porphyry Vein Mineralisation.
- Intermediate Sulfidation Epithermal Mineralisation.
- Iron Skarn and Carbonate Replacement Mineralisation.

Oxidation of hypogene mineralisation has resulted in the formation of gossans, massive manganese oxide, and goethitic / jarositic assemblages hosting fine-grained free gold. The oxidised cap is underlain by primary and secondary sulfide mineralisation. Çöpler is a geologically complex system due to structural disturbance and multiple-stage diorite intrusions. The initial mineralisation concept model, based on geochemistry of an epithermal system overlying a copper–gold porphyry dome, continues to hold true with current modelling.

1.6 Exploration

Exploration completed since Anatolia's involvement in the Çöpler project commenced in 1998 has included:

- geological and reconnaissance mapping,
- rock chip, grab, soil, channel, and stream sediment geochemical sampling,
- ground geophysical surveys including ground magnetic, complex resistivity / induced polarisation (IP), time domain IP, and controlled source audio-frequency magneto-telluric (CSAMT) surveys,
- a regional helicopter-borne geophysical survey,
- reverse circulation (RC) and diamond core (DD) drilling programmes, and
- acquisition of satellite imagery.

Other related work has included:

- mining technical studies,
- geotechnical and hydrogeological studies,
- environmental and social baseline studies,
- studies in support of project permitting,
- metallurgical testwork and metallurgical studies, and
- condemnation evaluations.

The principal exploration technique at the Çöpler project has been RC and DD drilling, conducted in several campaigns starting in 2000. Initially, exploration was directed at evaluating the economic potential of the near-surface oxide mineralisation for the recovery of gold by either heap leaching or conventional milling techniques.

1.7 Drilling

Drilling at the Çöpler deposit commenced in 2000, and since that time a total of 2,635 holes have been drilled for 373,561.9 m. A total of 68 DD holes have been completed in 2021 (18,491.8 m).

Step-out drilling at the Çöpler deposit has defined most of the lateral boundaries of the mineralisation. There has been additional development drilling, as well as condemnation drilling of areas planned for infrastructure during the last few years.

- Infill drilling programmes have been conducted since 2007 to improve confidence in the short-term mine planning.
- Drilling in 2014 focused on mineralisation confirmation with a twin-hole programme. The Çöpler deposit continues to be tested using RC and DD drilling as production proceeds.
- Development drilling continued in 2015 by improving sample coverage at depth in the Manganese Zone and along structural boundaries in the Main Zone. In addition to the drilling of in situ mineralisation, a stockpile drilling programme began in December 2015 to confirm sulfide stockpile ore grade, grade distribution, and mineralogy.
- Drilling in 2016–2020 mainly focused on target generation to supplement the amount of oxide material in production. This was focused on the Main Zone, the West pit, and the Saddle areas. These drilling programmes aimed to test continuation of the main gold-bearing structures based on a re-interpretation of the Çöpler structural and mineralisation settings. In-pit drilling campaigns continue.
- Drilling at the Çöpler deposit in 2020–2021 focused on confirmation of copper mineralisation. A total of 199 DD holes were drilled for 41,521.7 m in this period.

Drilling at Çakmaktepe commenced in 2012 and has resulted in the definition of three distinct mineralised zones: East, Central, and North. As production proceeded within the Çakmaktepe Central and East pits, additional targets were generated to provide push-back options around the pit design. A total of 136 DD holes have been completed since 2019 to test for continuation of the Çakmaktepe mineralisation to the north and the east.

Since the initial discovery of mineralisation at Ardich, Anagold has undertaken several drilling programmes to better-define the geological model and to attempt to increase resource inventories. Anagold has completed 531 DD holes for 111,004.35 m at Ardich from late-2017 to 31 December 2021, including holes for metallurgical testing and hydrogeological studies. Drillholes AR01–AR427 are included in an updated geological model for Ardich, developed in late-2021.

Drilling at Bayramdere commenced in 2007 as part of the near-mine exploration strategy. Since that time 120 holes have been drilled at Bayramdere for a total of 10,734.2 m.

1.8 Sampling Method, Approach and Analyses

From 2004 through late-2012, drillhole samples were prepared at ALS İzmir, Turkey (ALS İzmir) and analysed at ALS Vancouver, Canada (ALS Vancouver), (collectively ALS Global).

From late-2012 through 2014, samples were prepared and analysed at ALS İzmir.

In 2015, samples were prepared and analysed at the SGS laboratory in Ankara, Turkey (SGS).

From 2015 to current, ALS İzmir is being used as the main laboratory and samples are being prepared and analysed there. Umpire analysis was completed by ACME Mineral Laboratories (ACME) in Ankara, Turkey.

ALS İzmir has ISO 9001:2008 certification, and ALS Vancouver is ISO/IEC 17025:2005 accredited for precious and base metal assay methods. SGS is certified to ISO 9001:2008 and OHSAS 18001. ACME is part of the Bureau Veritas (BV) group, globally certified to ISO9001:2008.

ALS Global and SGS are specialist analytical testing service companies; both are independent of Anagold.

Sampling and quality assurance and quality control (QA/QC) programmes have been in place for all RC and DD drilling conducted since the first drill programme. The QA/QC programme is currently still in use, although the insertion rates have been modified over time.

Anagold operates an on-site laboratory at Çöpler for assay of production samples. The on-site laboratory is certified to ISO 17025:2017 but is not independent. It is primarily used in grade control.

1.9 Data Verification

Data verification procedures are well-established at the project. Routine ongoing checking of all data is undertaken prior to being uploaded to the database. This is followed by campaign-based independent data verification audits at milestone stages throughout data collection programmes.

For drillhole data, verification includes the checking of Topcon differential global positioning system (DGPS) collar coordinates relative to topographic surveys, checking of down-hole surveys relative to adjacent readings and planned dip and azimuth of the hole, checking logged data entries to ensure they are consistent with log key sheets, cross-checking a subset of assay data with the original laboratory reports, and submission of and review of QA/QC data.

The QA/QC programme has historically consisted of a combination of QA/QC sample types that are designed to monitor different aspects of the sample preparation and assaying process: Blanks are routinely inserted in order to identify the presence of contamination through the sample preparation process; a variety of CRM standards are routinely inserted in order to monitor and measure the accuracy of the assay laboratory results over time; Field duplicates are routinely inserted as a means of monitoring and assessing sample homogeneity and inherent grade variability and to enable the determination of bias and precision between sample pairs; laboratory duplicates are inserted as a means of testing the precision of the laboratory measurements; and inter-laboratory pulp duplicates are submitted to alternative independent laboratory to assess for bias or drift. The rate of submission has been modified over time but is currently 3%–5% for blanks, CRMs, and duplicates, and 5%–10% for field duplicates.

None of the verification programmes have identified material issues with the supporting data.

1.10 Metallurgical Testwork

1.10.1 Oxide Testwork

The heap leaching facilities were commissioned at the Çöpler mine site in late-2010 and have operated continuously since that time. Oxide heap leach operations were continuing at the CDMP21TR effective date.

Metallurgical testwork on Çakmaktepe oxide material for heap leaching has been undertaken at the on-site metallurgical laboratory, initially under the supervision of Kappes, Cassiday & Associates. The initial testwork in 2015 undertook bottle roll and column leach tests. The results are comparable with the Çöpler oxide ore, with similar behaviour and leach kinetics. Subsequently, Çakmaktepe oxide ore has been heap leached along with Çöpler oxide ore. Oxide column testwork on oxide ore continues at the on-site laboratory.

Metallurgical testwork on Ardich oxide material for heap leaching has been undertaken at McClelland laboratories and supervised by Metallurgium. An initial testwork programme, including bottle roll and column leach, was carried out in 2019. This initial programme identified two distinct domains with respect to gold recovery based on sulfur content: <1% and 1%–2%. The column testwork results indicated that the listwanite, dolomite, and jasperoid lithologies have physical properties amenable to heap leaching. This initial test programme was followed up with further testwork in 2020.

Analysis of the results of the metallurgical testwork and a review of the existing recovery models for use in economic analysis were undertaken in 2020. This was done for the oxide and sulfide processing, including the flotation circuit. The resulting recoveries have been used in the economic analysis for the CDMP21TR.

Oxide gold recoveries vary by lithology for Çöpler in the range 62.3%–78.4%, at Çakmaktepe the range is 61%–80%. At Ardich the testwork suggest recoveries will vary in the range 40%–73%. The average oxide recovery in the Reserve Case is 61%.

1.10.2 Sulfide Testwork

The sulfide process plant commenced commissioning in Q4'18. The plant consists principally of a pressure oxidation (POX) leach followed by a cyanide leach to recover gold.

Significant testwork had been conducted on sulfide ores prior to commissioning of the sulfide plant, with pilot plant testwork campaigns and a significant number of batch variability tests on POX / cyanide leach completed.

Whilst a POX / cyanide leach circuit was implemented, significant work had also been undertaken on flotation of the gold-bearing sulfides as a process route, although ultimately this option was not selected for development. Flotation of a partial stream of the plant feed was considered to maximise the available capacity of the plant, including the POX autoclave and available oxygen supply. Further flotation testwork demonstrated that the addition of a small flotation plant into the existing sulfide process route would allow optimisation and maximisation of already installed capacities.

The testwork indicates that sulfur recovery through flotation is estimated to be 75% to concentrate with a corresponding 55% gold recovery. Flotation tails gold recovery is estimated at 43%.

The current determination of POX gold recovery is based on assessment of results for the pilot testwork programmes undertaken prior to commencement of operations and benchmarked with the existing operating data. An equation has been derived to calculate gold recovery by material type for all ore that is subject to POX; this includes direct POX feed and flotation concentrate. The Reserve Case average sulfide gold recovery is 91%.

1.11 Mineral Resources

1.11.1 Resource Modelling

1.11.1.1 Çöpler Deposit

The Çöpler deposit includes four mine areas: Main, Manganese, Marble, and West. The current Çöpler resource model, which was constructed by Anagold personnel, was completed in February 2016.

The cut-off date for the drillholes database was 15 July 2015. The data extract contained 1,957 drillholes with a total of 297,798.2 m of drilling. Of this, a total of 1,880 drillholes have collar coordinates within the extents used to construct the resource model. In general, the drillhole spacing ranged from 5–60 m, averaging approximately 20 m. Most drillholes are either vertical or inclined at 60°.

Wireframes were constructed for the four main geological units: diorite, metasediment, marble, and manganese-rich diorite. Drillhole data and surface mapping were developed into 3D solids that represent the major rock types using implicit modelling techniques. This process included generating contact surfaces used to define the division boundaries that represent the geological faults and lithological contacts.

The resource estimation method at Çöpler was developed to address the variable nature of the gold mineralisation while honouring the bi-modal distribution of the sulfur mineralisation that is critical for mine planning (material with a total sulfur grade <2% is sent to the heap leach while material with total sulfur grade ⩾2% is sent to the sulfide stockpile for eventual processing at the POX plant). Since no obvious correlations were observed between gold and total sulfur, gold and sulfur were domained and estimated separately. Gold showed little correlation with lithology and was domained by mining areas to reflect the different trends of the mineralisation that commonly follow structures and/or lithological contacts. Due to the strong correlation between sulfur content and lithology, sulfur was domained by lithology. However, since each lithology may contain <2% S and ⩾2% S material, each lithology was additionally separated into <2% S and ⩾2% S sub-domains.

Probability assigned constrained kriging (PACK) was used to estimate the gold content of the mineralisation within an expanded mineralised wireframe. A probabilistic envelope was generated within the expanded gold shape to define the limits of the economic mineralisation. The wireframe and probabilistic envelope were used to prevent potentially economic assays from being 'smeared' into non-economic zones, and conversely to restrict waste assays from diluting the potentially economic mineralisation. Two PACK cell models were constructed for gold. The first (low-grade gold) model was applied to <2% S material that can be processed by heap leaching, and the second (high-grade gold) model was later applied to ⩾2% S material to be processed by the POX plant.

Once constructed, the gold models were calibrated to historical production data, categorised by sulfur content (<2% S and ⩾2% S), and mining area. Estimates were classified into Mineral Resource categories based on drillhole density and data quality.

Density values were assigned to the cell model based on lithological domain and depth below the surface.

1.11.1.2 Çakmaktepe Deposit

The Çakmaktepe deposit is located 6 km east of the current Çöpler pit and includes four areas: North, Central, East, and South-east. The current Çakmaktepe resource model, which was constructed by Anagold personnel, was completed in February 2020.

The drillhole dataset used to develop the February 2020 resource model contained a total of 1,109 holes with a drilling date range of September 2007–October 2019. The total drilled metres input into the modelling was 119,001.1 m. Original sample lengths are predominately 1 m in length with some 2 m sampling across areas presumed to be waste. The mean sample length was 1.02 m. The shortest interval was 0.1 m with maximum length 3.1 m. Composited samples 5 m in length were used for statistical analysis, construction of interpretation boundaries, and grade estimation.

Mineralisation at Çakmaktepe follows structural controls and designated lithological contact orientations. Mineralised zones often incorporate multiple lithological units along the boundary rather than being hosted by a single rock type. For this reason, grade shells were constructed for gold and copper to allow estimation concordant with the mineralised zones instead of being controlled by samples residing within a single lithological unit. Mineralised trends were honoured in 3D with no specific grade cut-off used to bound the mineralised shapes. The resulting shapes for gold and copper are lenticular with thicknesses ranging from 5–40 m. On average, thicknesses are of the order of 6 m.

Sulfur grades correlate with lithological units: higher sulfur values are associated with diorite and metasediment, and lower sulfur values are in association with gossan, jasperoid, ophiolite, and marble.

A single geological cell model with 5 m x 5 m x 5 m parent cells was constructed to include the four deposit areas. Gold, silver, copper, sulfur, and carbon were estimated using inverse distance interpolation (ID) weighted to the power of three (ID3) and 5 m drillhole composites. Gold, copper, and silver were estimated using grade shells as hard boundaries. Sulfur and carbon estimates were constrained by modelled lithological units. All grade shell boundaries for metal estimates were treated as hard. Domains were treated as soft boundaries allowing the selection of samples from nearby domains.

Density values were assigned to the cell model based on lithological domain.

1.11.1.3 Ardich Deposit

The Ardich deposit is located 1.5 km north of Çakmaktepe and includes two areas: Main and East. The current Ardich resource model, which was constructed by OreWin, was completed in 2021.

The drillhole dataset used to develop the January 2022 resource model contained a total of 427 diamond core drillholes with a drilling date range of September 2017–May 2021. The total drilled metres for this Ardich dataset was 87,038.25 m. Original sample lengths are predominately 1 m (77.5% of the samples). The shortest assayed interval was 0.2 m, the maximum length 3.8 m, and the mean sample length was 1.19 m. Samples were composited to 1 m length for use in statistical analysis, construction of interpretation boundaries, and grade estimation.

The Ardich Mineral Resource estimate was based on a 3D geological solids model developed within constraining fault blocks. High-angle faults cross-cut the deposit creating rotated structural blocks that have moved up and down relative to each other. There are 25 distinct fault block domains in the 2021 model.

The main lithological units: ophiolite, listwanite, dolomite, jasperoid, and cataclasite, are disrupted by the faults. Owing to the offsets at fault boundaries and the variable thicknesses of lithologies from one fault block to the next, the lithological interpretations have been completed separately for each fault block.

Discrete domains for grade estimation are defined by the fault block and lithology interpretations. As the amount of drill data increased, the understanding of the structural and lithological domains has developed such that a total of 221 unique fault block / lithology domains exist in the 2021 model.

Gold distribution is related to the lithological contact zones and structural intersections. These zones tend to be narrow and localised. Mineralised trends generally follow the orientations of the structural features, further nuanced at the lithological contacts as they change within each of the fault blocks. Control of the gold estimation in the model is accomplished with the use of the fault block and lithology domains as hard boundaries to (a) limit the samples informing estimation in each lithological unit to only those of the same fault block, and (b) to orientate local search neighbourhoods within each domain (dynamic anisotropy). Unlike previous Ardich models, grade shells were not used to constrain estimation in the 2021 model.

A cell model with 10 m x 10 m x 5 m parent cells was constructed to cover the entire Ardich deposit. Sub-celling to 2.5 m x 2.5 m x 1.25 m was permitted to honour interpreted boundaries. Further sub-celling to a minimum of 0.25 m was permitted at the topographic surface. Estimation of a suite of 13 grades (including Au with and without top cuts) and density was undertaken using ordinary kriging. A nearest neighbour estimation of Au was completed for validation purposes.

1.11.1.4 Bayramdere Deposit

The Bayramdere deposit is located approximately 6.3 km east of the Çöpler mine and 5 km south-east of Iliç. It is within the Kartaltepe Mining Licence 7083. This licence is an operational licence and is 50% SSR-owned.

The Bayramdere mineralisation has an overall strike length of approximately 300 m. Mineralisation is localised within three stacked, shallow-dipping lodes that vary in depth between 30–40 m below topography. Mineralisation appears to be open to the east and south.

A resource model for Bayramdere was completed in 2016. Separate mineralisation domains were created for gold, silver, copper, and sulfur. In the creation of mineralised domains, a minimum mining width of 2.5 m was used based on anticipated open pit mining methods. Grade estimation was limited to the interpreted domains. Outside the mineralised domains a 'mineralised waste' estimate was completed. Lithological domains were used for estimates outside of the mineralisation domains. Ordinary kriging was used to estimate gold, silver, and copper mineralisation into parent cells of 10 m x 10 m x 5 m size with sub-celling permitted to 2 m x 2 m x 1 m to better honour the domain boundaries.

Density was assigned as a default for each of the mineralisation and lithological domains.

Although a small deposit, Bayramdere is relatively high-grade and can support a high-stripping ratio to access mineralisation.

1.11.2 Reasonable Prospects for Eventual Economic Extraction

All Mineral Resources in the CDMP21TR were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold, or as otherwise specified. The Initial Assessment has been prepared to demonstrate economic potential of the Mineral Resources at the Çöpler Deposit. The Initial Assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have modifying factors applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that this economic assessment will be realised. The Initial Assessment Case is described in Section 24 of the CDMP21TR.

1.11.3 Mineral Resources Estimates

Mineral Resources and Mineral Reserves in the CDMP21TR meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). Mineral Resources were estimated by Sharron Sylvester BSc (Geology), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology. Mineral Resources are presented on a project basis and have an effective date of 31 December 2021.

Mineral Resources are reported exclusive of Mineral Reserves and have been summarised by project, resource classification, and oxidation state in Table 1.3.

Table 1.4 shows the cut-off values, metallurgical recoveries, and SSR ownership percentage associated with the Mineral Resources.

Table 1.3 Summary of CDMP21TR Mineral Resources Estimates Exclusive of Mineral Reserves (as at 31 December 2021) Based on $1,750/oz Gold Price

Mineral Resource Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine – Oxide							
Measured	81	1.39	4.67	0.16	4	12	281
Indicated	27,173	0.84	2.30	0.16	737	2,012	97,057
Measured + Indicated	27,254	0.84	2.31	0.16	740	2,024	97,339
Inferred	35,021	0.90	6.87	0.13	1,016	7,741	97,941
Çöpler Mine – Sulfide							
Measured	151	0.83	3.72	0.18	4	18	590
Indicated	47,084	1.06	3.66	0.19	1,608	5,535	198,365
Measured + Indicated	47,235	1.06	3.66	0.19	1,612	5,553	198,955
Inferred	49,798	1.24	13.60	0.17	1,982	21,773	181,890
Çakmaktepe – Oxide							
Measured	–	–	–	–	–	–	–
Indicated	3,341	1.55	8.33	–	167	894	–
Measured + Indicated	3,341	1.55	8.33	–	167	894	–
Inferred	1,205	0.85	4.04	–	33	157	–
Ardich – Oxide							
Measured	2,840	1.67	3.99	0.02	153	364	1,031
Indicated	9,794	1.01	2.74	0.00	317	861	410
Measured + Indicated	12,634	1.16	3.02	0.01	469	1,226	1,442
Inferred	13,896	1.27	3.47	0.02	570	1,550	5,181
Ardich – Sulfide (Incl. sulfide and sulfide-with-Cu)							
Measured	234	5.76	8.25	0.04	43	62	215
Indicated	1,410	2.07	3.80	0.03	94	172	900
Measured + Indicated	1,645	2.59	4.44	0.03	137	235	1,115
Inferred	3,226	2.64	4.53	0.01	274	470	576
Bayramdere – Oxide							
Measured	–	–	–	–	–	–	–
Indicated	145	2.34	20.82	–	11	97	–
Measured + Indicated	145	2.34	20.82	–	11	97	–
Inferred	8	2.17	19.95	–	1	5	–
CDMP21 Mineral Resources – Oxide Subtotal							
Measured	2,920	1.67	4.01	0.02	156	376	1,313
Indicated	40,454	0.95	2.97	0.11	1,231	3,865	97,467
Measured + Indicated	43,374	0.99	3.04	0.10	1,387	4,241	98,780
Inferred	50,130	1.00	5.86	0.09	1,619	9,453	103,122
CDMP21 Mineral Resources – Sulfide Subtotal							
Measured	386	3.82	6.47	0.09	47	80	805
Indicated	48,494	1.09	3.66	0.19	1,702	5,707	199,265
Measured + Indicated	48,880	1.11	3.68	0.19	1,749	5,787	200,071
Inferred	53,024	1.32	13.05	0.16	2,256	22,243	182,465
CDMP21 MINERAL RESOURCES – OVERALL TOTAL (Exclusive of Mineral Reserves)							
Measured	3,306	1.92	4.30	0.03	204	457	2,118
Indicated	88,948	1.03	3.35	0.15	2,933	9,572	296,733
Measured + Indicated	92,254	1.06	3.38	0.15	3,136	10,029	298,851
Inferred	103,154	1.17	9.56	0.13	3,875	31,695	285,587

1. Mineral Resources are reported based on 31 December 2021 topography surface.
2. Mineral Resources are reported exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis. Çöpler Mineral Resources are located on ground held 80% by SSR, Çakmaktepe and Bayramdere Mineral Resources are located on ground held 50% by SSR, and approximately 96% of Ardich Mineral Resources are located on ground held 80% by SSR, with the remainder located on ground 50% held by SSR.
4. Oxide definitions: At Çöpler: oxide is defined as material <2% total sulfur and sulfide material is ⩾2% total sulfur. At Ardich and Çakmaktepe, oxide is comprised of low-sulfur (LS) oxide (<1% total sulfur) and high-sulfur oxide (⩾1% and <2% total sulfur). At Bayramdere: oxide is defined as material <2% total sulfur.
5. Sulfide definitions: At Ardich, sulfide is comprised of standard sulfide material (⩾2% total sulfur) and sulfide-with-Cu material (sulfide with Cu>0.10%). There is no sulfide material at Çakmaktepe or Bayramdere.
6. At Çöpler and Ardich: sulfide cut-off uses an NSR value in $/t based on gold price $1,750/oz, silver price $22.00/oz Ag and copper price $3.95/lb with allowances for payability, deductions, transport, and royalties.
7. The point of reference for Mineral Resources is the point of feed into the processing facility.
8. All Mineral Resources in the CDMP21TR were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells ($1,400/oz for gold and $19/oz for silver for Bayramdere, and $1,750/oz for gold, $22/oz for silver for all other projects).
9. Totals may vary due to rounding.

Table 1.4 Summary of Cut-off Values, Metallurgical Recoveries, and SSR Ownership of CDMP21TR Mineral Resources Estimates Exclusive of Mineral Reserves (as at 31 December 2021) Based on Gold Price $1,750/oz, Silver Price $22.00/oz Ag and Copper Price $3.95/lb

Mineral Resource Classification	Tonnage (kt)	Grades			Cut-off Value/s	Metallurgical Recovery (%)	SSR Ownership (%)
		Au (g/t)	Ag (g/t)	Cu (%)			
Çöpler Mine – Oxide							
Measured	81	1.39	4.67	0.16	0.19–0.24 g/t Au	62.3–78.4	80
Indicated	27,173	0.84	2.30	0.16			
Measured + Indicated	27,254	0.84	2.31	0.16			
Inferred	35,021	0.90	6.87	0.13			
Çöpler Mine – Sulfide							
Measured	151	0.83	3.72	0.18	$34.88/t NSR or >0.10% Cu and $7.68/t NSR	Au 55–91 Ag 10–45 Cu 84–98	80
Indicated	47,084	1.06	3.66	0.19			
Measured + Indicated	47,235	1.06	3.66	0.19			
Inferred	49,798	1.24	13.60	0.17			
Çakmaktepe – Oxide							
Measured	–	–	–	–	0.36–0.76 g/t Au	38.0–80.0	50
Indicated	3,341	1.55	8.33	–			
Measured + Indicated	3,341	1.55	8.33	–			
Inferred	1,205	0.85	4.04	–			
Ardich – Oxide							
Measured	2,840	1.67	3.99	0.02	0.23–0.41 g/t Au	40.0–73.0	75
Indicated	9,794	1.01	2.74	0.00			76
Measured + Indicated	12,634	1.16	3.02	0.01			75
Inferred	13,896	1.27	3.47	0.02			65
Ardich – Sulfide (Incl. sulfide and sulfide-with-Cu)							
Measured	234	5.76	8.25	0.04	$36.25/t NSR or >0.10% Cu and $9.05/t NSR	Au 55–91 Ag 10–45 Cu 84–98	78
Indicated	1,410	2.07	3.80	0.03			71
Measured + Indicated	1,645	2.59	4.44	0.03			75
Inferred	3,226	2.64	4.53	0.01			71
Bayramdere – Oxide							
Measured	–	–	–	–	0.35–0.50 g/t Au	75	50
Indicated	145	2.34	20.82	–			
Measured + Indicated	145	2.34	20.82	–			
Inferred	8	2.17	19.95	–			
CDMP21 Mineral Resources – Oxide Subtotal							
Measured	2,920	1.67	4.01	0.02	As Above	As Above	75
Indicated	40,454	0.95	2.97	0.11			75
Measured + Indicated	43,374	0.99	3.04	0.10			75
Inferred	50,130	1.00	5.86	0.09			74
CDMP21 Mineral Resources – Sulfide Subtotal							
Measured	386	3.82	6.47	0.09	As Above	As Above	78
Indicated	48,494	1.09	3.66	0.19			80
Measured + Indicated	48,880	1.11	3.68	0.19			80
Inferred	53,024	1.32	13.05	0.16			79
CDMP21 MINERAL RESOURCES – OVERALL TOTAL (Exclusive of Mineral Reserves)							
Measured	3,306	1.92	4.30	0.03	As Above	As Above	76
Indicated	88,948	1.03	3.35	0.15			77
Measured + Indicated	92,254	1.06	3.38	0.15			77
Inferred	103,154	1.17	9.56	0.13			77

1. Mineral Resources are reported based on 31 December 2021 topography surface.
2. Mineral Resources are reported exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis. SSR Ownership is an average based on location of Mineral Resources (gold) relative to licenses: Çöpler and part of Ardich are on Anagold 80:20 ground on which SSR holds 80% rights, and Çakmaktepe, Bayramdere and the remainder of Ardich are on Kartaltepe 50:50 ground on which SSR holds 50% rights. Totals and Ardich ownership percentages are weighted averages.
4. Oxide definitions: At Çöpler: oxide is defined as material <2% total sulfur and sulfide material is ⩾2% total sulfur. At Ardich and Çakmaktepe, oxide is comprised of low-sulfur (LS) oxide (<1% total sulfur) and high-sulfur oxide (⩾1% and <2% total sulfur). At Bayramdere: oxide is defined as material <2% total sulfur.
5. Sulfide definitions: At Ardich, sulfide is comprised of standard sulfide material (⩾2% total sulfur) and sulfide-with-Cu material (sulfide with Cu>0.10%). There is no sulfide material at Çakmaktepe or Bayramdere.
6. At Çöpler and Ardich: sulfide cut-off uses an NSR value in $/t based on gold price $1,750/oz, silver price $22.00/oz, and copper price $3.95/lb with allowances for payability, deductions, transport, and royalties.
7. The point of reference for Mineral Resources is the point of feed into the processing facility.
8. All Mineral Resources in the CDMP21TR were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells ($1,400/oz for gold and $19/oz for silver for Bayramdere, and $1,750/oz for gold, $22/oz for silver for all other projects).
9. Totals may vary due to rounding.

1.12 Mineral Reserves Estimates

Mineral Resources and Mineral Reserves in the CDMP21TR meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). Mineral Reserves were estimated by Bernard Peters BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director - Mining. Mineral Reserves are presented on a project basis and have an effective date of 31 December 2021.

Mineral Reserves have been summarised by project, reserve classification, and oxidation state in Table 1.5 and in Table 1.6.

Table 1.5 Summary of CDMP21TR Mineral Reserves Estimates (as at 31 December 2021) Based on $1,350/oz Gold Price

Mineral Reserve Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	2,204	1.22	11.17	0.13	87	792	6,304
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	2,204	1.22	11.17	0.13	87	792	6,304
Çöpler Mine – Sulfide							
Proven Mineral Reserve	408	2.02	6.69	–	26	88	–
Probable Mineral Reserve	35,828	2.13	4.96	–	2,455	5,713	–
Probable – Stockpile	12,468	2.25	–	–	900	–	–
Total Mineral Reserve	48,703	2.16	3.70	–	3,382	5,801	–
Çakmaktepe Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	274	1.26	10.91	–	11	96	–
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	285	1.32	10.49	–	12	96	–
Ardich – Oxide Reserve							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	381	464	208
Probable Mineral Reserve	13,305	1.74	1.98	0.01	742	849	2,933
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	20,050	1.74	2.04	0.01	1,124	1,313	3,141
Ardich – Sulfide							
Proven Mineral Reserve	1,871	5.55	10.83	–	334	651	–
Probable Mineral Reserve	2,253	3.13	4.35	–	227	315	–
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	4,124	4.23	7.29	–	560	966	–
CDMP21 Mineral Reserves – Oxide Subtotal							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	381	464	208
Probable Mineral Reserve	15,783	1.66	3.42	0.03	840	1,736	9,237
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	22,539	1.69	3.04	0.02	1,222	2,200	9,445
CDMP21 Mineral Reserves – Sulfide Subtotal							
Proven Mineral Reserve	2,278	4.92	10.09	–	360	739	–
Probable Mineral Reserve	38,081	2.19	4.92	–	2,682	6,028	–
Probable – Stockpile	12,468	2.25	–	–	900	–	–
Total Mineral Reserve	52,827	2.32	3.98	–	3,942	6,768	–
CDMP21 MINERAL RESERVES – OVERALL TOTAL							
Proven Mineral Reserve	9,024	2.55	4.15	0.00	741	1,203	208
Probable Mineral Reserve	53,863	2.03	4.48	0.01	3,522	7,765	9,237
Probable – Stockpile	12,479	2.25	–	–	901	–	–
Total Mineral Reserve	75,366	2.13	3.70	0.01	5,164	8,968	9,445

1. Mineral Reserves are reported based on 31 December 2021 topography surface.
2. The Mineral Reserves were scheduled based on End-of-August 2021topography surface. Small differences between the Mineral Reserve statement and the production schedule may occur.
3. Mineral Reserves are shown on a 100% basis. Çöpler and part of Ardich are on Anagold 80:20 ground on which SSR holds 80% rights, and Çakmaktepe and the remainder of Ardich are on Kartaltepe 50:50 ground on which SSR holds 50% rights.
4. Mineral Reserve cut-offs are based on $1,350/oz gold price; average oxide recoveries are 61% and average sulfide recoveries are 91%.
5. The point of reference for Mineral Reserves is the point of feed into the processing facility.
6. Cut-off values are shown in Table 1.6. All cut-off values include allowance for royalty payable. There are no credits for silver or copper in the cut-off calculations.
7. There is no Çakmaktepe sulfide Mineral Reserve or Bayramdere Mineral Reserve.
8. Economic analysis has been carried out using a long-term gold price of $1,600/oz.
9. Totals may vary due to rounding.

Table 1.6 Summary of Cut-off Values, Metallurgical Recoveries, and SSR Ownership of CDMP21TR Mineral Reserves Estimate (as at 31 December 2021) Based on $1,350/oz Gold Price

Mineral Reserve Classification	Tonnage (kt)	Grades			Cut-off Value/s (g/t Au)	Metallurgical Recovery (%)	SSR Ownership (%)
		Au (g/t)	Ag (g/t)	Cu (%)			
Çöpler Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	2,204	1.22	11.17	0.13	0.47–0.59	62.3–78.4	80
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	2,204	1.22	11.17	0.13	0.47–0.59	62.3–78.4	80
Çöpler Mine – Sulfide							
Proven Mineral Reserve	408	2.02	6.69	–	1.05	85	80
Probable Mineral Reserve	35,828	2.13	4.96	–			
Probable – Stockpile	12,468	2.25	–	–			
Total Mineral Reserve	48,703	2.16	3.70	–			
Çakmaktepe Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	274	1.26	10.91	–	0.52–0.71	14–80	50
Probable – Stockpile	11	2.69	–	–			
Total Mineral Reserve	285	1.32	10.49	–			
Ardich – Oxide							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	0.44–0.80	40–73	77
Probable Mineral Reserve	13,305	1.74	1.98	0.01			
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	20,050	1.74	2.04	0.01	0.44–0.80	40–73	77
Ardich – Sulfide							
Proven Mineral Reserve	1,871	5.55	10.83	–	1.11	83	78
Probable Mineral Reserve	2,253	3.13	4.35	–			72
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	4,124	4.23	7.29	–	1.11	83	75
CDMP21 Mineral Reserves – Oxide Subtotal							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	0.44–0.80	14–80	77
Probable Mineral Reserve	15,783	1.66	3.42	0.03			77
Probable – Stockpile	11	2.69	–	–	0.52–0.71	14–80	50
Total Mineral Reserve	22,539	1.69	3.04	0.02	0.44–0.80	14–80	77
CDMP21 Mineral Reserves – Sulfide Subtotal							
Proven Mineral Reserve	2,278	4.92	10.09	–	1.05–1.11	83–85	78
Probable Mineral Reserve	38,081	2.19	4.92	–			79
Probable – Stockpile	12,468	2.25	–	–			80
Total Mineral Reserve	52,827	2.32	3.98	–			79
CDMP21 MINERAL RESERVES – OVERALL TOTAL							
Proven Mineral Reserve	9,024	2.55	4.15	0.00	0.44–1.11	14–85	77
Probable Mineral Reserve	53,863	2.03	4.48	0.01			79
Probable – Stockpile	12,479	2.25	–	–			80
Total Mineral Reserve	75,366	2.13	3.70	0.01			78

1. Mineral Reserves are reported based on 31 December 2021 topography surface.
2. The Mineral Reserves were scheduled based on End-of-August 2021topography surface. Small differences between the Mineral Reserve statement and the production schedule may occur.
3. Mineral Reserves are shown on a 100% basis. SSR Ownership is an average based on location of Mineral Reserves (gold) relative to licenses: Çöpler and part of Ardich are on Anagold 80:20 ground on which SSR holds 80% rights, and Çakmaktepe and the remainder of Ardich are on Kartaltepe 50:50 ground on which SSR holds 50% rights. Totals and Ardich ownership percentages are weighted averages.
4. Mineral Reserve cut-offs are based on $1,350/oz gold price; average oxide recoveries are 61% and average sulfide recoveries are 91%.
5. The point of reference for Mineral Reserves is the point of feed into the processing facility.
6. All cut-off grades include allowance for royalty payable. There are no credits for silver or copper in the cut-off grade calculations.
7. There is no Çakmaktepe sulfide Mineral Reserve or Bayramdere Mineral Reserve.
8. Economic analysis has been carried out using a long-term gold price of $1,600/oz.
9. Totals may vary due to rounding.

1.13 Mining Method

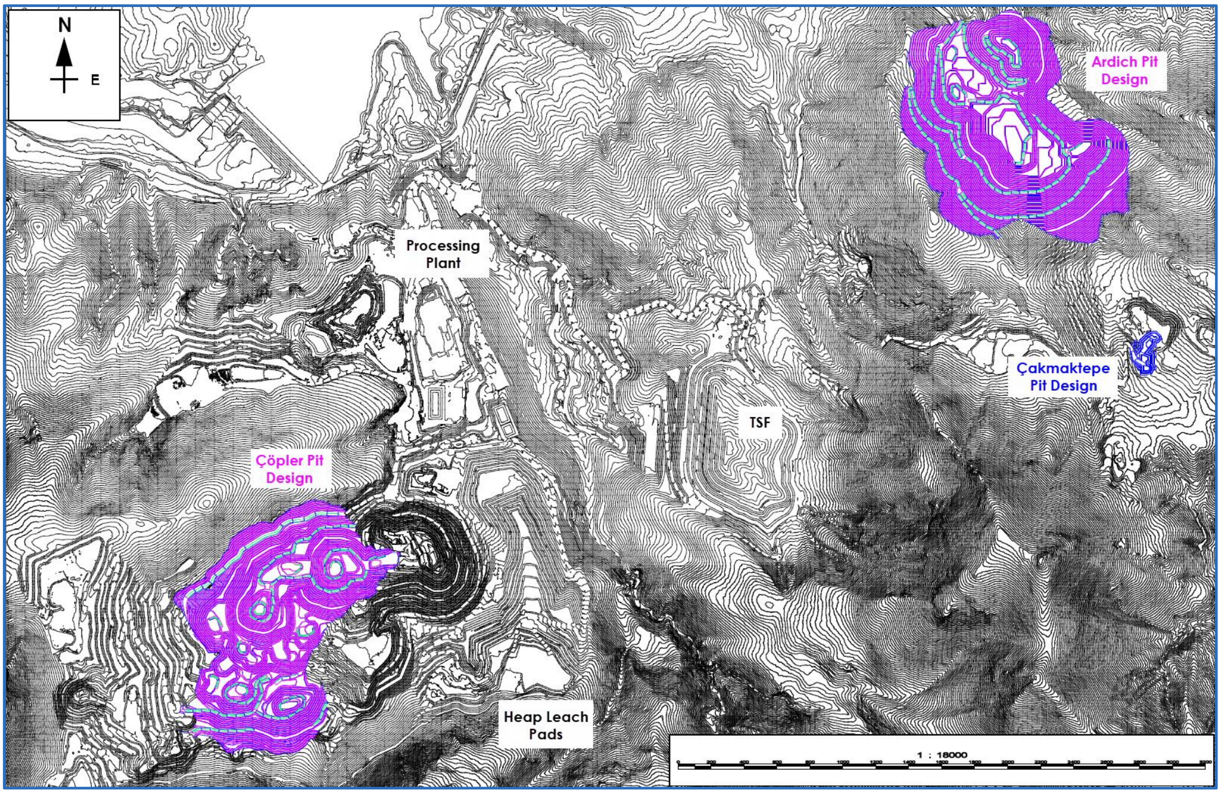
Open pit mining at the Çöpler project is carried out by a mining contractor and managed by Anagold. The mining method is a conventional open pit method with drill and blast and utilising excavators and trucks operating on bench heights of 5 m. The mining contractor provides operators, line supervisors, equipment, and ancillary facilities required for the mining operation. Anagold provides management, technical, mine planning, engineering, and grade control functions for the operation.

Anagold currently operates a sulfide process plant and an oxide heap leach facility. Costs are based on the actual operational costs and the project budget assumptions.

Production schedules and costs have been updated based on current site performance and contracts.

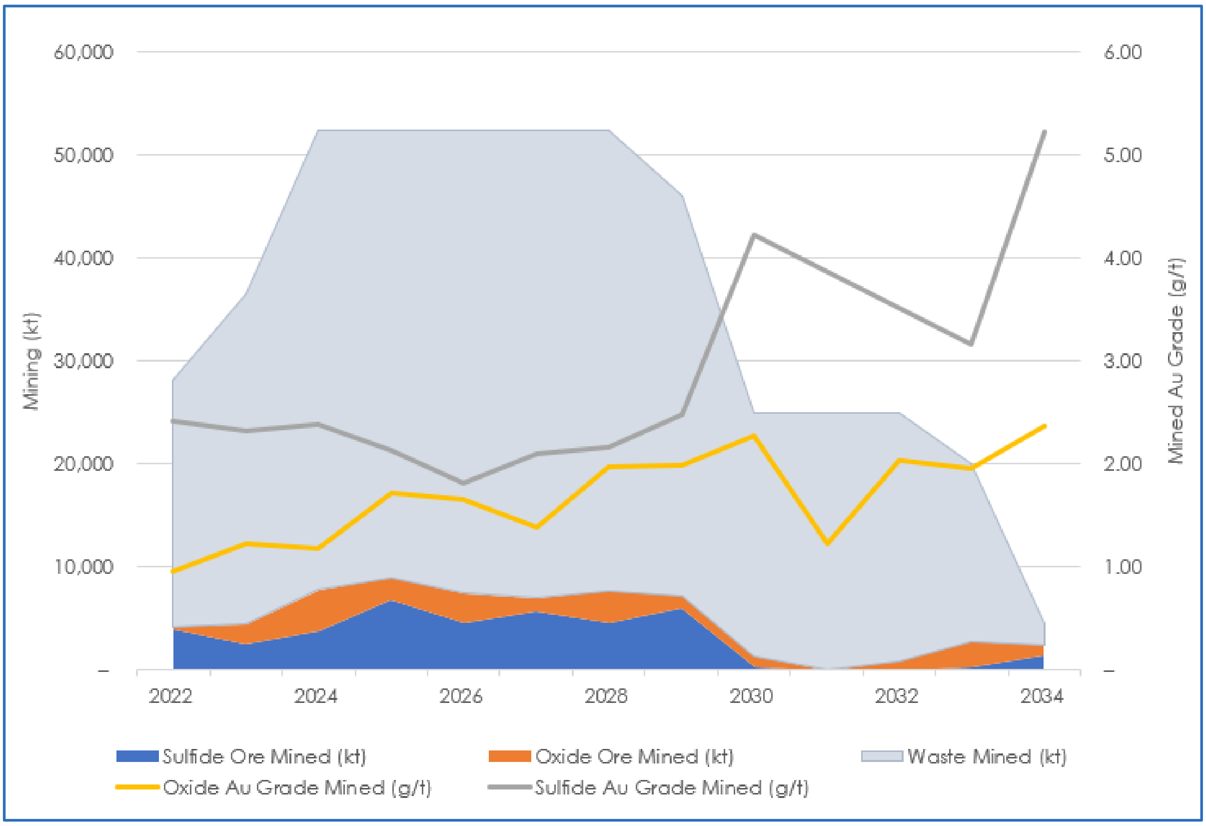
Pit designs for the Çöpler pit were reviewed and updated in 2021. The Ardich pit designs were prepared in 2021 and 2022. The pit designs included in the Reserve Case are shown in Figure 1.4. The Reserve Case mining production is shown in Figure 1.5.

Figure 1.4 CDMP21TR Ultimate Pit Designs



Anagold, 2022

Figure 1.5 CDMP21TR Reserve Case Mining Production



OreWin, 2022

1.14 Recovery Methods

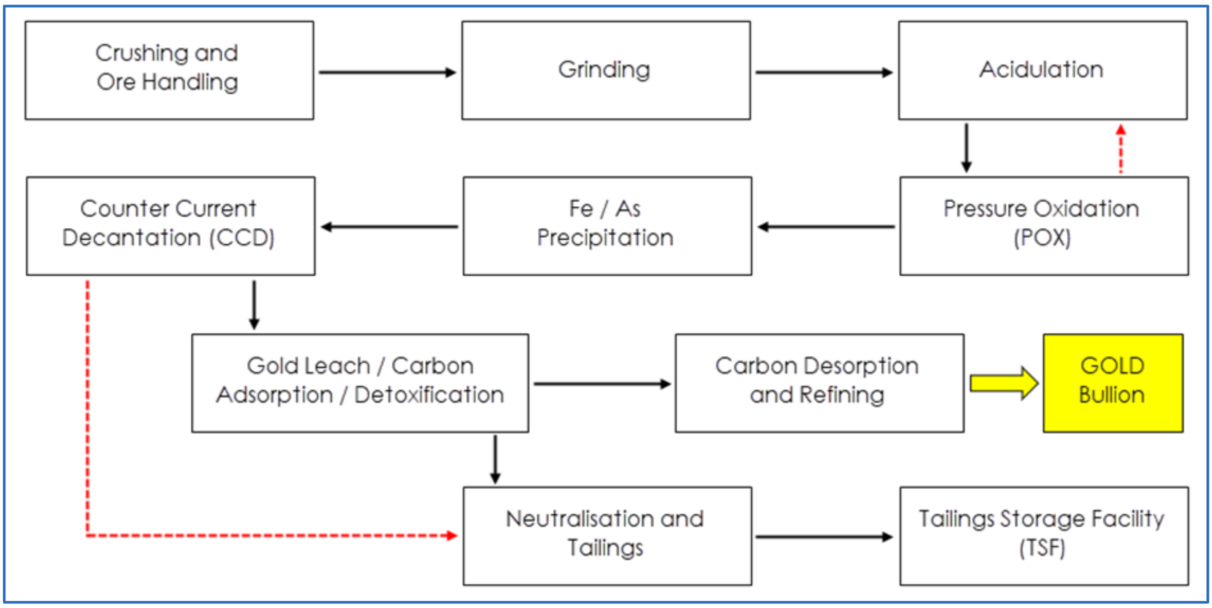
1.14.1 Sulfide Plant

The sulfide plant commenced commissioning in Q4'18. The basic flow sheet is shown in Figure 1.6 and comprises:

- Crushing and ore handling
- Grinding
- Acidulation
- Pressure oxidation
- Iron / arsenic precipitation
- Counter Current Decantation (CCD)
- Gold leach, carbon adsorption, and detoxification
- Carbon desorption and refining
- Neutralisation and tailings
- Tailing Storage Facility (TSF)

The sulfide plant performance from Q4'18 up to Q4'21, including commissioning and ramp-up, has achieved greater-than-design throughputs and approaches design gold recovery for the ore types processed.

Figure 1.6 Process Flow Sheet for Sulfide Plant



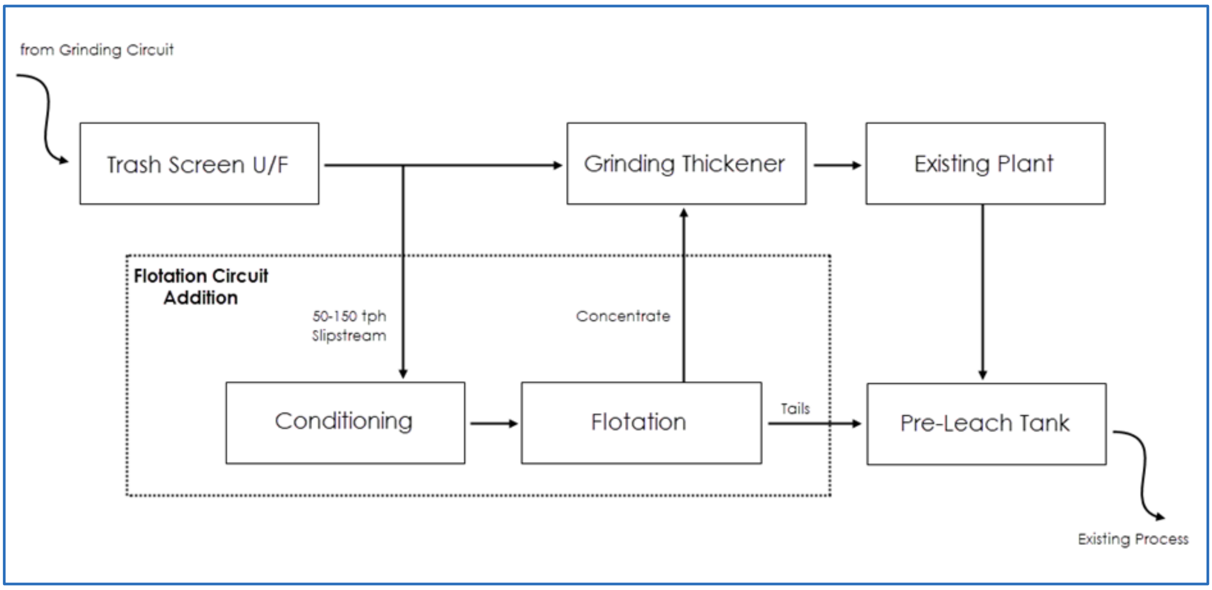
Anagold, 2020

The incorporation of a new flotation circuit in the existing sulfide plant to upgrade sulfide sulfur (SS) to fully utilise POX autoclave oxidation capacity is complete and commissioning commenced in January 2021. This addition to the sulfide plant is incorporated between grinding and acidulation, as shown in Figure 1.7, by taking a bleed / slip stream from the grinding thickener feed, floating gold-bearing sulfides, rejecting acid-consuming carbonates and returning the sulfide concentrate to the grinding thickener to be combined with direct POX feed. The gold not recovered to concentrate that remains in the flotation tails is directed to the gold leach circuit feed to recover this remaining gold, albeit at lower gold recoveries than ore that is treated through the POX autoclave circuit.

This will increase overall plant maximum throughput rate to 400 tonnes per hour (tph), allowing the grinding and POX circuit to operate at their maximum demonstrated capacities. The grinding circuit maximum volumetric flow throughput will increase from an original design limit of 306 tph to 400 tph, fully utilising latent capacity within the crushing and grinding circuit. The flotation plant is designed to operate in the throughput range of 50–150 tph to produce a concentrate that will supplement the feed ore SS to maximise autoclave SS up to 13.75 tph at a maximum autoclave feed rate of 280 tph.

Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph SS to be treated. This oxygen utilisation efficiency along with increased oxygen availability is upside to the CDMP21TR Reserve Case.

Figure 1.7 Flotation Block Flow Diagram



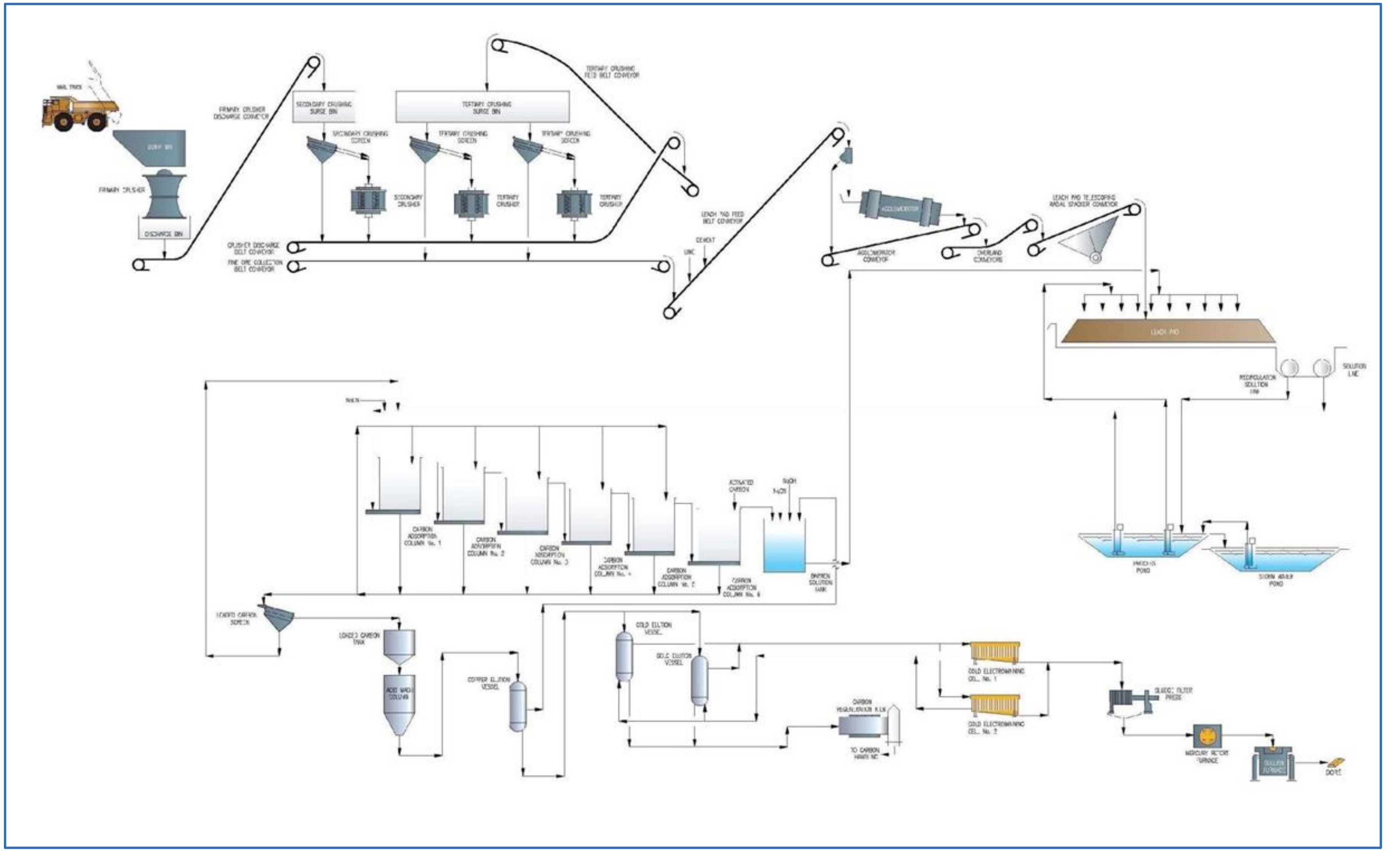
Anagold, 2020

1.14.2 Oxide Ore Heap Leach Processing

In the Reserve Case production is predominantly from sulfide ore. The maximum oxide ore placed in any year is 4.0 Mt for a total production of 22.5 Mt.

The oxide heap leaching and associated facilities were commissioned in the second half of 2010 and initial gold production was achieved in Q4'10. The process was originally designed to treat approximately 6.0 Mtpa of ore by three-stage crushing (primary, secondary, and tertiary) to 80% passing 12.5 mm, agglomeration, and heap leaching on a lined heap leach pad with dilute alkaline sodium cyanide solution. Gold is recovered through a carbon in-column (CIC) system, followed by stripping of metal values from carbon, electrowinning and smelting to yield a doré (containing gold and silver) suitable for sale. Control of copper in leach solutions is undertaken in a sulfidisation, acidification, recovery, and thickening (SART) plant, which also regenerates cyanide. The oxide ore heap leach process flow sheet is shown in Figure 1.8.

Figure 1.8 Process Flow Sheet for Oxide Ore Heap Leach



Anagold, 2021

1.14.3 Project Infrastructure

1.14.3.1 Infrastructure

The facility infrastructure supports the mine and process areas of oxide heap leach and sulfide plant. The existing infrastructure, and the tailings storage and heap leach pad area once the planned expansion is complete, will be sufficient for the current Mineral Reserves. The infrastructure for the addition of flotation to the sulfide plant will be supported by the existing facility infrastructure with some components modified to meet the addition of the flotation circuit.

The current leach pad consists of four phases designed to accommodate approximately 58 Mt of oxide ore heap with a nominal maximum heap height of 100 m above the pad liner. An additional two phases (phase 5 and phase 6), with a capacity of 20 Mt will be added to accommodate the oxide to be mined from Ardich.

The Tailings Storage Facility (TSF) is developed and constructed in stages. The development of TSF 1 includes seven phases. TSF 1 phase 3 construction has been completed and approval for use was granted in October 2021 by the Ministry of Environment, Urbanisation and Climate Change (MoEUCC). Ongoing work in ensuring sufficient long-term capacity for storage of tailings has been undertaken. Studies by Anagold have determined that the effect of the addition of the flotation circuit to the sulfide plant would result in an increase in the solids content and improvement in the final settled density based on an increase in the rate of tailings consolidation.

TSF 1 has sufficient storage capacity (70.8 Mt) to accommodate the CDMP21TR tailings. Scoping level investigations have identified additional TSF sites. An adjacent site, TSF 2, has been the subject of a PFS level study and can provide approximately 20 Mt of net additional tails storage capacity, if required in the future. A detailed design of TSF 2 has been substantially progressed. In November 2021 an application project package was submitted to the MoEUCC. Project design and approval finalisation is expected by Q3'22.

1.15 Market Studies

The markets for gold and silver doré are readily accessed and available to gold producers. Currently, 100% of the gold and silver is delivered to the Istanbul Gold Refinery. Copper precipitate is currently produced from the SART plant and sold into local markets in Turkey. The sulfide plant does not currently include a copper circuit and the analysis of copper recovery at Çöpler has been considered in the Initial Assessment as part of the CDMP21TR.

1.16 Environmental and Permitting

The Çöpler mining and processing operations involve open pit mining from multiple pits, construction of multiple waste rock dumps (WRD) to accommodate mined materials, processing of oxide ores and placement on a heap leach pad, and processing of sulfide ores with placement of tailings in a TSF. These activities and facilities are carried out on treasury, pasture, and forestry lands.

In addition to the direct impacts on the involved lands, the operations impact the surrounding lands and the local communities. Physical impacts may include changes to local surface and groundwater (including potential pollution), air quality impacts particularly from dust, and increased noise and vibration from mining and processing activities.

Operation of the Çöpler mining and processing facilities, and subsequent mining at Çakmaktepe, have been investigated and authorised by means of a series of Environmental Impact Assessments (EIAs), with positive decisions obtained from the Turkish Ministry of Environment, Urbanisation and Climate Change. These EIAs include specific actions designed to address all material impacts of the mining and processing operations. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project.

The original 2008 EIA obtained on 16 April 2008 included three main open pits (Manganese, Marble, and Main), five WRDs, a heap leach pad, a processing plant, and a TSF. The 2008 project description involved only the oxide resources.

The Çöpler mine started its open pit and heap leach operation in 2010 and first gold was poured in December 2010. Additional EIA investigations have been submitted and approved, as required, to support on-going mining and processing operations, including:

- EIA to allow operation of a mobile crushing plant, approved 10 April 2012.
- EIA to allow waste dump capacity expansion, oxide capacity expansion to 23,500 tpd and a SART plant, approved 17 May 2012.
- EIA to allow the sulfide plant and heap leach area expansion, approved 24 December 2014.
- EIA to allow the Çakmaktepe satellite pits expansion, approved 26 January 2017.
- EIA to allow a Çakmaktepe capacity increase, approved 9 August 2018.
- EIA to allow a second capacity expansion at Çöpler, including heap leach pads 5 and 6, TSF expansion, and operation of a flotation plant, approved 7 October 2021.

In addition, pending EIA processes include:

- EIA to allow second capacity increase on the Çakmaktepe EIA to include initial mining from Ardich in the EIA project description file. The EIA project description file was submitted in October 2020 and a Public Hearing was held in November 2020. All public institutions gave positive feedback regarding the report and the approval process is ongoing with the MoEUCC.

Following the EIA positive decisions, additional licences and permits were required to be issued by government agencies consistent with the Turkish governing laws and regulations. These include land access permits (pasture and forestry), operational environmental licences and permits, and workplace opening and operating permits, licences, and certificates.

1.17 Capital and Operating Costs

Capital and operating cost estimates have been developed based on the current project costs, the mine and process designs, and discussions with potential suppliers and contractors. The sulfide growth costs include the capital cost for the flotation circuit. The estimated capital costs are to a feasibility level of accuracy and include a contingency of 10%.

1.18 Capital Costs

Capital costs have been split into growth and sustaining costs. The sustaining costs also include the reclamation costs for closure.

Growth capital costs in the Reserve Case includes costs for:

- Ardich establishment and mine development
- Heap leach phases 5 and 6
- Road relocation, studies, and project management
- Explosives magazine relocation

Sustaining capital in the Reserve Case includes costs for:

- TSF expansion
- Project team
- Technical services
- Administration
- Assay laboratory
- Mining
- IT
- Sulfide and oxide processing
- Environment
- Mineral / lands rights
- Health and safety
- Security
- Supply chain
- Reclamation

The CDMP21TR Reserve Case capital costs in 2022 for heap leach is $7.8M and for POX is $28.53M. Total capital over the life-of-mine (LOM) including reclamation and closure is $626M.

1.19 Operating Costs

Operating costs were estimated based on current site cost performance and contract costs, including actual operational costs for labour, consumables, contracts, and the Anagold budget assumptions. Operating costs have a base date of Q4'21 with no allowance for escalation. LOM average operating costs are shown in Table 1.7.

Table 1.7 Summary of LOM Average Operating Costs

Cost	Total LOM ($M)	5-Year Average per year ($/t ore)	LOM Average per year ($/t ore)
Mining	766	14.98	10.15
Process	2,225	27.79	29.49
Site Support and G&A	473	7.14	6.27
Operating Costs	3,464	49.91	45.91

Mining costs include waste stripping costs

1.20 CDMP21TR Reserve Case

The Reserve Case production includes 22.6 Mt at 1.69 g/t Au oxide ore processed by heap leaching and 52.9 Mt at 2.33 g/t Au processed in the sulfide plant. Total production is 75.4 Mt at 2.14 g/t Au. Total gold production is 4.4 Moz. Mining at the Çöpler pit is completed in 2029 and at Ardich in 2034. Oxide heap leach stacking is completed in in 2034, while sulfide processing will continue from stockpiles until 2042.

The Reserve Case results include:

- After-tax NPV at a 5% real discount rate is $1.73 billion.
- Mine life of 21 years.

An IRR is not reported as the operation is cash positive in each year of the mine plan until closure. The Reserve Case average all-in sustaining cost (AISC) is $966/oz gold. Key results of the Reserve Case economic analysis are shown in Table 1.8.

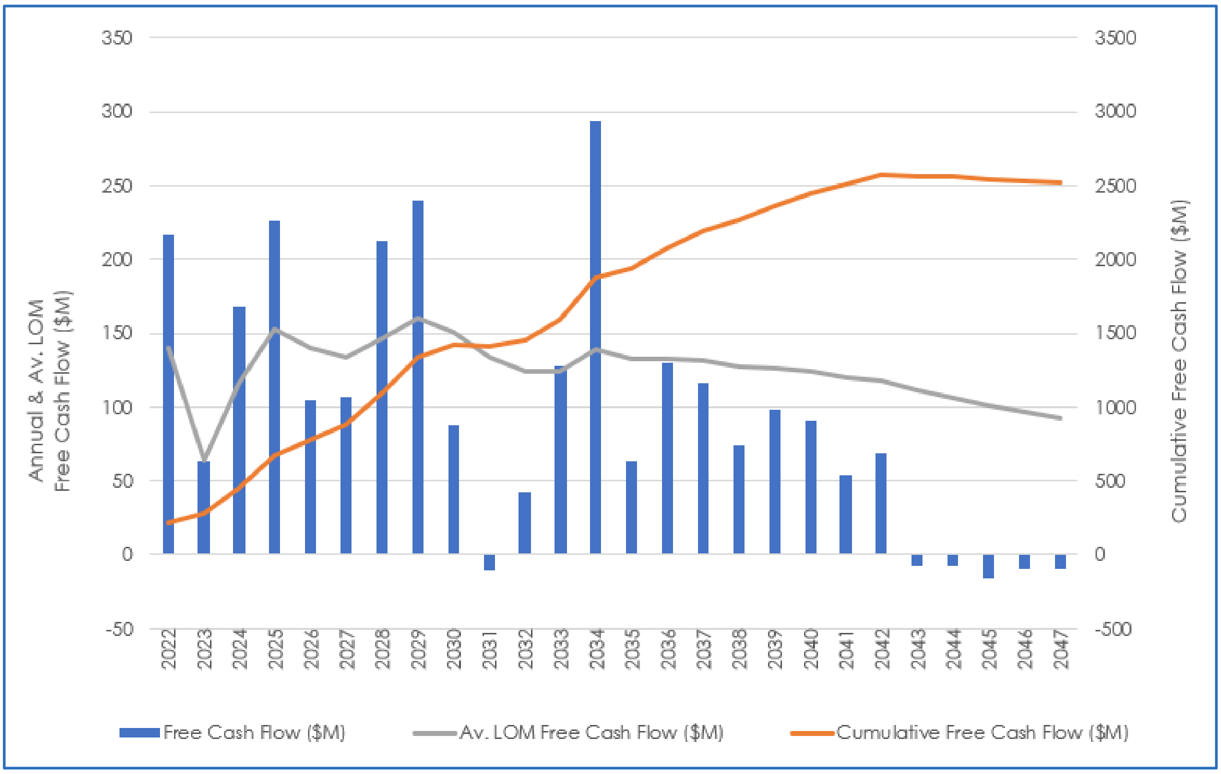
Table 1.8 CDMP21TR Reserve Case Results Summary

Item	Unit	Reserve Case
Oxide Processed		
Heap Leach Quantity	kt	22,557
Au Feed Grade	g/t	1.69
Sulfide Processed		
Quantity Milled	kt	52,892
Au Feed Grade	g/t	2.33
Total Processed		
Processed	kt	75,448
Gold Feed Grade	g/t	2.14
Total Gold Produced		
Oxide – Gold	koz	765
Sulfide – Gold	koz	3,604
Total – Gold	koz	4,369
Oxide – Gold Recovery	%	61
Sulfide – Gold Recovery	%	91
5-Year Annual Average		
Average Gold Produced	kozpa	278
Free Cash Flow	$Mpa	158
Total Cash Costs (CC)	$/oz gold	880
All-in Sustaining Costs (AISC)	$/oz gold	1,071
Key Financial Results		
LOM Total Cash Costs (CC)	$/oz gold	803
LOM All-in Sustaining Costs (AISC)	$/oz gold	966
Site Operating Costs	$/t treated	45.91
After-Tax NPV5%	$M	1,732
Mine Life	years	21

5-Year Annual Average is for the period 1 January 2021 through 31 December 2026

The after-tax cash flow is shown in Figure 1.9. The NPV results for before and after-tax over a range of discount rates is shown in Table 1.9. The sulfide and oxide production profiles are shown in Figure 1.10 and gold production in Figure 1.11. Cash costs are shown in Table 1.10.

Figure 1.9 Reserve Case After-Tax Cash Flow

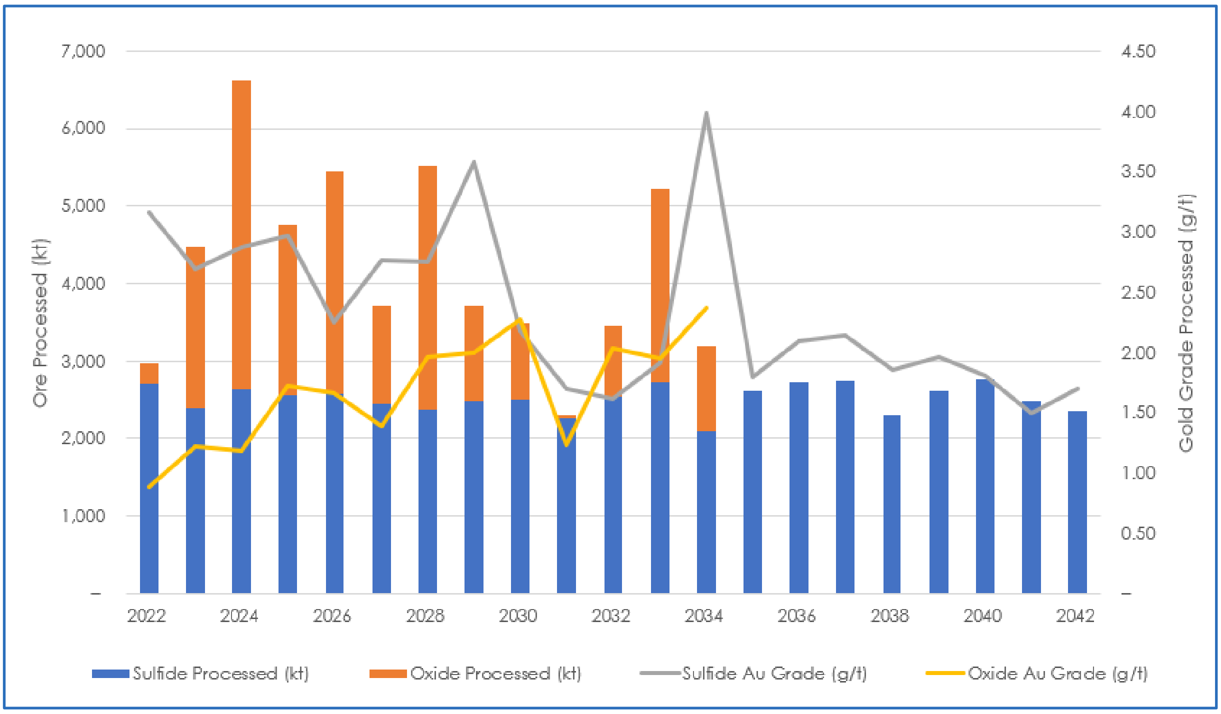


OreWin, 2022

Table 1.9 CDMP21TR Reserve Case Before and After-Tax NPV

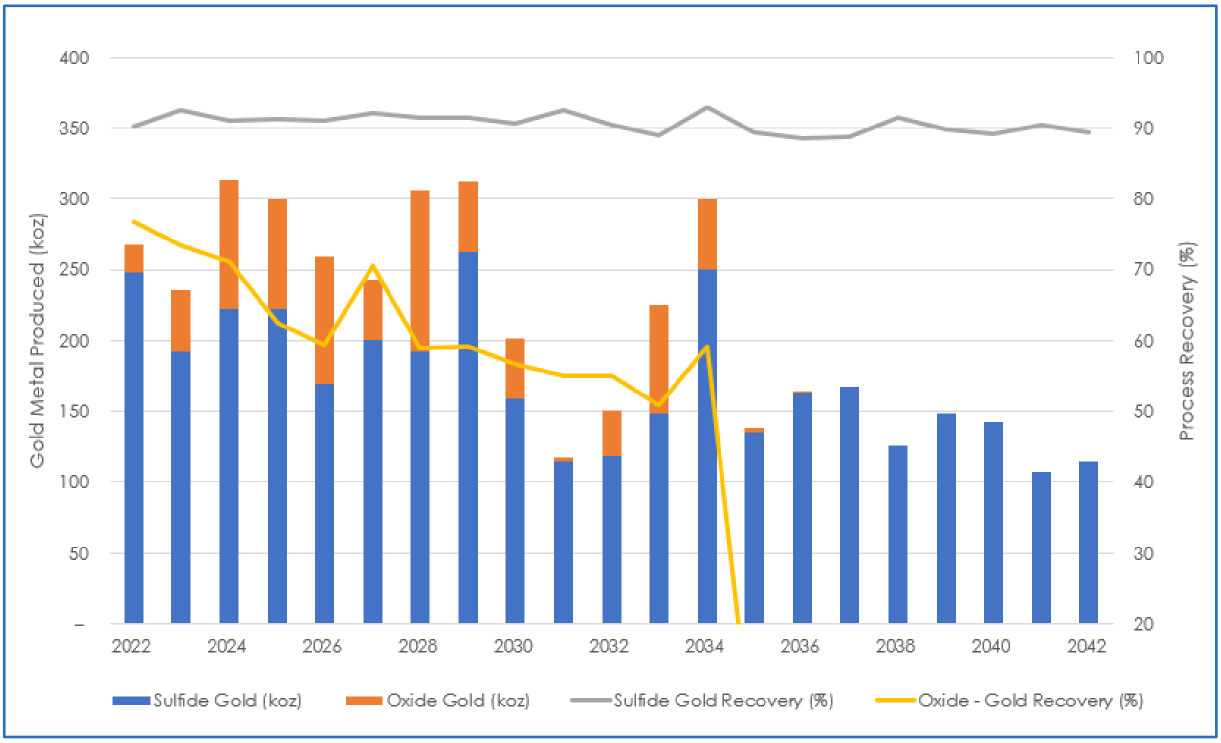
Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	2,729	2,555
5%	1,824	1,732
10%	1,322	1,268
12%	1,185	1,140

Figure 1.10 Reserve Case Processing



OreWin, 2022

Figure 1.11 Reserve Case Gold Production



OreWin, 2022

Table 1.10 CDMP21TR Reserve Case Cash Costs

Description	Units	Reserve Case
Mining and Rehandle	$M	766
Process, Freight, and Refining	$M	2,031
Site Support	$M	393
Royalties	$M	353
Total Production Costs	$M	3,543
Total Cash Costs (CC)	$/oz gold	803
Sustaining Capital	$M	442
Fixed Lease Payments	$M	192
Site G&A	$M	81
Total All-in Sustaining Costs	$M	4,257
All-in Sustaining Costs	$/oz gold	966

Process, Freight, and Refining includes by-product credits and excludes fixed lease costs.
Royalties are calculated in the period incurred and applied to cash flow in the subsequent year.

A financial model was prepared using the Reserve Case production schedule and operating and capital assumptions on an annual basis. The assumptions for taxes and royalties were provided by Anagold. The corporate tax rate in Turkey is 23% in 2022 but will revert to 20% from 2023. The royalty rate for precious metals under Turkish Mining Law is variable and tied to metal prices. As Çöpler ores are processed on-site, the applicable royalty rate is subject to a further 40% reduction for certain qualifying operating costs. The average royalty calculated as a proportion of gross revenue in the Reserve Case is approximately 4.9%.

Metal prices were estimated after analysis of consensus industry metal price forecasts and metal prices used in other comparable studies. The metal prices used for the economic analysis are shown in Table 1.11.

Table 1.11 CDMP21TR Economic Analysis Metal Price Assumptions

Metal Price	Units	2022	2023	2024	2025	Long- Term
Gold	$/oz	1,800	1,740	1,710	1,670	1,600
Silver	$/oz	24.00	23.00	22.00	21.00	21.00
Copper	$/lb	4.00	3.80	3.80	3.80	3.40

The estimates of cash flows have been prepared on a real basis with a base date of Q4'21 and a mid-year discounting is used to calculate NPV. All monetary figures have a base date of Q4'21 with no allowance for escalation and are expressed in US dollars (US$) unless otherwise stated.

1.21 Çöpler Initial Assessment Case

The Initial Assessment Case is a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

The project currently has two processing methods:

- Sulfide process plant
- Heap leach oxide processing facility

The sulfide plant includes the crushing, grinding, flotation and pressure oxidation to produce gold and small amounts of silver. The heap leach facility produces gold and small amounts of silver and copper.

The scenario for the Initial Assessment Case analysis includes additional processing options to recover copper from the sulfide Mineral Resource. The two processing options are:

- Copper Concentrator producing a copper concentrate and a pyrite concentrate.
- Sodium Hydrosulfide (NaSH) copper recovery circuit to be installed in the current sulfide plant.

The copper concentrator would make a copper concentrate for sale to smelters and a pyrite concentrate to be fed into the autoclaves in the sulfide plant. The pyrite concentrate would have a high gold content and also provide sulfur as a source of fuel for the autoclaves. The copper concentrator capacity is 1.8 Mtpa.

The Çöpler Mineral Resource has been selected for the Initial Assessment analysis, as the other Mineral Resources at the project do not have significant amounts of copper.

Implementation of the copper recovery options will require capital expenditures and will also provide opportunities for operational cost and productivity improvements. The Çöpler Copper Case shows the results of a shorter term analysis using the Reserve Case metal prices and the impact of the estimated capital and potential cost savings from economies of scale and reallocation of shared and fixed costs.

For the Initial Assessment economic analysis, the Ardich and Çakmaktepe Mineral Reserve has been included in the cash flow analysis without change from the Reserve Case. This is to allow the analysis to quantify the impact of the copper concentrator and NaSH circuit and demonstrate the potential of the additional Mineral Resources.

A separate analysis of the Initial Assessment Case was prepared using only Measured and Indicated Mineral Resources (MI Case). Comparison of the initial years of the Initial Assessment and the MI Case showed only 1.4% of the material processed in the first nine years of the Initial Assessment is Inferred Mineral Resource. Most of the Inferred material is processed in years 10 to 20 and does not exceed 50% of the total processing in any one year.

The Initial Assessment Case production is oxide of 41.8 Mt at 1.26 g/t Au, 59.7 Mt at 2.45 g/t Au of sulfide, and an additional 24.9 Mt at 0.50 g/t Au and 0.20% Cu amenable to concentrator treatment for a total of 126.4 Mt at 1.67 g/t Au. The gold production in the Initial Assessment Case is 5.4 Moz and 164 Mlb of copper. Copper is produced from all three processing streams. The impact of including the copper concentrator as a processing facility is to expand the Çöpler pit, which ceases mining in 2043. Additional production in the Initial Assessment Case comes from feed of 1.8 Mtpa to the copper concentrator and from additional sulfide and oxide processing feed that is exposed when the pit gets deeper. Total capital including contingency of 25% for the copper concentrator and the copper recovery circuit in the sulfide plant is $218M. The capital costs have an accuracy of ±50%.

The Initial Assessment Case results include:

- After-tax NPV at a 5% discount rate of $2.00 billion.
- Mine life of 22 years.

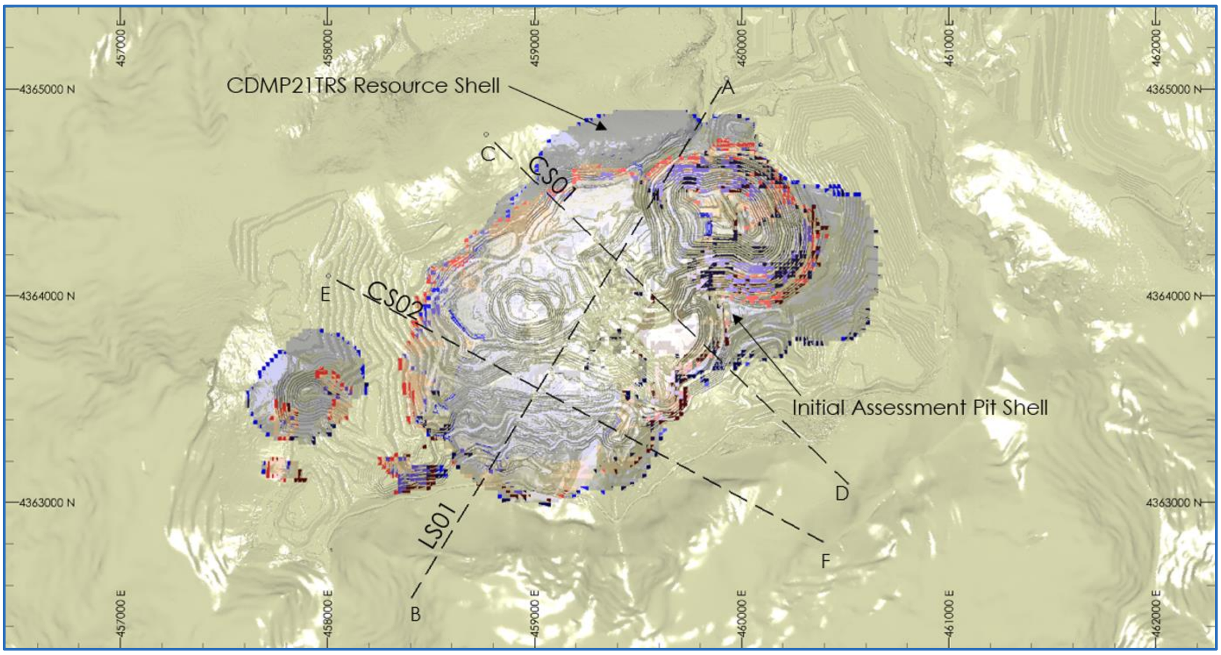
The initial Assessment Case shows an average AISC of $924/oz gold.

The Initial Assessment Case is a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

A plan and section showing the Initial Assessment pit shell and the Reserve Case pit design are shown in Figure 1.12 and Figure 1.13.

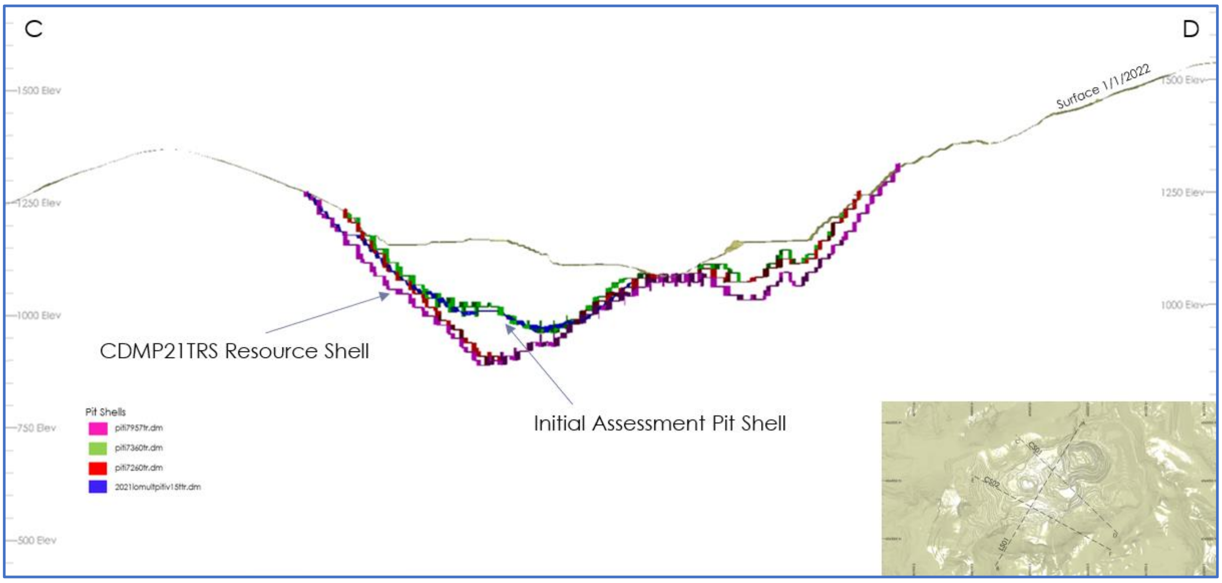
Key results of the Initial Assessment Case economic analysis are shown in Table 1.12. The after-tax cash flow is shown in Figure 1.14. The sulfide and oxide production profiles are shown in Figure 1.15 and Figure 1.16 respectively. The NPV results for before and after-tax over a range of discount rates is shown in Table 1.13. Gold unit costs are shown in Table 1.12.

Figure 1.12 Çöpler Plan Initial Assessment Pit Shell, Resource Shell, and Reserve Pit Design



OreWin, 2022

Figure 1.13 Çöpler Long-section Initial Assessment Pit Shell, Resource Shell, and Reserve Pit Design



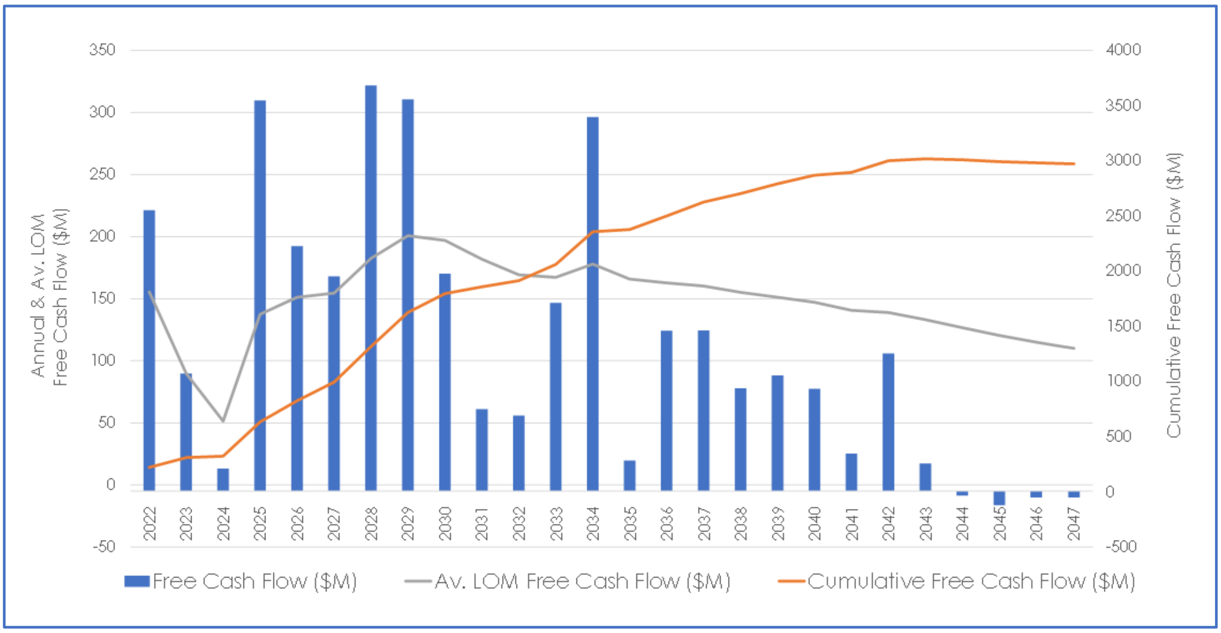
OreWin, 2022

Table 1.12 Initial Assessment Case Results Summary

Item	Unit	Initial Assessment Case
Oxide Processed		
Heap Leach Quantity	kt	41,792
Au Feed Grade	g/t	1.26
Sulfide Processed		
Quantity Milled	kt	59,654
Au Feed Grade	g/t	2.45
Cu Concentrator Processed		
Quantity Milled	kt	24,939
Au Feed Grade	g/t	0.50
Cu Feed Grade	%	0.20
Total Gold Produced		
Oxide – Gold	koz	1,068
Sulfide – Gold	koz	4,078
Cu Concentrator – Gold	koz	222
Total – Gold	koz	5,368
Total Copper Production	Mlb	164
5-Year Annual Average		
Average Gold Produced	kozpa	300
Free Cash Flow	$Mpa	165
Total Cash Costs (CC)	$/oz gold	761
All-in Sustaining Costs (AISC)	$/oz gold	938
Key Financial Results		
LOM Total Cash Costs (CC)	$/oz gold	783
LOM All-in Sustaining Costs (AISC)	$/oz gold	924
Site Operating Costs	$/t treated	43.79
After-Tax NPV5%	$M	2,004
Mine Life	years	22

5-Year annual average is for the period 1 January 2022 through 31 December 2026

Figure 1.14 Initial Assessment Case After-Tax Cash Flow



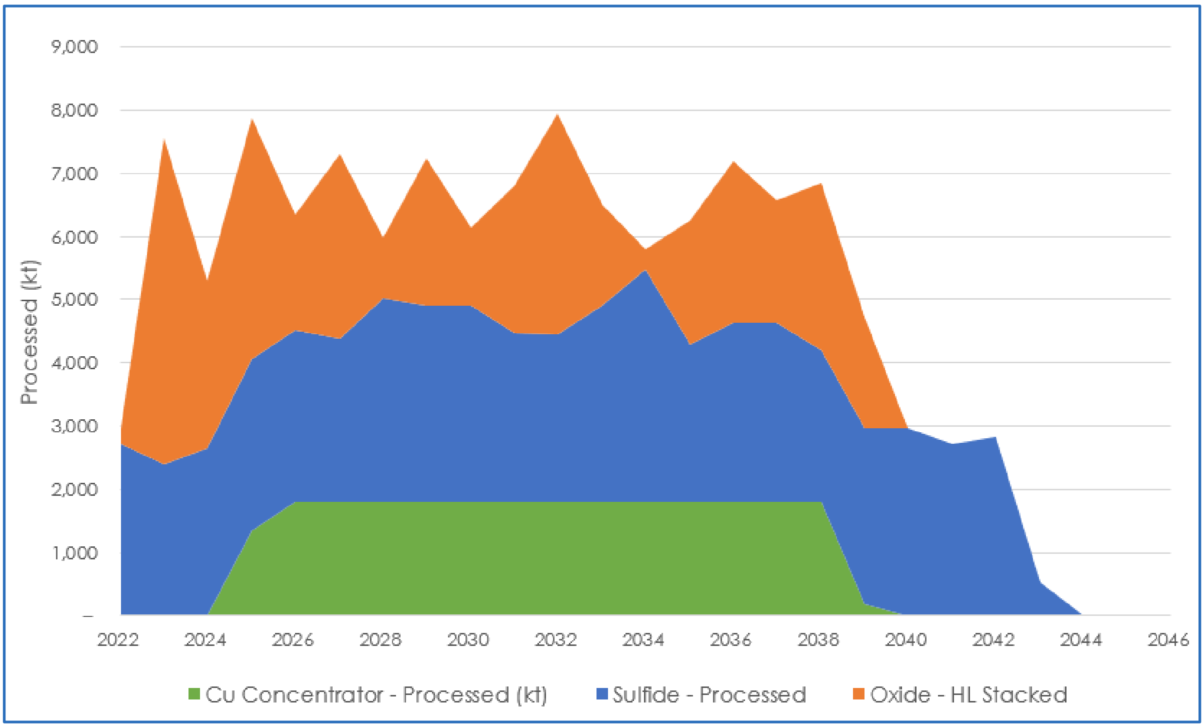
OreWin, 2022

Table 1.13 CDMP21TR Initial Assessment Case Before and After-Tax NPV

Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	3,301	2,958
5%	2,194	2,004
10%	1,571	1,457
12%	1,398	1,304

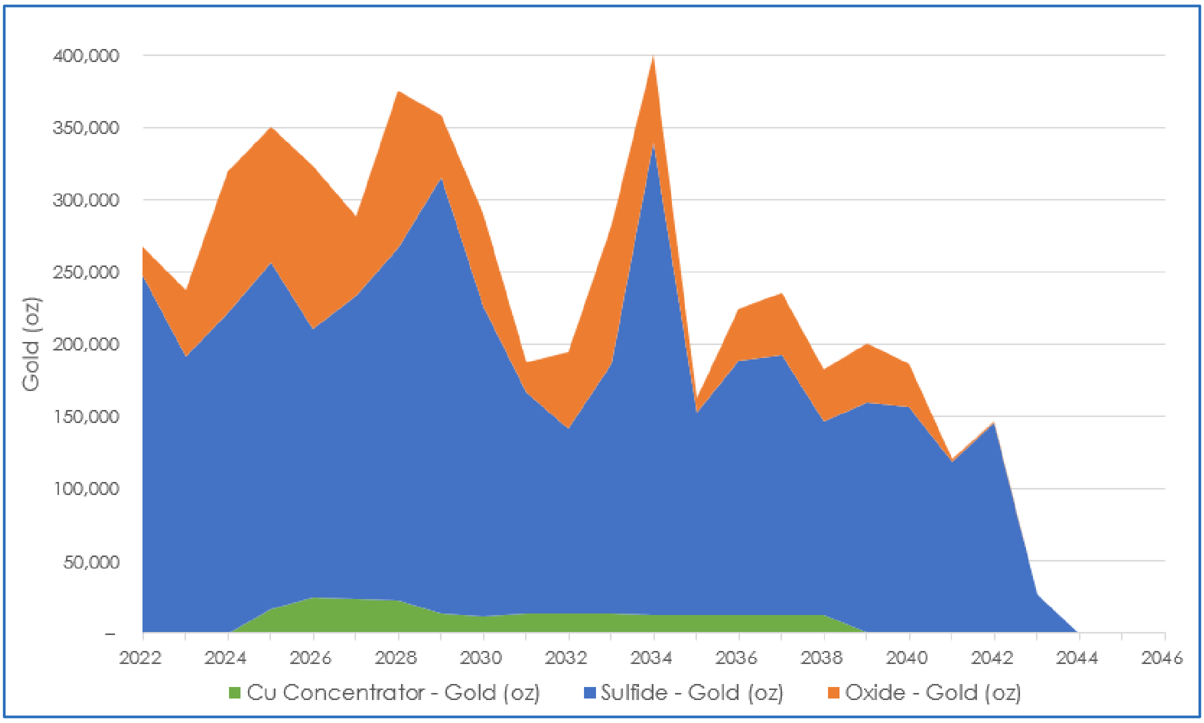
OreWin

Figure 1.15 Initial Assessment Case Processing



OreWin, 2022

Figure 1.16 Initial Assessment Case Gold Production



OreWin, 2022

The Initial Assessment has been prepared to demonstrate economic potential of the Mineral Resources at the Çöpler Deposit. The Initial Assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have modifying factors applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that this economic assessment will be realised.

Costs, taxation, and royalties used in the Initial Assessment Case were the same as in the Reserve Case. The metal prices used for the economic analysis are shown in Table 1.11.

The estimates of cash flows have been prepared on a real basis with a base date of Q4'21 and a mid-year discounting is used to calculate NPV.

1.22 Interpretation and Conclusions

The CDMP21TR has identified additional Mineral Resources and additional Mineral Reserves when compared to prior studies.

The Initial Assessment Case has demonstrated that there is significant economic potential that may be derived from the copper in the Çöpler Mineral Resource. Given this economic potential it is then concluded that it is valid to report the Mineral Resources using the Mineral Resource metal prices and the Resource pit shell.

Further study and analysis will be required to advance the understanding of this potential.

1.23 Recommendations

Key recommendations from the CDMP21TR are:

- Continue to update and evaluate the Çöpler District Master Plan as the existing Mineral Resources and Mineral Reserves are updated and as new prospects are advanced.
- Undertake infill drilling at Çöpler and update the copper Mineral Resource estimate.
- Prepare further studies of the copper recovery options.
- Conduct Geotechnical reviews and re-evaluation of updated pit designs.
- Optimisation of the sulfide flotation circuit, POX, and process operation.
- Metallurgical testwork on future oxide, sulfide, and copper ore sources.
- Optimisation of the oxide heap leach circuit.
- Optimisation of the mining rates to increase gold production.
- Stockpile reconciliation and management studies.
- Review and adapt ore control and stockpiling strategies to optimise recovery and throughput and maximise gold production.
- Reconcile monthly blend and gold production with predictive modelling.
- Continue exploration drilling at Ardich.
- Conduct geotechnical studies of Ardich.

- Conduct reconciliation studies of Çöpler.
- Update Çöpler and Ardich resource models and estimates.
- Further study of Initial Assessment Case and advance to next stage of study:
  - Geotechnical studies
  - EIA and permitting
  - Blasting studies
  - Metallurgical studies

2 INTRODUCTION

2.1 SSR Mining Inc.

The Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.

Exploration tenures surrounding the project area and mining at Çakmaktepe are subject to joint venture agreements between SSR and Lidya that have varying interest proportions. SSR controls 50% of the shares of Kartaltepe Madencilik Sanayi ve Ticaret Anonim Şirketi (Kartaltepe) and 30% of Tunçpinar Madencilik Sanayi ve Ticaret Anonim Şirketi (Tunçpinar). Lidya holds the remaining 50% of Kartaltepe and 70% of the Tunçpınar. Ownership percentages of the Mineral Resources are shown in Table 1.4 and of the Mineral Reserves in Table 1.6.

In most cases, the parent company will be referred to as SSR throughout this Technical Report even though it may have been Alacer or Anatolia at the time referenced in the report. Anagold remains the operating company for the Çöpler project and is the entity that undertakes the day-to-day work for the project.

2.2 Terms of Reference and Purpose of the Report

The CDMP21TR is an Independent Technical Report (TR) on the Çöpler project, prepared for SSR as part of the strategy for expansion of the Çöpler project. The CDMP21TR was prepared by OreWin, working with SSR, Anagold, and their consultants.

The primary source of data for the CDMP21TR is the Çöpler District Master Plan 2021.

This Report uses metric measurements except where otherwise noted. The currency used is US dollars ($) unless otherwise stated.

2.3 Qualified Persons

The Qualified Persons (QPs) are:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director - Mining, was responsible for the overall preparation of the CDMP21TR and, the Mineral Reserve estimates, Sections 1 to 6; Sections 13; Sections 15 to 27.
- Sharron Sylvester, BSc (Geol), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director - Geology, was responsible for the preparation of the Mineral Resources, Sections 1 to 12; Section 14; Sections 23 to 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

- Mr Bernard Peters visited the project 13–17 May 2019, 15–21 September 2019, 14–18 October 2019, 18–21 November 2019, and 27 February to 4 March 2020. The site visits included briefings from geology and exploration, mine, processing, environmental, permitting, and corporate personnel, site inspections of current and future areas for mining and plant and infrastructure, and discussions with other consultants. In addition, several visits to SSR's head office in Denver Colorado were undertaken during the same timeframe for the purpose of project-related meetings.
- Sharron Sylvester visited the project 13–17 May 2019, 15–21 September 2019, 14–18 October 2019, 18–21 November 2019, and 27 February to 4 March 2020. The site visits included briefings from geology and exploration, mine, processing, environmental, permitting, and corporate personnel, site inspections of current and future areas for mining and plant and infrastructure, and discussions with other consultants. In addition, several visits to SSR's head office in Denver, Colorado were undertaken during the same timeframe for the purpose of project-related meetings. Visits to analytical laboratories were planned to be undertaken but not completed due to global travel restrictions related to Covid-19.

2.5 Effective Dates

The report has several effective dates, as follows:

- Effective date of the Report: 31 December 2021.
- Date of drillhole database close-out for the Çöpler Mineral Resource estimate: 15 July 2015.
- Date of drillhole database close-out for the Çakmaktepe Mineral Resource estimate: 31 October 2019.
- Date of drillhole database close-out for the Ardich Mineral Resource estimate: 29 May 2021.
- Effective date of Mineral Resources: 31 December 2021.
- Effective date of Mineral Reserves: 31 December 2021.

2.6 Information Sources and References

The reports and documents listed in Section 6.1 (Previous Technical Reports), Section 3 (Reliance on Other Experts), and Section 27 (References) of this report were used to support the preparation of the report. Additional information was sought from SSR and Anagold personnel where required.

3 RELIANCE ON OTHER EXPERTS

OreWin has relied on the following information provided by SSR in preparing the findings and conclusions in this Technical Report regarding the following aspects of modifying factors:

- Macroeconomic trends, data, and assumptions, and interest rates.
  - This has been used in Sections 19 and 22
- Marketing information and plans within the control of the registrant.
  - This has been used in Sections 19 and 22
- Legal matters outside the expertise of the qualified person, such as statutory and regulatory interpretations affecting the mine plan.
  - This has been used in Sections 4 and 20
- Environmental matters outside the expertise of the qualified person.
  - This has been used in Sections 4 and 20
- Accommodations the registrant commits or plans to provide to local individuals or groups in connection with its mine plans.
  - This has been used in Sections 4 and 20
- Governmental factors outside the expertise of the qualified person.
  - This has been used in Sections 4 and 22

The source for all this information is the Çöpler District Master Plan 2021.

OreWin considers it reasonable to rely on SSR because SSR employs professionals and other personnel with responsibility in these areas and these personnel have the best understanding of these areas. OreWin is not qualified to provide advice on legal, permitting and ownership matters.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The CDMP21TR is an independent Technical Report, in accordance with NI 43-101, prepared for SSR on the Çöpler project (the project), located in Turkey. The project consists of several mining licences covering Mineral Resources on the Çöpler, Çakmaktepe, Ardich, and Bayramdere deposits, Mineral Reserves on the Çöpler and Çakmaktepe open pit mines, oxide and sulfide processing facilities, and supporting infrastructure.

The Çöpler project is in east central Turkey, 120 km west of the city of Erzincan, in Erzincan Province, 40 km east of the iron-mining city of Divriği (one-hour drive), and 550 km east of Turkey's capital city, Ankara. The nearest urban centre, Iliç, (approximate population 3,800), is located approximately 6 km east of the current Çöpler pit. Figure 4.1 illustrates the location of the project within the country of Turkey and indicates the deposit's proximity to surrounding communities.

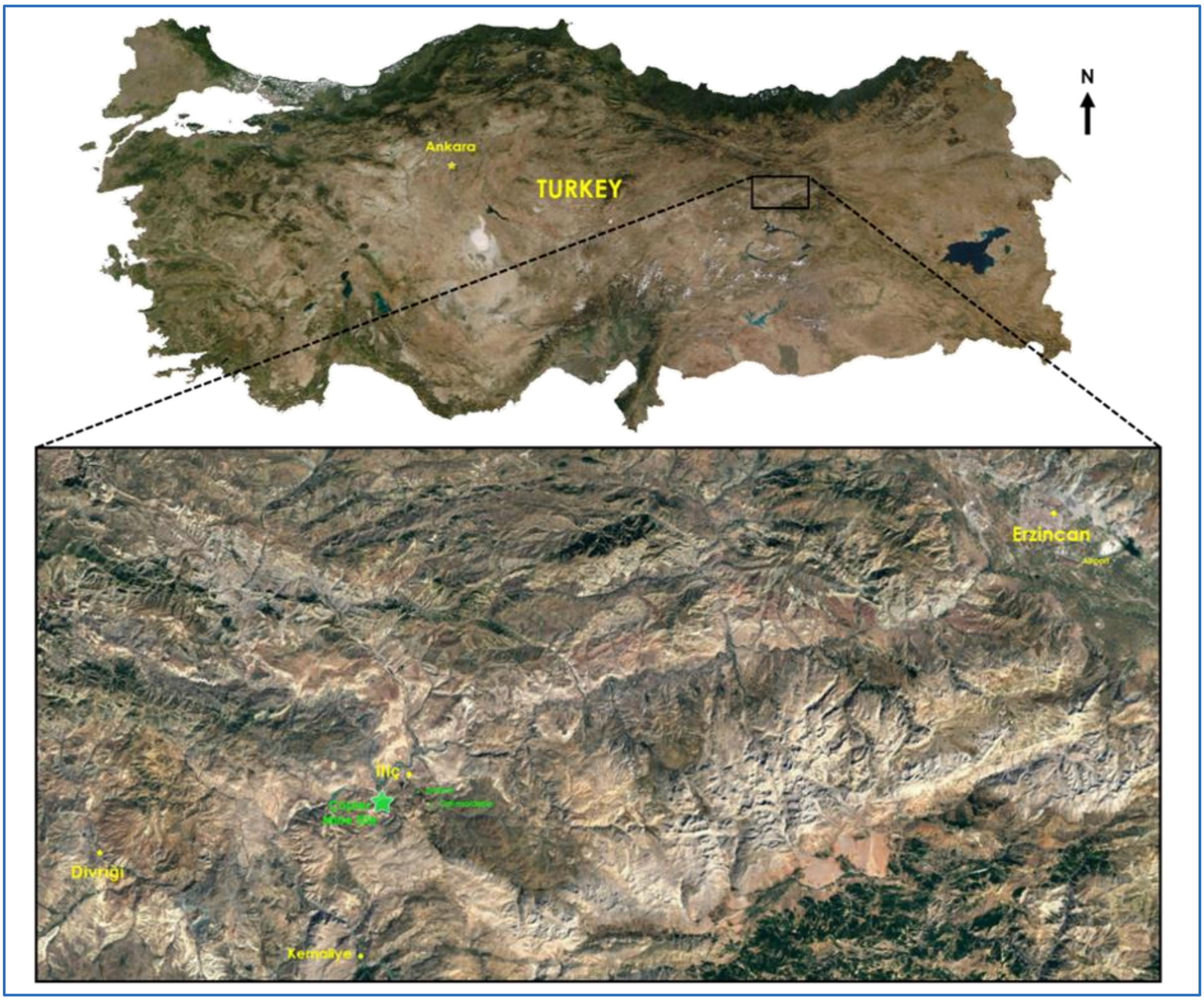
The Çöpler project uses the European 1950 (E1950) datum coordinate system, which is a Turkish Government requirement. The Çöpler deposit is in UTM6 zone 37N of the E1950 coordinate system. The Çöpler project centroid is situated at approximately 459,975 mE and 4,364,420 mN and has an approximate elevation of 1,160 m above mean sea level (mamsl).

The Çöpler mining operations are located 900 m south-west of the Iliç-district centre, 650 m south of the Bahçe neighbourhood, 250 m south of the Çöpler village, and 180 m north of the Sabırlı village. The project site lies within the licence areas numbered 847, 49729, and 20067313 (Figure 4.2), which have been granted by the General Directorate of Mining and Petroleum Affairs (MAPEG).

The Çakmaktepe satellite mining operation is located 6 km east of the current Çöpler pit and 1.5 km south of Iliç. The Çakmaktepe pits are located within Kartaltepe Licence 1054. Ore mined at Çakmaktepe is hauled and treated at the Çöpler facilities.

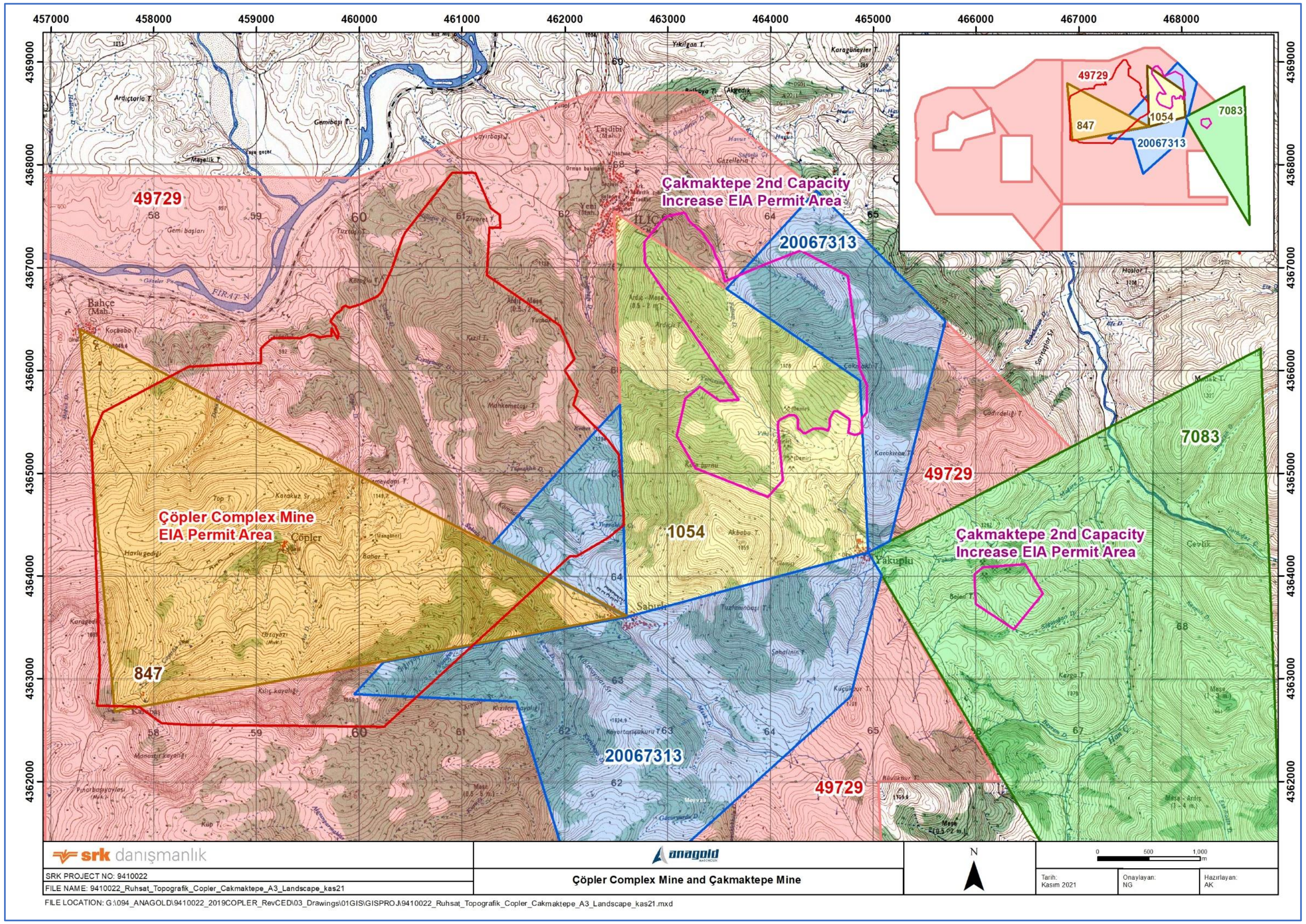
The Çöpler operation's currently permitted Environmental Impact Assessment (EIA) boundary incorporates 1,747 ha, whereas the footprint of the mine units covers a combined 1,089 ha. The currently permitted Çakmaktepe EIA boundary incorporates 290 ha. Pending approval, Çakmaktepe EIA boundary will increase to 360 ha with the second capacity increase.

Figure 4.1 Location of the Project



Anagold, 2020

Figure 4.2 Çöpler Project Licence and Surrounding Licences (UTM Grid)

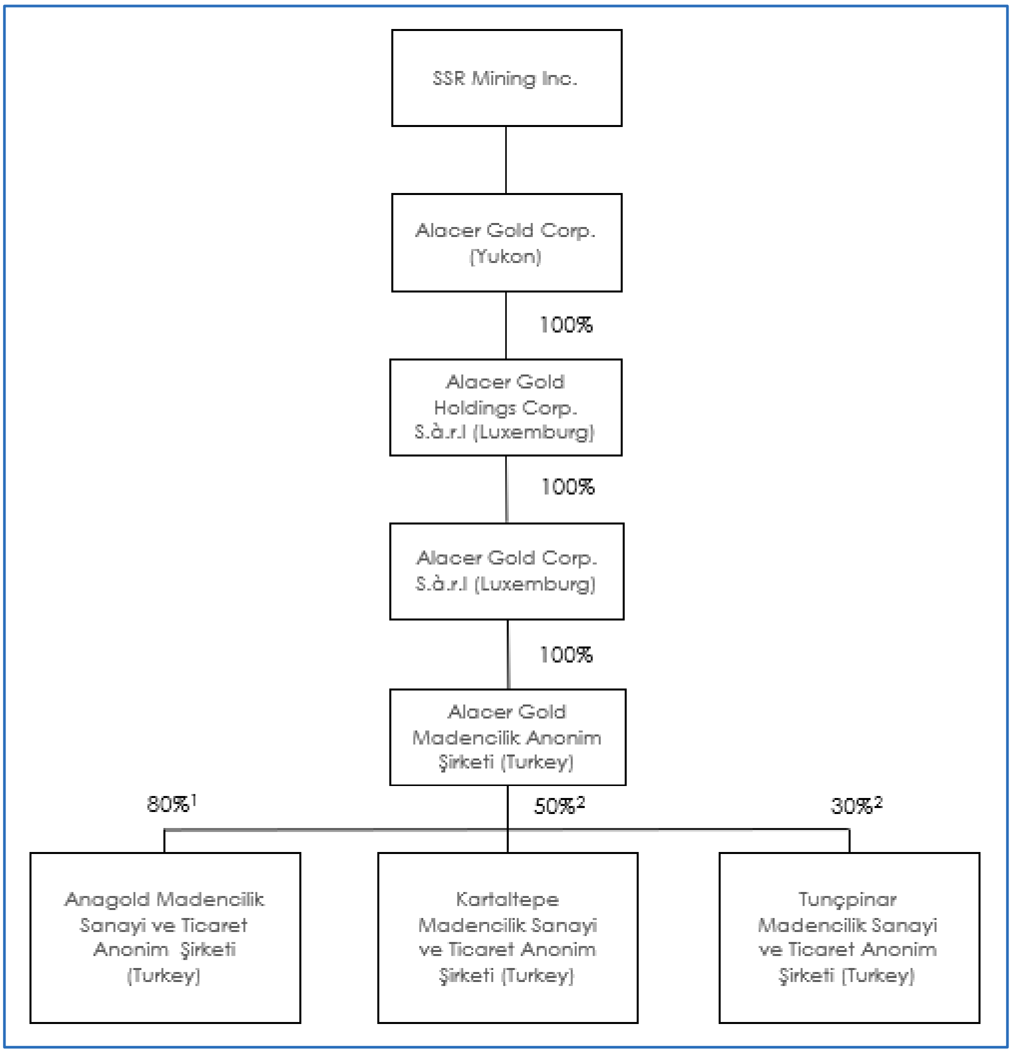


Anagold, 2021

4.2 Ownership

SSR controls the Çöpler project through a series of companies that own the licence areas. The company structure that links SSR to the Çöpler project is shown in Figure 4.3.

Figure 4.3 Ownership



1 Lidya holds 18.5% of this entity and Bank Kombetare Tregtare SHA, a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%

2 Lidya holds the remaining 50% of Kartaltepe and 70% of the Tunçpınar

The Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.

Exploration tenures surrounding the project area and mining at Çakmaktepe are subject to joint venture agreements between SSR and Lidya that have varying interest proportions. SSR controls 50% of the shares of Kartaltepe Madencilik Sanayi ve Ticaret Anonim Şirketi (Kartaltepe) and 30% of Tunçpinar Madencilik Sanayi ve Ticaret Anonim Şirketi (Tunçpinar). Lidya holds the remaining 50% of Kartaltepe and 70% of the Tunçpınar.

Ownership percentages of the Mineral Resource are shown in Table 1.4 and of the Mineral Reserves in Table 1.6.

The license that hosts the Çöpler deposit, including the Mineral Resources and Mineral Reserves, is wholly-owned by Anagold. Çakmaktepe is wholly-owned by Kartaltepe. Ardich, Mavialtin, Bayramdere, Aslantepe, and Findiklidere have areas owned by both Anagold and Kartaltepe.

The Initial Assessment Case has only analysed Mineral Resources located on the Anagold licence.

4.3 Mineral Tenure

Anagold holds the exclusive right to engage in mining activities within the Çöpler project area. Anagold holds six granted licences (Table 4.1) covering a combined area of approximately 16,600 ha. Mineral title is held in the name of Anagold. Kartaltepe holds eight licences covering approximately 9,200 ha. The total near-mine tenement package is approximately 25,800 ha. Anagold currently holds sufficient surface rights to allow continued operation of the mining operation in the Reserve Case. The major licence boundaries are shown in Figure 4.4.

The granted licences include two clay borrow pit licences, numbered 76817 and 76818.

The Çöpler mine and associated infrastructure are hosted within the triangular-shaped concession 847. Anagold has received approval from the Mining Affairs Committee to grant extensions to the three Anagold licences that had expired (76817, 76818, and 50237). Licenses 76817 and 76818 have been extended to 15 July 2029 and Licence 50237 has been extended until 21 March 2028.

Anagold has confirmed that charges and administrative expenses due to the Turkish Ministry of Energy and Natural Resources, Directorate General of Mining and Petroleum Affairs (MAPEG) have been paid, and all Anagold licences were in good standing as of 31 December 2021.

The mined Çakmaktepe pits are all on Kartaltepe Licence 1054. Bayramdere prospect is on Kartaltepe Licence 7083.

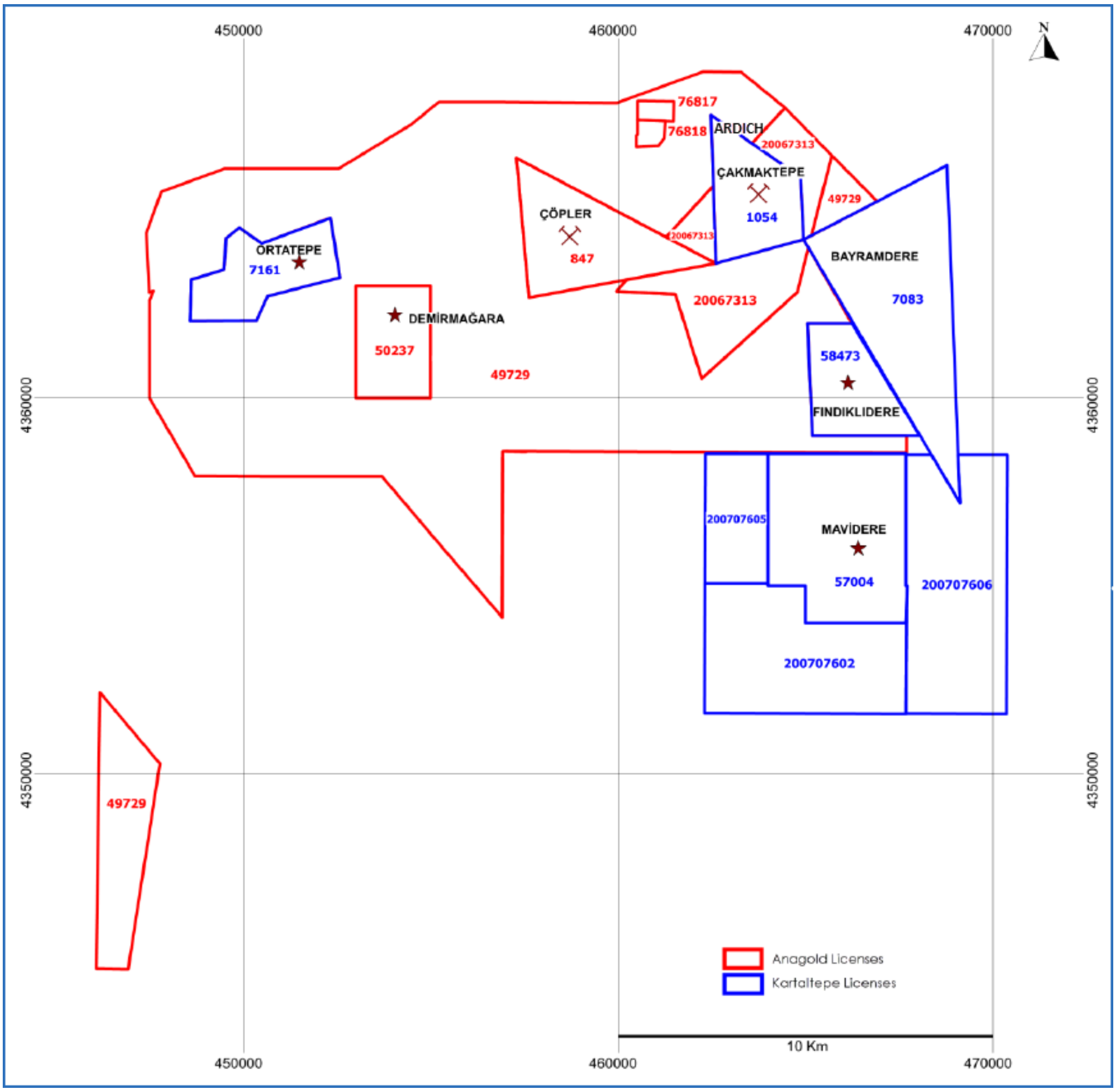
The three expired Kartaltepe licences (200707602, 200707605, and 200707606) were combined, and an operation project was prepared and submitted to receive an operation licence. The process continues. Kartaltepe maintains ownership of these licences during this process.

OreWin

Table 4.1 Granted Licences and Operating Permits

Province	Town	Village	Registration No.	Licence No.	Licence Area (ha)	Licence Type	Licence Group	Operation Permit	Operation Permit Area (ha)	Licence Issue Date	Licence Expiry Date	Licensee	Project
Erzincan	İliç	Çöpler	1027313	847	941.92	Operation	IV (Metallic)	Au+Ag+Cu+Hg Mn	Au+Ag+Cu+Hg: 941.92 Mn: 941.92	6/11/1986	6/11/2026	Anagold	Çöpler-Çöpler Saddle
Erzincan	İliç	Çöpler	2384036	49729	13,747.51	Operation	IV (Metallic)	Au+Ag+Cu+Mo	909.50	4/08/2016	4/08/2026	Anagold	Ardich-Çöpler Saddle-Kiziltepe-Meseburnu
Erzincan	İliç	Ortatepe	2386272	50237	600.00	Operation	IV (Metallic)	Au	18.07	21/03/2018	21/03/2028	Anagold	Elmadere-Demirmagara
Erzincan	İliç	Sabırlı	3095732	20067313	1,184.91	Operation	IV (Metallic)	Au+Ag+Cu	216.41	25/10/2021	25/10/2031	Anagold	Çakmaktepe Se-Ardich
Erzincan	İliç	Çöpler	3201587	76817	49.32	Operation	I-B (Brick Tile Clay)	Clay	6.68	15/07/2019	15/07/2029	Anagold	Clay Licence
Erzincan	İliç	Çöpler	3201588	76818	49.09	Operation	I-B (Brick Tile Clay)	Clay	49.09	15/07/2019	15/07/2029	Anagold	Clay Licence
Total (ha)					**16,572.75**								
Erzincan	Kemaliye	Kabataş	2450158	57004	1,564.69	Operation	IV (Metallic)	Au+Cu	931.87	2/09/2018	2/09/2023	Kartaltepe	Mavidere
Erzincan	Kemaliye		3129489	200707602	1,572.23	Pending Operation	IV (Metallic)	–	–	2/08/2007	2/08/2012	Kartaltepe	Mavidere
Erzincan	Kemaliye		3129490	200707605	577.92	Pending Operation	IV (Metallic)	–	–	2/08/2007	2/08/2012	Kartaltepe	Mavidere
Erzincan	Kemaliye		3129496	200707606	1,818.11	Pending Operation	IV (Metallic)	–	–	2/08/2007	2/08/2012	Kartaltepe	Mavidere
Erzincan	İliç		1032544	58473	606.60	Operation	IV (Metallic)	Fe+Cu	7.54	16/11/2017	16/11/2027	Kartaltepe	Findiklidere
Erzincan	İliç	Yakuplu	1032719	7083	1,756.55	Operation	IV (Metallic)	Au+Ag+Cu+Fe Cr	Au+Ag+Cu+Fe: 175.00 Cr: 607.47	2/04/2011	2/04/2021	Kartaltepe	Bayramdere-Aslantepe-Saridere
Erzincan	İliç	Yakuplu	1027026	1054	660.87	Operation	IV (Metallic)	Au+Ag+Cu+Fe	359.33	30/07/2017	30/07/2027	Kartaltepe	Çakmaktepe
Erzincan	İliç	Ortatepe	2003094	7161	642.68	Operation	IV (Metallic)	Fe	214.65	7/05/2013	7/05/2023	Kartaltepe	Ortatepe
Total (ha)					**9,199.65**								

Figure 4.4 Tenure Layout Plan



Anagold, 2020

4.4 Surface Rights

Anagold currently holds sufficient surface rights to support the Reserve Case oxide heap leach mining operations and sulfide processing and tailings disposal.

4.5 Taxation

The Turkish government implemented a temporary taxation rate increase from 20% to 22% for the periods of 2018–2020 25% for 2021 and 23% for 2022. From 2023 onwards, the effective tax rate is expected to return to 20%.

The CDMP21TR economic analysis applies a corporate tax rate of 20%.

For tax purposes, a 20% accelerated depreciation rate is applicable for both the oxide and sulfide capital. The depreciation period is 10-years for general mining equipment, if not specifically defined by the tax office.

Investment incentive certificates are available for investments that promote economic development. Investment incentive certificates can be classified as strategic in specific circumstances and such certificates provide additional incentives. Anagold received a strategic incentive certificate for the sulfide process plant. An investment incentive certificate generates credits that offset corporate income taxes generated by the investment. The amount of investment credits generated from the investment incentive certificate is based on eligible capital expenditures. The investment credits generated by the strategic investment incentive certificate reduce the corporate tax rate to a minimum of 2% in a given tax period until the last quarter of 2023, thereafter it is assumed subsequent non-strategic investment incentive certificates will be available and the minimum rate will be 4%. Incentive tax credits can be carried forward to future tax periods indefinitely until exhaustion. Incentive tax credits and other tax pools are determined in the local currency, Turkish Lira, and subject to devaluation and revaluation as fluctuations against the US dollar occur. The cash flow model is prepared on a constant Turkish Lira basis.

Value-added tax (VAT) in Turkey is levied at 18% and the project is eligible for the Turkish exemptions for mining projects and mining equipment purchases. In the CDMP21TR assumes the cash flows are not subject to VAT.

Import duties are not included in the capital cost estimate for mining related imported equipment because they are exempted in the incentive certificates.

4.6 Royalties

Under Turkish Mining Law, the royalty rate for precious metals is variable and tied to metal prices. The Çöpler project is subject to a mineral production royalty that is based on a sliding scale to gold price and is payable to the Turkish government. In September 2020 a presidential decree was issued, increasing the prescribed royalty rates by 25%.

Table 4.2 details the relevant prescribed royalty rates along with the revised rates following the September 2020 presidential decree. The royalties are calculated on total revenue with deductions allowed for processing and haulage costs of ore. Revenue from by-products (silver and copper) is included in the total revenue used for royalty calculations.

The royalty rates outlined in Table 4.2 apply to gold production from heap leaching. Royalty rates are reduced by 40% for ore processed in country, as an incentive to process ore locally. As the Çöpler project produces its gold doré on-site, the Çöpler project is eligible for a 40% reduction to the royalty rate for gold produced from POX processing.

Table 4.2 Gold Royalty Rates

Metal Price ($/oz Gold)		Prescribed Royalty Rate (%)	Royalty After 40% In-Country Processing Incentive (%)
From	To		
0	800	1.25	0.50
800	900	2.50	1.00
900	1,000	3.75	1.50
1,000	1,100	5.00	2.00
1,100	1,200	6.25	2.50
1,200	1,300	7.50	3.00
1,300	1,400	8.75	3.50
1,400	1,500	10.00	4.00
1,500	1,600	11.25	4.50
1,600	1,700	12.50	5.00
1,700	1,800	13.75	5.50
1,800	1,900	15.00	6.00
1,900	2,000	16.25	6.50
2,000	2,100	17.50	7.00
2,100	+	18.75	7.50

The Çöpler project effective life-of-mine (LOM) royalty rate based on the financial model metal price assumptions and applicable deductions is approximately 4.9%.

Other than the royalty payments, there are no other known back-in rights, payments, or other agreements and encumbrances to which the property is subject.

4.7 Environmental Liabilities

There are no known existing environmental liabilities for the Çöpler project, except for Anagold's obligation for ultimate reclamation and closure.

4.8 Permits

The EIA permitting for the Çöpler mine oxide ore was completed in April 2008 with the issuance of an EIA positive certificate. All the necessary operation permits have already been obtained for the oxide inventory. These include:

- explosive storage permit
- permit for water abstraction from groundwater sources
- EIA positive certificate for power transmission line construction
- environmental permits and licences
- land acquisition permits for forest areas and pasturelands
- workplace opening permit
- hazardous workplace permit
- operating permits.

The EIA permitting process for the Sulfide Expansion Project was commenced on 7 April 2014 and completed with the receipt of an 'EIA Positive Statement' on 24 December 2014. In addition to an EIA approval, other permits required for the Sulfide Expansion Project involved an expanded workplace opening permit, additional operating permits, and land acquisition permits for forest areas and pasture lands.

Additional EIA studies conducted, and environmental permits received for the Çöpler mine since the start of the gold mining operations are as follows:

- EIA permit, dated 10 April 2012, for the operation of mobile crushing plant.
- EIA permit, dated 17 May 2012, for the capacity expansion involving:
  - Increasing operation rate to 23,500 tpd.
  - Increasing Çöpler waste rock dump (WRD) footprint area.
  - Adding a sulfidisation, acidification, recovery, and thickening (SART) plant to the process in order to decrease the cyanide consumption due to the high-copper content of the ore.
- EIA permit, dated 24 December 2014, for the capacity expansion involving:
  - Sulfide plant expansion
  - Heap leach area expansion
- EIA permit, dated 26 January 2017, for the Çakmaktepe satellite pits expansion.
- EIA permit, dated 9 August 2018, for the Çakmaktepe expansion for the new defined Central pit.
- EIA permit dated 7 October 2021 for the second capacity expansion involving:
  - Heap leach pads 5 and 6
  - TSF expansion
  - Operation of floatation plant.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Çöpler project is accessed from the main paved highway between Erzincan and Kemaliye, crossing the Karasu River and passing by the village of İliç. From İliç there is an additional 4.5 km of road to reach the Çöpler mine site.

The Ankara to Erzincan railway line, operated by the Turkish State Railway Company, (TCDD), runs parallel to the south bank of the Karasu River and passes within 2 km north of the site at a point between the train stations at İliç and Bağıştaş. The railway line connects the site with Ankara and the west as well as with seaports to the north on the Black Sea, and to the south on the Mediterranean Sea. Overnight passenger sleeper cars are available between Erzincan and Ankara.

The reservoirs of the Bağıştaş I & II hydro-electric power plants (HEPP) are 350 m and 1,800 m away from the Çöpler mine site, respectively. The embankment of Bağıştaş I Dam originally covered a portion of the existing highway, railroad, and railroad station until these were relocated before dam construction was completed. Construction routes for the railroad and highway were located between the new Çöpler village and the Çöpler mine site. The bridge on the north-east side of İliç was relocated to further east of the embankment.

There are regular commercial airline flights from Istanbul and Ankara to the regional cities of Erzincan, Erzurum, Malatya, Elazığ, and Sivas. Driving from the regional cities to the project site takes between two to four-hours on paved highways. Driving from Ankara to the site takes approximately eight-hours.

5.2 Local Resources and Infrastructure

The district of İliç has a population of approximately 3,800 inhabitants and is located approximately 6 km east of the current Çöpler pit. The district has a hospital, schools, municipal offices, a fire station, a police station, and a Gendarmerie post. The primary economic activity in the region is sheep herding for wool, meat, and dairy products. Other agricultural activities include bee keeping for honey production and, some wheat farming along the Karasu River. Additionally, there is some light manufacturing and grain milling performed in İliç.

The workforce for the Anagold exploration programmes has primarily included residents drawn from the local communities of Çöpler, İliç, and Sabırlı.

Turkish telecommunications are up to European standards. High-speed, fibre-optic internet access is in operation at the mine site.

Initially, electrical power at 380 V and 50 Hz, was available in İliç and at the mine site. This was upgraded to support the project by the construction of a 40 km long 154 kV power line from the sub-station at Divriği to the mine site. The power supply was further upgraded when the hydroelectric dam near the mine site was commissioned. Çöpler is now connected to the national grid by a 6 km 154 kV powerline from the Bağıştaş sub-station.

Sufficient local fresh water supply exists to support the mining and processing operations. Ground water resources include seven production wells with a 25,728 m^3/day extraction permit. Further information on project infrastructure is included in Section 18. Section 20.3 contains additional data on the project social setting.

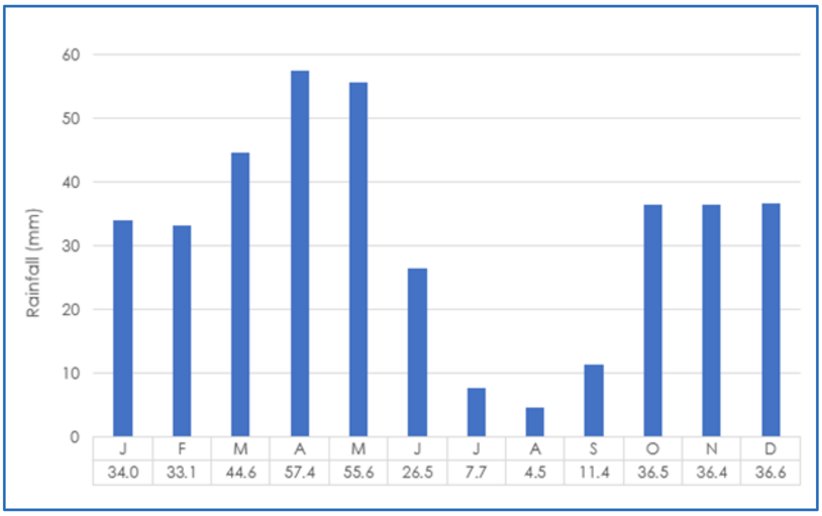
5.3 Climate

Site climate data were developed during previous studies. No additional climate data were generated for the CDMP21TR.

Mining operations are conducted year-round. The climate is typically continental with cold wet winters and hot dry summers. In winter, the night-time temperature can drop to –25°C although the average is usually a few degrees below freezing. The July temperature frequently exceeds +40°C but the climate is usually pleasantly warm outside of these extremes. The average monthly temperature ranges from +3.7°C for the coldest month of January to +23.9°C for August, the warmest month.

Most precipitation occurs in the winter and spring. Monthly average rainfall values are shown in Figure 5.1. The average annual rainfall for the site is 384.3 mm. Snowfall is common during the period mid-November through February, but with little, if any, accumulation. Snow depth assessments are based on the Divriği meteorological weather station, located 41 km west of the project area, which shows maximum snow-pack depths at approximately 200 mm for 1985.

Figure 5.1 Average Monthly Rainfall for Çöpler Project Area



Anagold, 2016

The frost depth is less than 0.3 m, based on local information, with 0.5 m selected as the design frost depth limit.

The maximum wind speed recorded at the Divriği weather station in 2004 ranges from 15–25 m/s, with variable directions mainly from the north, south, and east.

5.4 Hydrogeology

SRK compiled and updated the project conceptual hydrogeological model with new geological data, established a new numerical model and used it to evaluate the hydrogeology of the project area.

5.4.1 Existing Data Evaluation, Field Investigation, and Hydrogeology Conceptual Model

Within the regional hydrology area, lithological units are defined in three main classes according to their underground water transport and transmission properties. These units are:

- Impervious units.
- Low permeate units: such units contain some thin layers that are more permeable than other layers with small extensions and provide water through sources with a flow rate of less than 1 L/s.
- Conductive units and very permeable units: Munzur Formation limestone and Quaternary alluvium units.

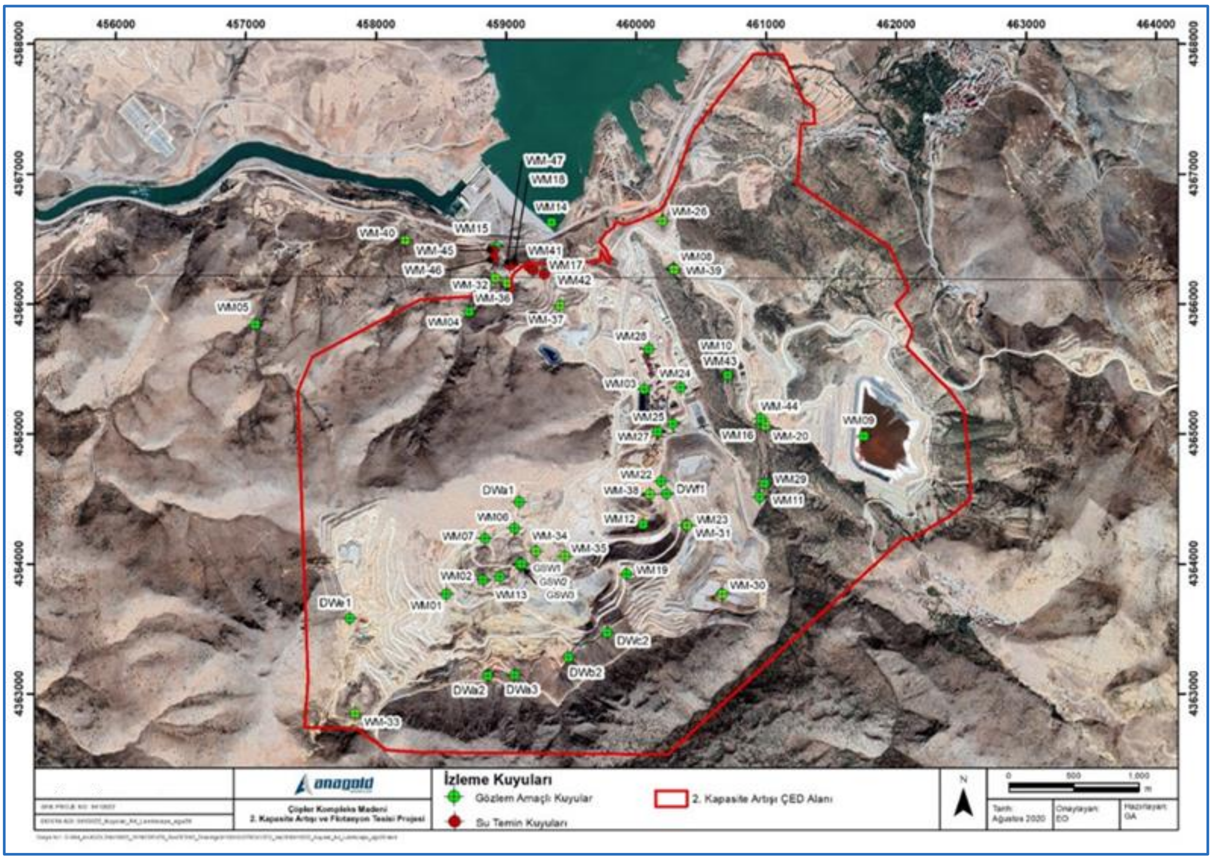
The regional geology is a complex structural assemblage of fault-bounded blocks including the following rock types:

- Limestone: grey to blue-grey, fine-grained to recrystallised marbles. Much of the unit displays various degrees of karst development. Bedding within the unit is indistinct to massive. This limestone group is also named the Çöpler limestone in the vicinity of the area where Mineral Resources have been estimated.
- Metasediment: fine-grained argillite sequences consisting of interbedded siltstones, shale units, marls, and sandy siltstones. The thermal and hydrothermal impact to this unit from the intrusions resulted in the creation of the skarns and hornfels.
- Ophiolitic mélange: ophiolitic mélange consists of diabase and serpentinite units. Serpentinisation is non-uniform and appears to be best developed near major fault zones.
- Diabase: the diabase is located within the upper zone of the ophiolitic mélange. In general, joint surfaces are covered with calcite and iron oxide sealing. In places, the rock mass shows blocky textures embedded in a fine matrix.
- Diorite to granodiorite intrusions: beige and light brown, medium to coarse-grained plutons. This formation has intruded into the pre-existing argillite's and Munzur limestone. This includes fine to medium-grained quartz, feldspar, biotite, and amphibole minerals.
- Skarn: the skarn zone is developed along the granodiorite contact with the limestone and ophiolitic mélange. This zone was developed under elevated pressure and temperature conditions during intrusion of the granodiorite mass. The skarn units are black to dark brown, silicified, moderately weathered and includes frequent solution cavities.

5.4.2 Well Installation

A total of 56 wells for groundwater observation, testing, and water supply purposes have been drilled. Forty-one of the wells were drilled prior to 2018, 10 were for groundwater control and slope stability studies in 2018, two were for waste storage area observation purposes, and three were developed in 2018 as part of the sulfide expansion project for additional water supply. Hydrogeology wells drilled are shown in Figure 5.2.

Figure 5.2 Groundwater Wells



Anagold, 2020

Groundwater is expected to be recharged through the infiltration of precipitation through secondary porosity in the bedrock terrain. Groundwater elevation data indicates that the flow direction is generally northward to the Karasu River through the Munzur limestone. During the resource drilling and sub-sequent monitoring well installation programmes, perched groundwater conditions were reported above the clay-altered intrusions. It is anticipated that the perched groundwater is present in restricted areas. The volume of water held in storage as perched groundwater is unknown.

Groundwater elevations at the Çöpler project range from 1,328.5 m at Well GMW-10 (southern end of the site) to 864.7 m at Well GMW-09 (northern end of the site). Observations of cavernous features (karst) during drilling and high-values of hydraulic conductivity from aquifer tests suggest an area of karst development in the limestone near the Karasu River, at boreholes GMW-09 and GMW-24. This was incorporated into the groundwater flow model as an area of high hydraulic conductivity near these wells and along the Sabırlı Fault.

5.5 Physiography

The Çöpler project is located in a roughly east–west oriented valley at altitudes of 1,100–1,300 m. The valley is surrounded by limestone mountains that rise to more than 2,500 m on the north and south sides of the project area. These mountains are at the western end of the Munzur range, which rises to more than 3,300 m between Ovacık and Kemah.

The region is sparsely vegetated, predominantly with semi-arid brush and scrub trees including dwarf oaks and junipers.

The following are the site data developed during previous studies for the design of the project:

- Latitude: 39° 25' North
- Longitude: 38° 32' East
- Elevation: 1,150 mamsl
- Frost depth: 500 mm
- Snow load: 145 kg/m^2
- Wind load: 40 m/sec, Exposure 'C'
- Earthquake zone: second order, Ao = 0.20
- Atmospheric pressure (average): 880.5 millibars
- Maximum design temperature: +40°C
- Minimum design temperature: –25°C
- Annual rainfall: 384 mm
- Maximum snowfall depth: 200 mm (estimated)
- Design maximum rainfall: 24-hours, 76 mm

6 HISTORY

The region around the Çöpler project has been subject to gold and silver mining dating back at least to Roman times, with historical bullion production estimated at approximately 50 koz of gold. A copper-rich slag pile of approximately 2.5 kt is located at the western edge of the district and is believed to be waste from ancient production. Although the district contains copper mineralisation, there appears to have been little production targeting copper. There are several additional minor slag piles scattered around the property thought to be from ancient, small-scale gold and by-product copper production.

The Turkish Geological Survey (MTA) carried out regional exploration work in the early 1960s that was predominately confined to geological mapping. In 1964, a local Turkish company started mining for manganese, continuing through until closing in 1973 and producing approximately 7.3 kt of manganese ore during its active life. Unimangan Manganez San A.Ş. (Unimangan) acquired the property in January 1979 and re-started manganese production, producing 1–5 ktpa of ore until ceasing operations in 1992.

In 1998, Anatolia Minerals Development Ltd (Anatolia) identified several porphyry-style gold–copper prospects in east central Turkey and applied for exploration licences for these prospects. This work was based upon the earlier work by MTA in the 1960s. During this effort, Anatolia delineated a prospect in the Çöpler basin formed by an altered and mineralised granodiorite, intruded metasediment, and limestone. This prospect and the supporting work were the basis for a joint venture agreement for exploration with Rio Tinto.

During the period of the joint venture, exploration drilling of the Çöpler deposit was completed and a Mineral Resource estimate was developed with three mineralised zones: Main, Manganese, and Marble. In January 2004, Anatolia acquired sole control over the project and maintained exclusivity until 2009, at which time a joint venture with Lidya was executed.

In February 2011, Anatolia merged with Avoca Resources Limited to form Alacer Gold Corp. (Alacer). In September 2020 Alacer merged with SSR.

Today the Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.

In most cases the company will be referred to as SSR throughout the CDMP21TR even though it may have been Alacer or Anatolia at the time referenced in the report.

The previous Technical Report was the 2020 Çöpler District Master Plan 2020 NI 43-101 Technical Report dated 27 November 2020.

The previous reporting of Mineral Resources and Mineral Reserves was in the SSR Annual Information Form. Those statements on Mineral Resources and Mineral Reserves have been used for comparison.

6.1 Previous NI 43-101 Technical Reports

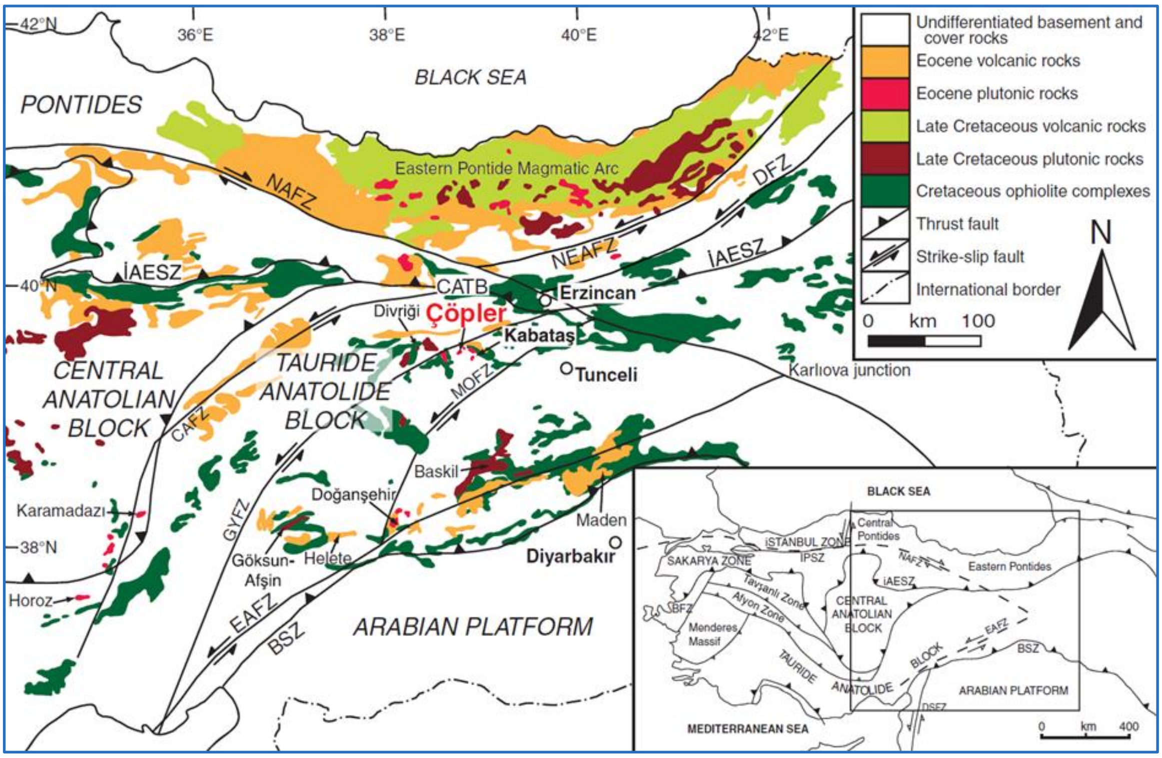
The following Technical Reports have been filed on the Çöpler project (in chronological order):

- Watts, Griffis and McQuat Limited, 2003. Update of the Geology and Mineral Resources of the Çöpler Prospect, 1 May 2003.
- Independent Mining Consultants, Inc., 2005. Çöpler Project Resource Estimate, 19 October 2005.
- Marek, J.M., Pennstrom, W.J., Reynolds, T., 2006. Çöpler Gold Project Feasibility Study, 30 May 2006.
- Marek, J.M., Moores, R.C., Pennstrom, W.J., Reynolds, T., 2007. Çöpler Gold Project, 2 March 2007, as amended 30 April 2007.
- Easton, C.L., Malhotra, D., Marek, J.M., Moores, R.C., and Pennstrom, W.J., 2008. Çöpler Gold Project East Central Turkey Preliminary Assessment Sulfide Ore Processing, 4 February 2008.
- Marek, J.M., Benbow, R.D., and Pennstrom, W.J., 2008. Çöpler Gold Project East Central Turkey, 5 December 2008 (amended and restated; supersedes 11 July 2008 version).
- Altman, K., Liskowich, M., Mukhopadhyay, D.K., and Shoemaker, S.J., 2011. Çöpler Sulfide Expansion Project Prefeasibility Study, 27 March 2011.
- Altman, K., Bascombe, L., Benbow, R.D., Mach, L., and Shoemaker, S.J., 2012. Çöpler Resource Update, Erzincan Province, Turkey, 30 March 2012.
- Altman, K., Bair, D., Bascombe, L., Benbow, R., Mach, L., and Swanson, B., 2013. Çöpler Mineral Resource Update, Erzincan Province, Turkey, 28 March 2013.
- Armstrong, D., Bascombe, L., Bohling, R., Kiel, R., Liskowich, M., Parker, H.M., Parshley, J., Seibel, G., and Swanson, B., 2014. Çöpler Sulfide Expansion Project Feasibility Study, Erzincan Province, Turkey, 29 July 2014.
- Bascombe, L., Benbow, R.D., Birch, R.G., Bohling, R., Francis, J., Khoury, C., Kiel, R., Liskowich, M., Marsden, J., Parker, H.M., Parshley, J., Seibel, G., and Statham, S., 2015. Çöpler Sulfide Expansion Project Feasibility Update, Erzincan Province Turkey, 27 March 2015.
- David, D., Kiel, R., Liskowich, M., Parshley, J., Marsden, J., Seibel, G., Parker, H., Bascombe, L., Benbow, R., Statham, S., Francis, J., and Smolonogov, S., 2016. Çöpler Mine, Erzincan Province, Turkey, 9 June 2016.
- OreWin Pty. Ltd., 2020. Çöpler District Master Plan 2020, 27 November 2020. (CDMP20TR)

7 GEOLOGICAL SETTING AND MINERALISATION

The Çöpler district is located near the north margin of a complex collision zone and to the south of the prominent North Anatolian Fault Zone (Figure 7.1). The collision zone, and subsequent crustal thickening, is related to the closure of the northern branch of the Neotethys ocean, resulting from the northward subduction and coming together of the Pontides and Tauride Anatolide Block in the Late Cretaceous to Early Tertiary. In this intensely-deformed tectonic region, east–west trending imbricated structures were cut by north–north-east trending strike-slip faults during the Late Cretaceous to Paleogene period.

Figure 7.1 Geological Setting of the Çöpler District



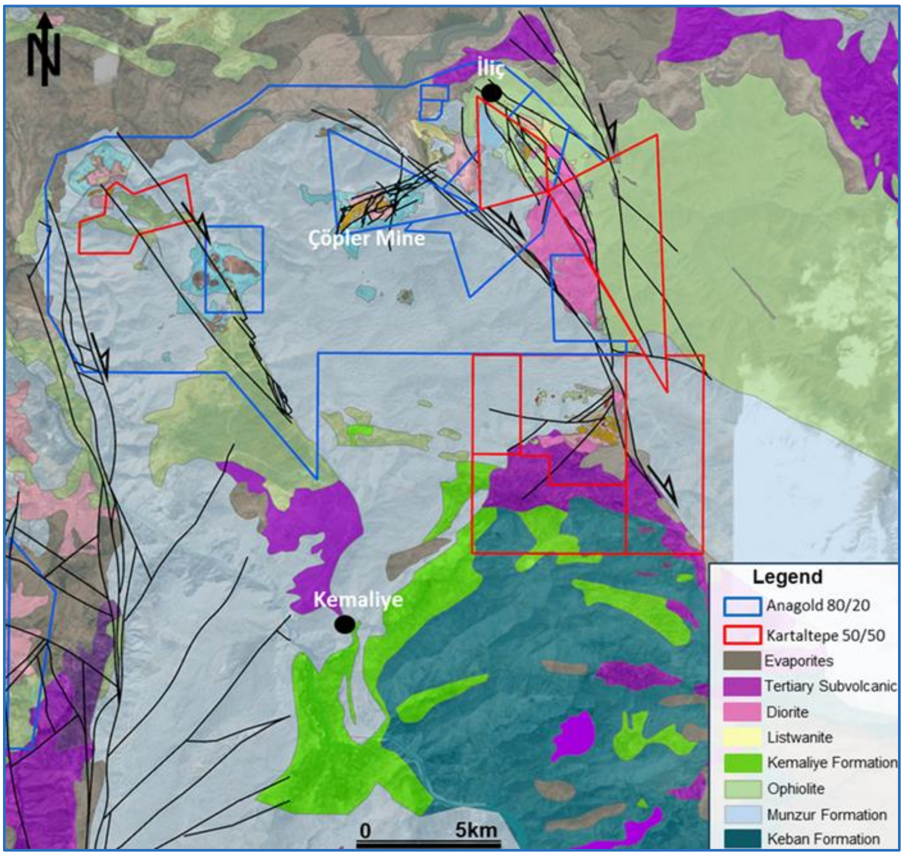
İmer, 2012

The Çöpler district deposits, including Çöpler, Çakmaktepe, Ardich, and Bayramdere, are within the Tethyan mineral belt, a terrane stretching from Indo-China to Europe through Eurasia that contains economically significant gold, copper, and base metal deposits.

Three main rock assemblages are exposed in the Çöpler district (Figure 7.2):

- The first assemblage includes the Keban, Munzur, and Kemaliye Formations. These units are tectonically overlain by ophiolitic nappes (Ovacık Formation of Özgül and Turşucu 1984).
- The second assemblage includes Middle Eocene magmatic and sedimentary rocks.
- The third assemblage includes the Oligocene to Recent sedimentary Sivas basin rocks.

Figure 7.2 Geological and Structural Map of the Çöpler District



Anagold, 2020

7.1 Geological Setting – Çöpler Deposit

7.1.1 Geology – Çöpler Deposit

The Çöpler deposit is centred on composite diorite to monzonite porphyry stocks that are part of the Eocene Çöpler Kabataş magmatic complex dated (by İmer et al., 2013) at:

- 43.8 ± 0.3 Ma and 44.2 ± 0.2 Ma (from 40 Ar / 39 Ar analysis of igneous biotite), and
- 44.1 ± 0.4 Ma (from igneous hornblende).

The magmatic rocks have intruded into both the Keban and Munzur Formations.

Rocks of the Permian to Upper Cretaceous Keban Formation shelf sequences vary in composition between siliciclastic and calcareous, with fine to medium-grained sandstone interbedded with mudstone, and locally thick sections of fine laminated mudstone. The sedimentary units are folded with a fold axis oriented at approximately 25→200 (plunge→ plunge direction) resolved from bedding measurements in the Çöpler pits. Limestone of the Upper Triassic to Late Cretaceous (Upper Campanian) Munzur Formation structurally overlies the folded Keban Formation with the contact represented by cataclasite at the base of the Munzur Formation. Intense shearing of the underlying sedimentary rocks is observed, with top-to-south kinematics.

Stratigraphically, the Munzur Formation overlies the Keban. However, mapping of the Munzur Formation to the north of Çöpler shows homoclinal structure with consistent bedding in the limestones (40 / 060, dip / dip-direction) indicating juxtaposition of structural blocks. The Munzur allochthon was thrusted onto Permo-Triassic metamorphic basement in the Late Cretaceous (Özgül and Turşucu 1984). This structural contact pre-dates Eocene Çöpler Kabataş intrusions, which appear to have intruded across the sheared contact between Keban Formation metamorphic rocks (Main Zone) and Munzur Formation limestone (Manganese Zone).

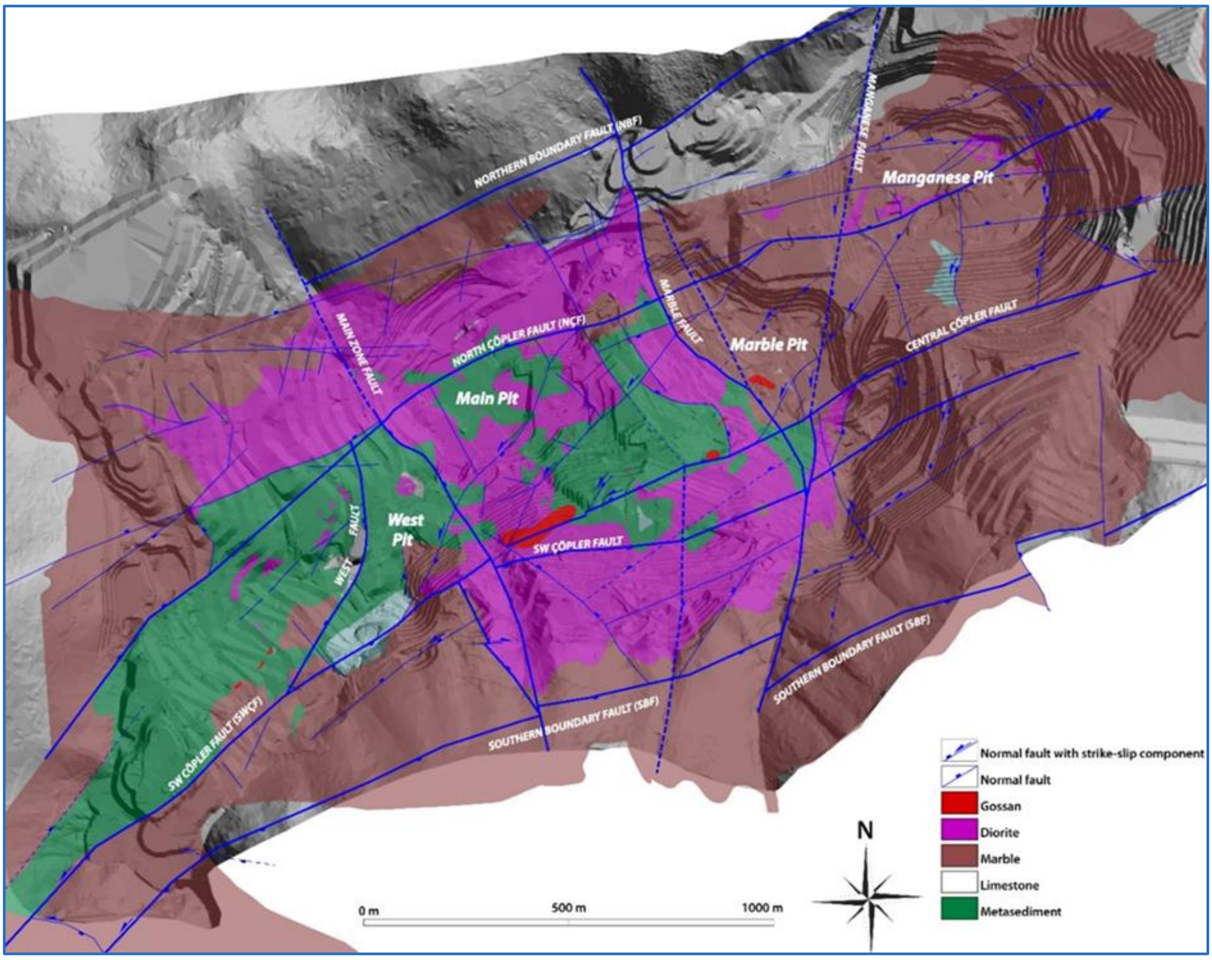
The Çöpler intrusion is a hornblende–quartz diorite-porphyry that shows strong argillic alteration. Some fresh outcrop occurs in the central part of the Main Zone and as remnants within the Manganese Zone. In its least-altered state, the diorite-porphyry is relatively pristine with well-preserved hornblende, biotite, and K-feldspar phenocrysts in a granular matrix of plagioclase and quartz with prominent magnetite. Flow alignment of the hornblende phenocrysts can be seen in places. Gradational transitions to argillic-altered rocks are evident in outcrop and drill core on a centimetre scale.

The primary control on the location of the Çöpler intrusion appears to have been the hornfels-carbonate contact. The contact of the Çöpler intrusion has a roughly rectilinear shape, suggesting control by pre-existing east–north-east trending faults, and by a set of north–north-west trending fractures. The north–north-west striking bedding may also have exerted a local control in the central part of the intrusion where many intrusive contacts are parallel to bedding and have a sill-like morphology. However, it is considered more likely that this reflects the north–north-west trending fracture control referred to above.

A pronounced ground magnetic anomaly is centred on the core of the porphyry, which has been modelled to reflect the potassically altered core of the stock-like barren porphyry system dipping steeply towards the south. In addition, there are several dykes and intrusive apophyses; most notably, a brecciated and strongly clay-altered intrusion centred on the Manganese Zone.

In the area of the Çöpler deposit, two dominant sets of faults are present. These faults are approximately parallel to the long axis of the deposit and are oriented east–north-east. These are referred to as longitudinal faults. The other set of faults are transverse to the longitudinal faults and referred to as cross-faults (Figure 7.3). The major cross-faults include from east to west; Manganese fault, Marble fault, Main Zone fault, and West fault.

Figure 7.3 Çöpler Deposit Geological Map



Anagold, 2020

The longitudinal faults include the Northern Boundary fault (NBF), North Çöpler fault (NÇF), Central Çöpler fault, South-West Çöpler fault, and Southern Boundary fault (SBF). The Central and South-West Çöpler faults dip to the south and were previously thought to be the same fault.

Weathering has resulted in oxidation of the mineralisation close to surface. The oxidised cap is underlain by primary and secondary sulfide mineralisation. In addition to the gold–silver–copper mineralisation of economic interest, arsenic, lead, magnesium, manganese, mercury, and zinc are also present.

7.1.2 Mineralisation – Çöpler Deposit

The gold, silver, and copper mineralisation of economic interest at Çöpler deposit area is exposed in four adjacent open pits from east to west: Manganese pit, Marble pit, Main pit, and West pit. The pits expose economic parts of the same orebody and the three eastern pits will likely join up as the mining progresses. The predominant rock types in the mine include limestone / marble, metamorphic rocks (mainly hornfels) and diorite-tonalite porphyry, locally with equigranular biotite-granodiorite intrusions. Supergene enrichment enhanced along syn-mineralisation and post-mineralisation structures plays an important role in localising high-grade gold mineralisation at lithological contacts, late-stage faults and shear zones, and fault / contact intersections.

Most of the gold mineralisation concentrated in six distinct areas in the deposit: Main, Main West, Main East, Manganese, Marble, and West. The mineralisation is considered to be related to fluids associated with diorite intrusions at depth and generally manifests as three closely related mineralisation styles across the six areas:

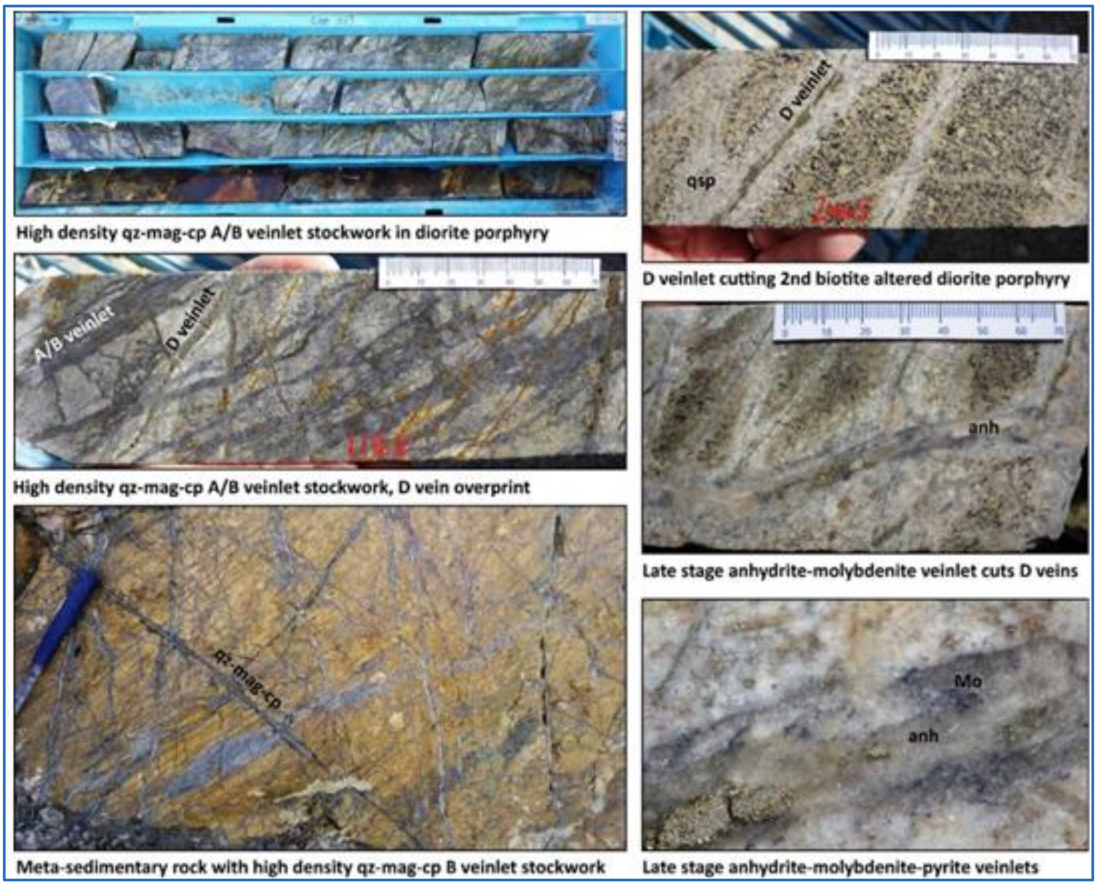
- Low-Grade Porphyry Vein Mineralisation.
- Intermediate Sulfidation Epithermal Mineralisation.
- Iron Skarn and Carbonate Replacement Mineralisation.

7.1.2.1 Three Mineralisation Styles at the Çöpler Deposit

Low-Grade Porphyry Vein Mineralisation

Sub-economic porphyry copper–gold–molybdenum mineralisation is characterised by well-developed alteration zones that are complex and superimposed on each other. Late-stage porphyry mineralisation is hosted in diorite-tonalite porphyry as dominant sheeted veinlet arrays and as stockworks in metamorphic wall rocks and intruded into a gold-deplete diorite-porphyry system (Figure 7.4). Porphyry veinlets are best exposed in the Main pit since the volume of outcropping intrusions is much greater than in other areas of the mine. Early, irregular high-temperature quartz–chalcopyrite–magnetite veinlets are overprinted by ‘D’ veinlets with pyrite±quartz and symmetric feldspar-destructive phyllic halos (Figure 7.4). Dense ‘A’/’B’ veinlets occur as sheeted arrays and lesser stockworks in the intrusions but form well-developed dense stockworks in the surrounding metamorphic wall rocks (Figure 7.4). Late-stage anhydrite veinlets with pyrite and molybdenite appear to overprint the ‘D’ veins, (Tripp, 2017; internal company report).

Figure 7.4 Çöpler Deposit Porphyry Vein Mineralisation

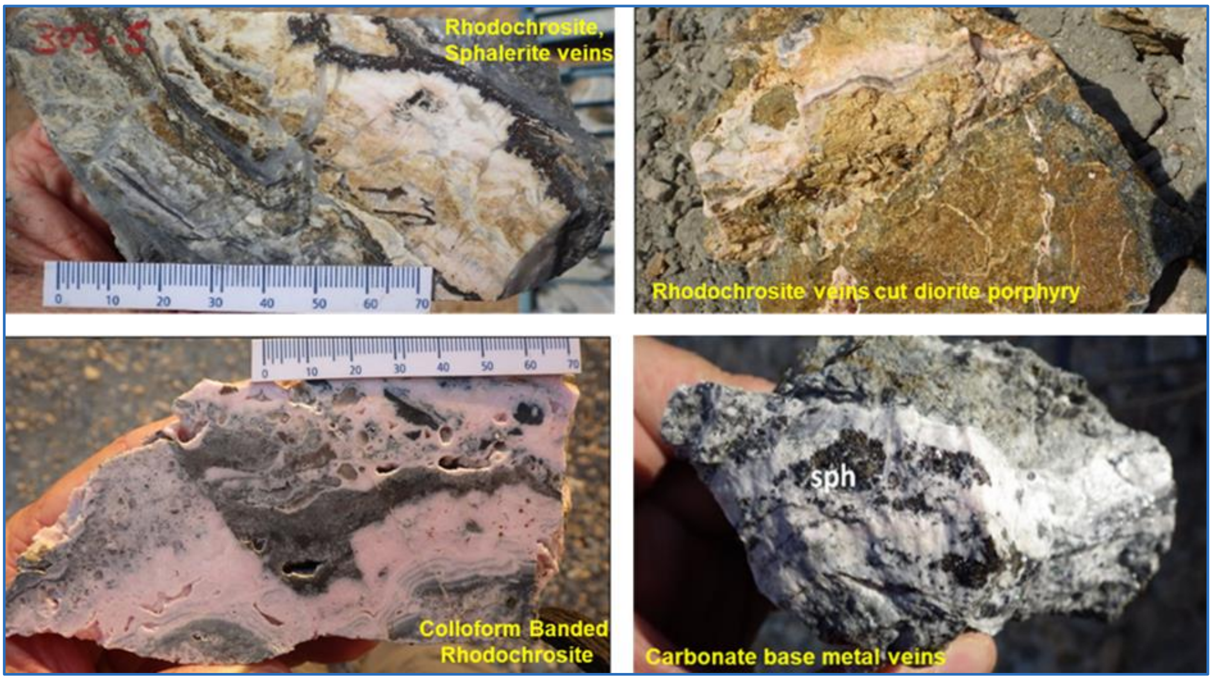


Anagold, 2020

Intermediate Sulfidation Epithermal Mineralisation

Intermediate sulfidation epithermal mineralisation is primarily observed in the Manganese pit as clusters of bright pink, banded, colloform, rhodochrosite base metal sulfide veins and breccia lodes, with a spatial association with elevated gold grades, (Figure 7.5). Carbonate base metal veins contain base metal sulfides sphalerite±galena±chalcopyrite in a gangue of calcite, ferroan dolomite, and/or rhodochrosite and realgar. In the Main pit, the base metal carbonate veins are coarsely crystalline whereas veins in the Manganese pit display brecciation, colloform banding, and locally quartz pseudomorphs of bladed calcite. The change in vein style suggests the Manganese pit represents a higher level position with respect to the mineralising system.

Figure 7.5 Çöpler Deposit Intermediate Sulfidation Epithermal Mineralisation

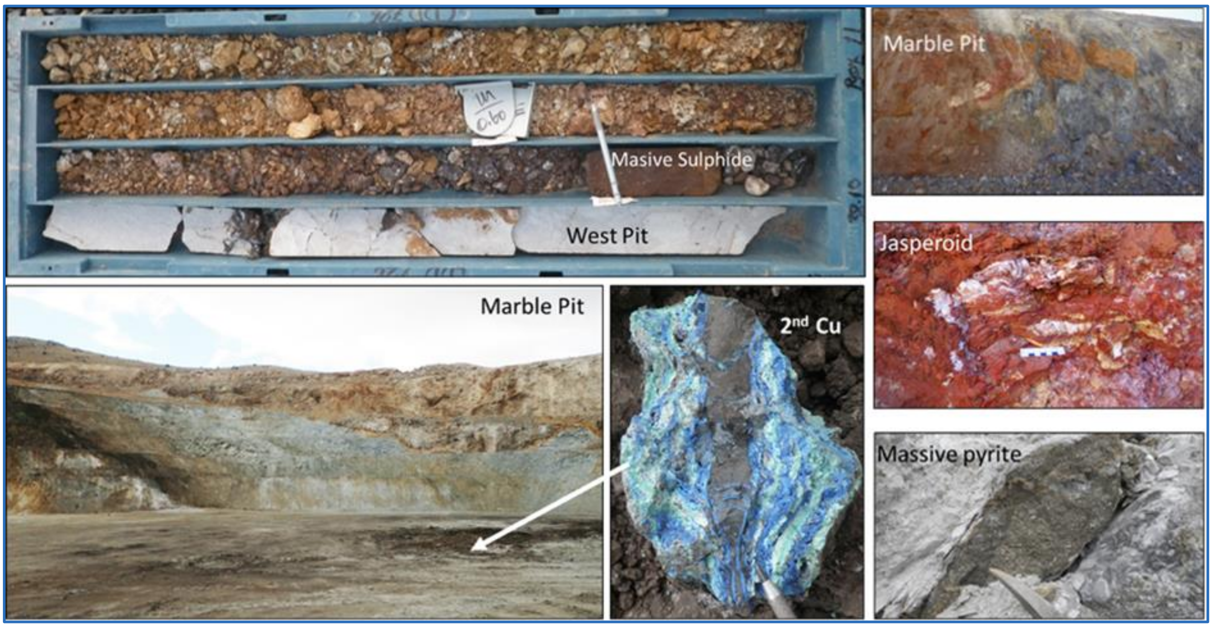


Anagold, 2020

Iron Skarn and Carbonate Replacement Mineralisation

Iron skarn and related carbonate replacement oxide gold mineralisation developed along faults, shear zones, and within karstic spaces. It is observed as iron oxide-rich zones as well as gossan-like and jarosite units developed by oxidation of previous pyrite-rich mineralisation, (Figure 7.6). This replacement type mineralisation appears to be derived from previously formed distal skarn mineralisation. Development of gossan and jasperoid is potentially related to weathering of primary Eocene sulfide deposits in situ or remobilised from a nearby source.

Figure 7.6 Çöpler Deposit Porphyry Vein Mineralisation



Anagold, 2020

7.1.2.2 Six Mineralisation Areas at the Çöpler Deposit

Main Zone Mineralisation

The Main Zone lies in the west portion of the Çöpler deposit and occupies a footprint of approximately 750 m north–south by 1 km east–west. Typical depths of mineralisation range to 200 m below surface. Disseminated quartz–pyrite–arsenopyrite epithermal veinlets are primarily hosted in diorite and metasediment with some marble-hosted mineralisation on the eastern margin of the zone. Oxidation and related oxide mineralisation extends to depths of approximately 40 m from surface, with the thickest oxidised zones proximal to ridges and thinning of strata in the intervening valleys.

Minor volumes of massive pyrite mineralisation occur within the Main Zone.

Main Zone West Mineralisation

Main Zone West is in the north-west corner of the Çöpler deposit at the contact between diorite, marble, and the basement metasediment. This mineralisation is hosted within narrow gossans located at the contact, and in sub-parallel veinlets containing disseminated sulfides within the marble and metasediment. Main Zone West has a strike length of approximately 750 m and is approximately 75 m wide.

Main Zone East Mineralisation

The Main Zone East represents a portion of the mineralisation lying between the Manganese Zone and Main Zone. The geology in this area is typified by narrow, weakly to moderately-mineralised gossans located at the contact between the basement metasediment and the overlying marble. It is postulated that the gossan is sourced from the diorite located in the Manganese Zone and has been emplaced along the metasediment / marble contact as the diorite has crystallised.

Manganese Zone Mineralisation

The Manganese Zone occupies the eastern end of the Çöpler deposit. This zone is approximately 650 m wide north–south by 650 m east–west. The pre-mining surface expression of this area consists predominately of marble. A moderately-sized intrusion of diorite occurs sub-surface. A large proportion of the Manganese Zone mineralisation is associated with the contact between this diorite and the surrounding marble. Mineralisation ranges from surface to approximately 400 m depth.

Free gold mineralisation occurs in the marble with minimal associated sulfides. Disseminated quartz–sulfide mineralisation occurs in clay-altered and brecciated diorites as well as locally carbonate-altered diorite. Moderate volumes of massive sulfide pyrite mineralisation occur within the Manganese Zone. It appears that 'leachable' mineralisation is a combination of free gold in marble and supergene oxidised mineralisation in both marble and diorite. Leachable oxide mineralisation occurs to +200 m below surface.

Marble Zone Mineralisation

The Marble Zone occurs in the south-eastern portion of the Çöpler deposit and is associated with a north-east striking fault contact between marble to the east and metasediment and intrusions to the west. The geology in this area is typified by large 'plugs' of gossan and diorite that have formed at the junctions between large-scale faults, where mineralising fluid flow has been considerable. The width of the Marble Zone is approximately 350 m, and the strike length is 300 m east–north-east. The depth of mineralisation ranges from surface to approximately 160 m below surface.

Mineralisation occurs as both disseminated sulfides in veinlets and massive sulfide along the marble contact. Oxidation has occurred along the north-east structure resulting in greater depths of oxidised mineralisation than that seen in the Main Zone.

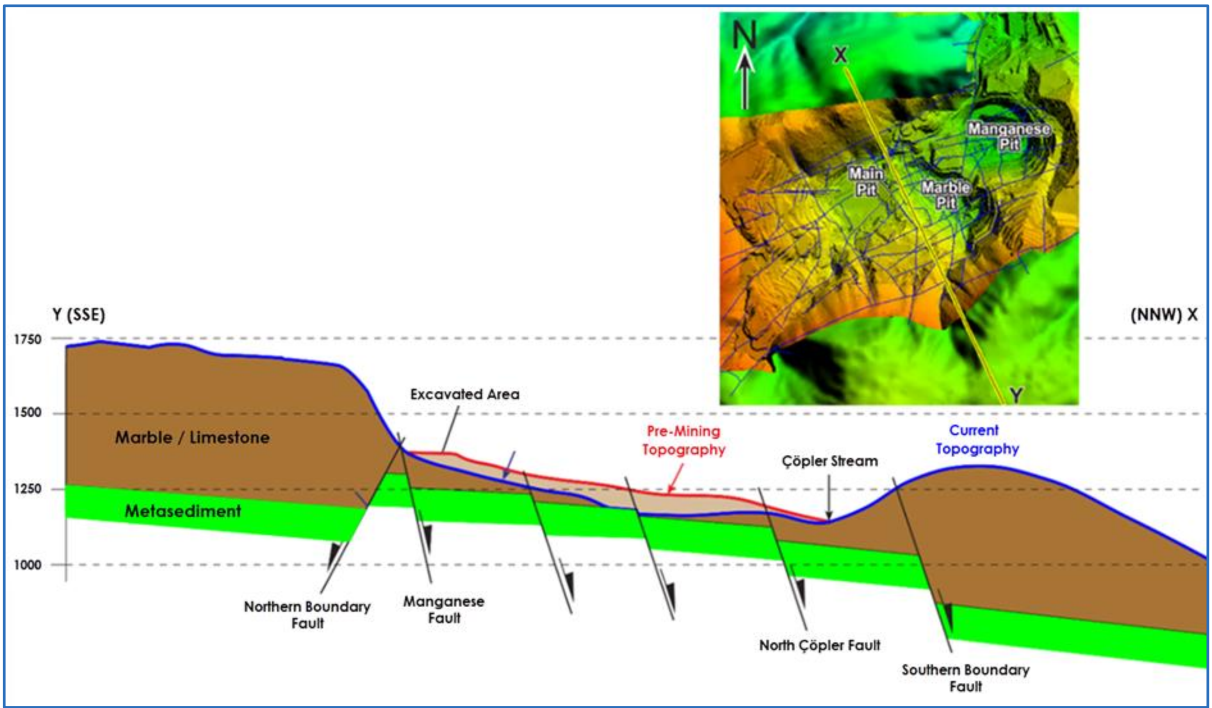
West Zone Mineralisation

The West Zone occupies the westernmost portion of the Çöpler deposit and is located at the contact between the basement metasediment and the overlying limestone/marble, where a large-scale north-east trending fault is located. Mineralisation is present within veinlets containing disseminated sulfides, massive sulfides, and oxidised gossan. The West Zone has a strike length of approximately 700 m north-east and is approximately 150 m wide. Multiple narrow mineralised zones are present sub-parallel to the faulted contact and occur to a depth of approximately 150 m below surface.

7.1.3 Structure – Çöpler Deposit

The Çöpler deposit area demonstrates trans-tensional deformation. The extensional deformation in the area dominates over strike-slip motion as indicated by the lack of compressional structures and the presence of normal movement on all faults. Structurally, the Çöpler deposit occurs in a horst-like feature developed within a sinistral trans-tensional strike-slip setting (Figure 7.7). The two boundary faults delimit the northern and southern extent of the gossan-like, oxidised, supergene, gold-bearing deposits. The northern and southern boundary faults are located almost at the present boundaries of the mine and they dip away from the mine, thereby defining the horst geometry. In addition, the deposit is traversed by several cross-cutting normal faults (with or without strike-slip components) in various orientations that complicate but localise the geometry and position of oxidised ore (Kaymakçı, 2017, internal company reporting).

Figure 7.7 Simplified Schematic of the Çöpler Deposit Structures (cross-section)



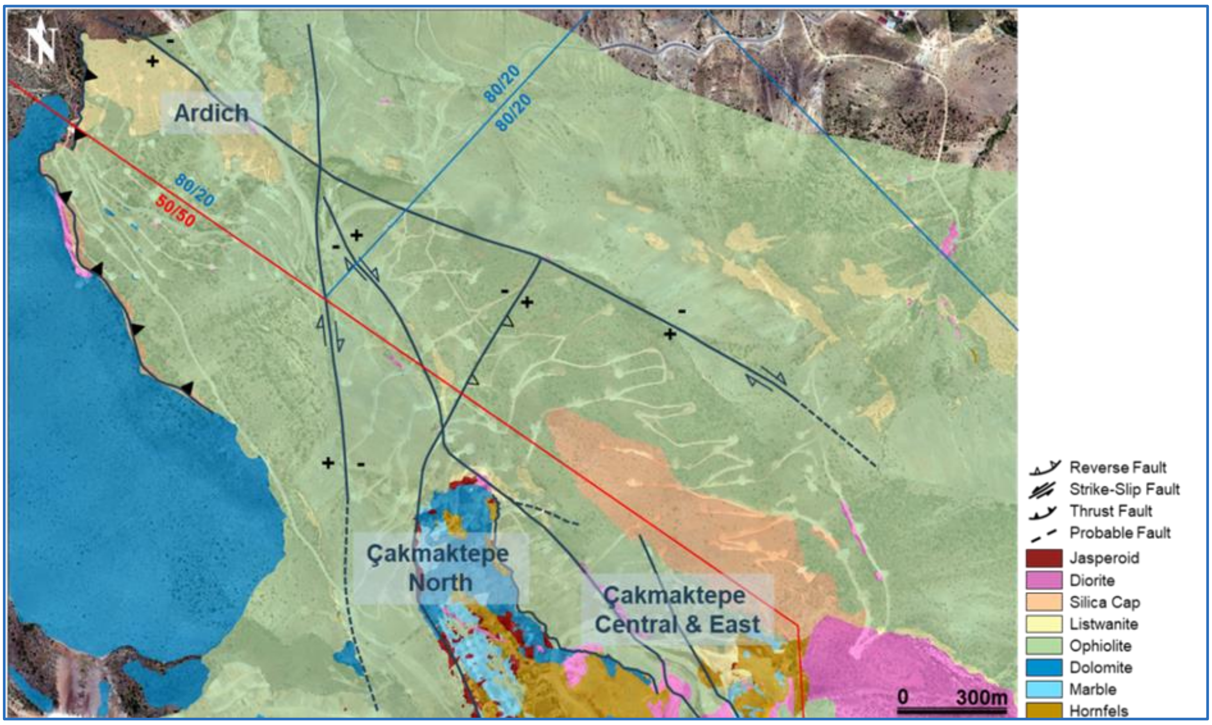
Anagold, 2020

7.2 Geological Setting – Çakmaktepe Deposit

7.2.1 Geology – Çakmaktepe Deposit

The Çakmaktepe deposit is made up of several mineralised zones (Figure 7.8). The deposit area mainly comprises Palaeozoic metamorphic rocks and marble belonging to the Keban Formation and Mesozoic platform carbonate such as the Munzur Formation limestone. All these units are tectonically overlain by ophiolitic mélange rocks. These ophiolitic rocks originated from the northern branch of the Neotethys ocean, the former position of which is delineated by the Ankara–Erzincan suture zone. The emplacement of the ophiolitic units took place at the end of the Upper Cretaceous with north to south motion.

Figure 7.8 Geological Map of the Çakmaktepe and Ardich Deposits



Anagold, 2020

The youngest units include Eocene and younger magmatic rocks, volcaniclastics rocks and sedimentary units that unconformably overlie and seal the Munzur Formation limestone, its basement and the ophiolitic units. All these units are intruded by intermediate igneous rocks that are exposed mainly at the northern and western parts of the Munzur mountains and southern margin of the Sivas Basin.

Listwanite formed in structurally deformed areas by the percolation of CO_2-rich fluids along the margins of ultramafic rocks within the ophiolite complex. Sulfidic jasperoid is present, a result of silica-sulfide metasomatism of Munzur Formation carbonate rocks. Both listwanite and jasperoid are important host rocks for gold and silver mineralisation.

7.2.2 Mineralisation – Çakmaktepe Deposit

The Çakmaktepe deposit is a structurally controlled gold–silver–copper deposit, displaying both epithermal and replacement mineralisation styles. Mineralisation is primarily associated with jasperoid and listwanite. At depth, mineralisation transitions below the base of oxidation to disseminated pyrite, vein sulfides, and massive sulfide horizons, generally occurring within shear zones, along shallow thrusts, in diorite sills, and on intrusion margins.

As with the Çöpler deposit, Çakmaktepe is thought to be the result of intrusive activity that generated suitable conditions for mineralisation of ophiolite, limestone, and hornfels lithologies (Figure 7.9). A complex system of faults enabled emplacement of diorite intrusions and transport of metalliferous fluids associated with the mineralising system. Steeply dipping, shear-hosted mineralisation characterises the deposits at Çakmaktepe North, whereas more shallowly dipping thrust-related mineralisation is characterised at Çakmaktepe East, Çakmaktepe South-East and Çakmaktepe Central. Key to each structurally associated style of mineralisation is the juxtaposition of ophiolites against limestone and hornfels.

Figure 7.9 Geological Mapping within Çakmaktepe Central Pit

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The Çakmaktepe North area is strongly sheared with epithermal characteristics and grade associations with intrusive diorite dykes. The bulk of the mineralisation is structurally confined to a major sub-vertical shear zone (Main Shear). The Main Shear varies in width from 5–40 m, has been defined to a depth of 200–250 m below surface, and dips at approximately 70° to the east. Surface mapping and sampling have defined the mineralised extent of the shear as being over 1 km in length.

Mineralisation at Çakmaktepe North is not solely contained within the shear zone, but also occurs along flat thrust structures and lithological contacts cut by the shear zone. Contacts between ophiolite and limestone, limestone and hornfels, and all lithologies in contact with intrusive diorite sills and dykes are generally mineralised. The listwanite horizon is the most favourable host rock for gold mineralisation. Diorite intrusions show evidence of hydrothermal activity that either takes the form of massive iron-dominated replacement (magnetite, specular hematite, or pyrite) or sheeted crystalline quartz vein bearing jasperoid closer to diorite contacts.

Other mineralised zones within the Çakmaktepe deposit are referred to as 'contact' styles of mineralisation where iron, sulfur, gold, copper, and silver have been emplaced along thrust surfaces where ophiolite is next to limestone and metasediment. Epithermal veining and replacement alteration and textures are prevalent. Skarn and metasomatic mineralisation occur in contact with intrusive diorite dykes, sills, and stocks.

Oxide mineralisation at Çakmaktepe is predominantly characterised by silica–iron–carbonate-rich jasperoid, less-siliceous iron-rich gossan, and epithermal veined and brecciated limestone.

7.3 Geological Setting – Ardich Deposit

7.3.1 Geology – Ardich Deposit

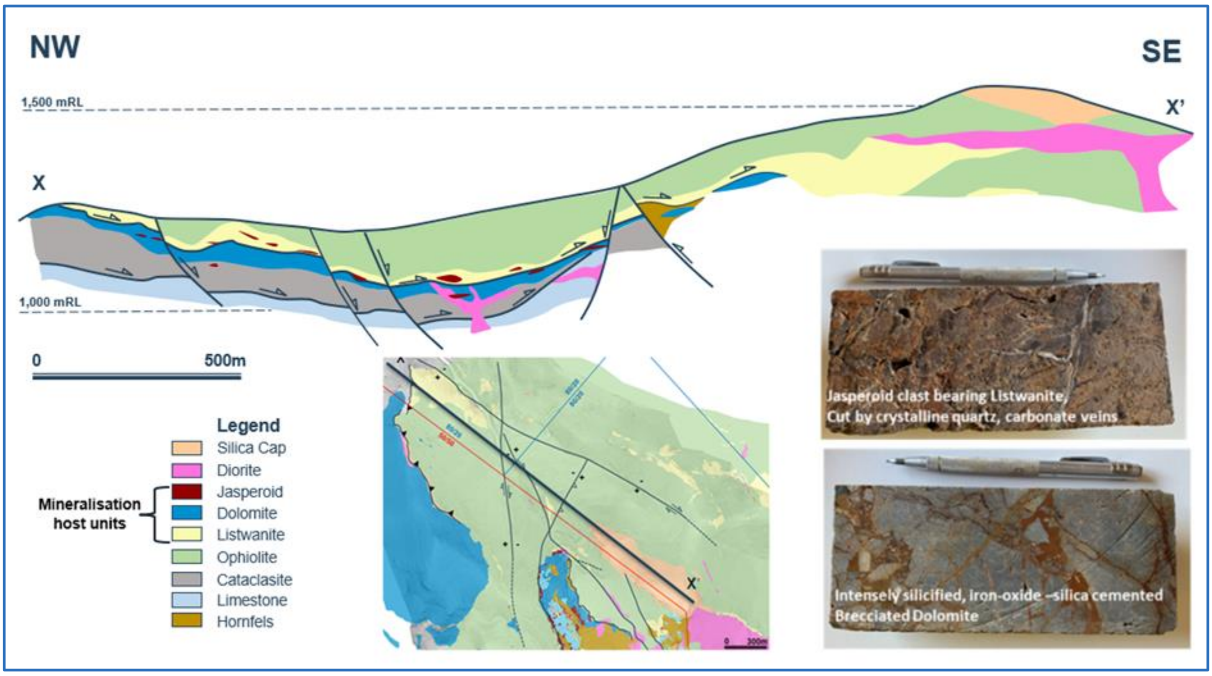
The Ardich deposit is located immediately to the north-west of the Çakmaktepe deposit (Figure 7.8). The north-western portion of Ardich and the Çakmaktepe North mineralised zone are near each other, as are the Ardich South-east and Çakmaktepe East mineralised zones. While there are some characteristic differences between Ardich and Çakmaktepe, the local geology is generally very similar.

The mineralisation at Ardich occurs at a higher stratigraphic level that that seen at Çakmaktepe. The emphasis at Ardich is on the ophiolitic mélange rocks that have been thrust into place on top of the basement metasediment and carbonate lithologies.

The local geology at Ardich is dominated by ophiolite, listwanite, and dolomite and limestone, with mineralisation occurring along low-angle thrust zones between ophiolite, listwanite, and dolomite and limestone (Figure 7.10). This occurs within a complex north-west trending structural zone that is cut by multiple high-angle faults that together result in multiple rotated fault blocks and mineralised zones.

The mineralisation at Ardich is considered to be related to fluids associated with diorite intrusions at depth, much like those observed at the Çöpler and Çakmaktepe deposits. Diorite dykes are present but not common at Ardich, unlike the adjacent Çakmaktepe deposit and nearby Çöpler deposit where diorite is a dominant lithology.

Figure 7.10 Schematic of Ardich Geological Setting with Mineralisation Examples



Anagold, 2020

7.3.2 Mineralisation – Ardich Deposit

The mineralisation at Ardich is related to crystalline and chalcedonic quartz veins within the brecciated and silicified listwanite and dolomite zones. The mineralisation is predominantly in the form of oxide, with sulfide mineralisation confined to limited pyrite-rich jasperoid zones. Clay / gossan in jasperoid or limestone karstic boundaries also contain high-grade gold across Ardich.

Gold grades increase at dolomite / listwanite contacts and within silica-rich listwanite that acts as horizontal traps for higher grade gold-bearing mineralisation. Increases in gold grade can be seen along the lithological contacts. Elevated grades can exist within either contact lithology. Several drillholes show a very rapid down-hole change in gold grade from mineralised to unmineralised material, indicating that mineralisation is tightly constrained instead of disseminated across the deposit. Due to these relationships, the three-dimensional model indicates that the main mineralised zone is tabular and sub-horizontal.

7.4 Geological Setting – Bayramdere Deposit

7.4.1 Geology – Bayramdere Deposit

The Bayramdere deposit is an oxide gold and copper deposit with similar geological and mineralisation characteristics to the Çakmaktepe and Ardich deposits. The geology is characterised by ophiolite thrust over the limestone and dolomite, which are in turn intruded by granodioritic stocks. Gossans are generally observed as lenses and confined by normal faults.

The Bayramdere deposit is structurally controlled, displaying a replacement gold (minor copper, minor silver) mineralisation style. The deposit is dominantly represented by near-surface oxide mineralisation, primarily associated with iron-rich gossan.

The Bayramdere deposit is thought to be the result of intrusive activity that generated suitable conditions for mineralisation. A complex system of faults enabled emplacement of diorite intrusions and transport of metalliferous fluids associated with the mineralising system. Key to each structurally associated style of mineralisation is the juxtaposition of ophiolite against limestone (±hornfels) to create the right geochemical conditions for the deposition of gold and other metals.

7.4.2 Mineralisation – Bayramdere Deposit

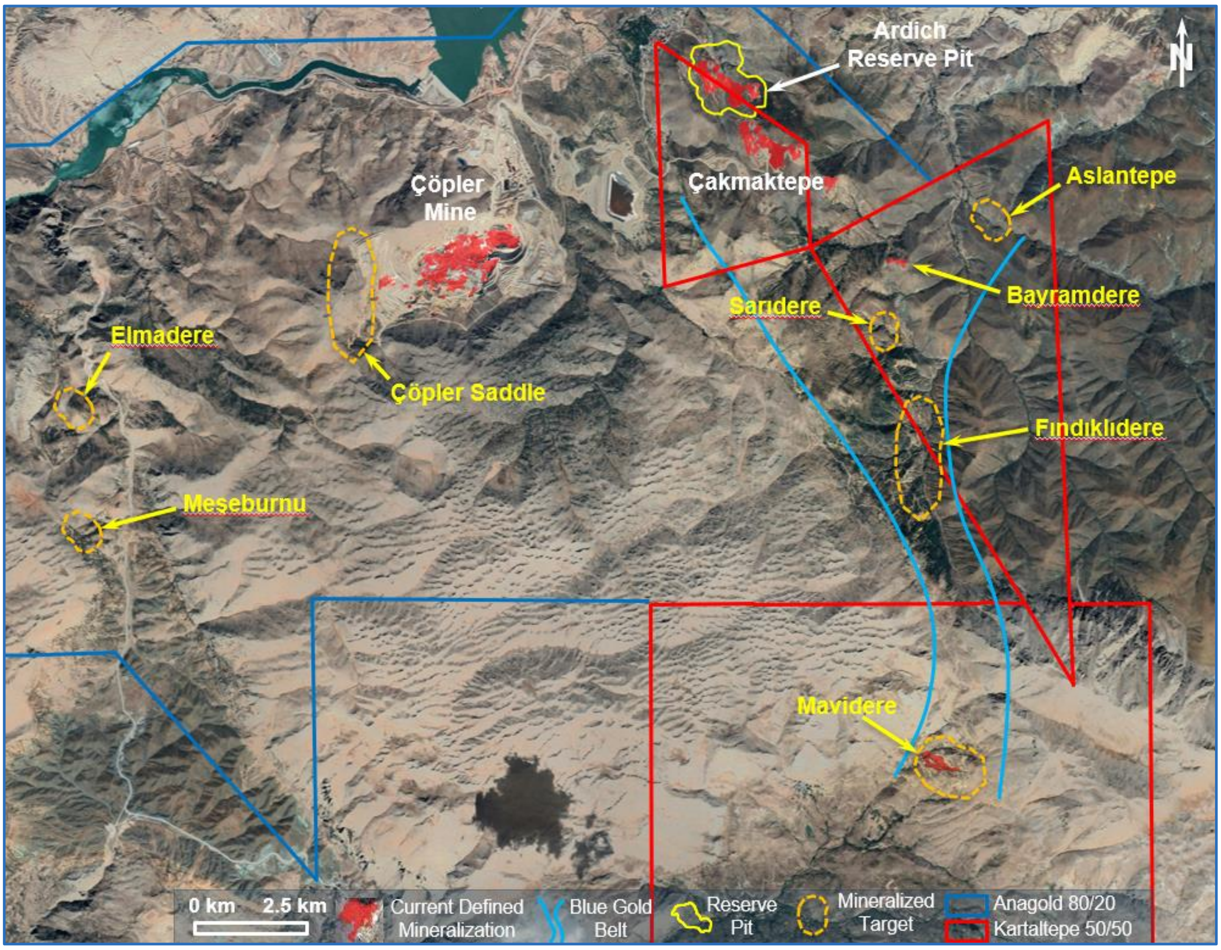
The Bayramdere mineralisation is localised within three stacked, shallow-dipping zones that formed at the contact of limestone and ophiolite, with mineralisation having replaced limestone along the contacts. The limestone / ophiolite contacts are low-angle thrusts, typified by limestone wedges within a dominantly ophiolite stratigraphy. Mineralisation occurs within shallow iron-rich gossan horizons.

7.5 Geological Setting – Regional Prospects and Targets

Since 2000, Anagold exploration programmes within the Çöpler district have identified several new gold-dominant and copper–gold prospects. The gold-dominant regional prospects include the Çöpler Saddle and Elmadere. Copper–gold prospects are Aslantepe, Sarıdere, Findiklidere and Mavidere porphyries located within the Mavialtin Porphyry Belt (Figure 7.11) and the early exploration stage Meşeburnu porphyry located west of the Çöpler deposit.

Each of these prospects is discussed below.

Figure 7.11 Çöpler District Exploration Projects



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7.5.1 Geology – Çöpler Saddle

The Çöpler Saddle prospect borders the western flank of the Çöpler mine. The Çöpler Saddle is associated with a shear zone defined as an arc-like structure that trends north–south for approximately 2 km, Figure 7.11. Along the shear zone, the geology is dominated by limestone, marble, and hornfels units that are in turn intruded by small-scale microdioritic to granodioritic stocks. These lithologies were subjected to silica-clay alteration with iron oxide developments along the local structures as well as clay-pyrite alteration. At the south of the zone, silica is mainly observed as jasperoid lenses, of approximately 2 m long and 1 m wide, which occur along the hornfels and marble contacts. At the centre of the zone, less silica is observed and larger gossan-like mineralised iron oxide bodies have formed.

7.5.2 Geology – Meşeburnu and Elmadere

The Meşeburnu and Elmadere prospects (former Demirmağara project licence group) are located approximately 7 km south-west of the Çöpler deposit (Figure 7.11). The area is covered by ophiolites, limestone, and metamorphic rocks that are intruded by dioritic to granodioritic stocks. Three types of mineralisation have been identified in the area:

- Gold-bearing skarn and jasperoid occurrences along limestone and granodiorite contacts.
- Epithermal gold mineralisation developed along ophiolite, listwanite, and limestone structural contacts (referred to as Elmadere mineralisation).
- Meşeburnu copper–gold porphyry mineralisation.

Gold-bearing skarn and jasperoid occurrences were tested with drilling between 2001–2017, however only short gold-mineralised intervals were intersected. Mapping and sampling in Elmadere and Meşeburnu prospects are ongoing to define drilling targets.

7.5.3 Geology – Mavialtin Porphyry Belt Prospects

The Mavialtin Porphyry Belt is a structural corridor approximately 6–7 km wide and extending over approximately 20 km from the Çakmaktepe deposit in the north to the Mavidere porphyry deposit in the south (Figure 7.11). The Mavialtin Porphyry Belt contains the Mavidere, Findiklidere, Saridere, and Aslantepe porphyry copper–gold prospects.

7.5.3.1 Geology – Mavidere

The Mavidere porphyry copper–gold mineralisation is hosted by hornblende–biotite monzonite to monzogranite to granodioritic phases of a shallow porphyritic intrusive hosted by metamorphic and crystallised limestone. At the centre of the porphyry system, the intrusive phases were subjected to mainly potassic alteration with clay and minor sericite overprinting covering an area of 800 m x 400 m. The porphyry system appears to continue underneath the moraine cover to the east and south.

Previous exploration activities included:

- surface mapping,
- geochemistry (soil, rock, stream sediment sampling),
- geophysical studies (Induced Polarisation (IP) and surface magnetics), and
- Reverse circulation (RC) and diamond core (DD) drilling.

The prospect was first drilled in 2001, with 1,780 m at eight locations. In 2008, 22 additional holes were drilled totalling 7,761 m, with the preliminary results announced in 2009. From 2011 through 2013, 77 DD holes totalling 20,653.3 m and 68 RC holes totalling 7,512 m were completed. Field studies and mapping in 2018 identified additional mineralised zones, some of which were drill tested in 2018 and 2019. Drillhole MD06, drilled in 2019, returned a highly prospective intercept of 269.1 m at 0.34% Cu and 0.55 g/t Au from surface.

7.5.3.2 Geology – Aslantepe

The geology of the Aslantepe porphyry copper–gold prospect is dominated by ophiolites thrusted over Jurassic to Cretaceous limestone, both of which are intruded by dioritic to granodioritic stocks and dykes. The Aslantepe intrusives outcrop in a narrow corridor subjected to propylitic, potassic, and clay alteration. The potassic zone is characterised by well-developed intense quartz–sulfide stockwork veinlets with secondary biotite, K-feldspar, and magnetite. In 2018, two additional DD holes were drilled at Aslantepe, with drillhole AT07 intersecting 63.9 m at 0.22% Cu and 0.45 g/t Au from 46.7 m down-hole. The mineralisation appears to be dipping underneath the ophiolites.

7.5.3.3 Geology – Sarıdere

The Sarıdere porphyry copper–gold prospect is covered by metamorphic limestone and ophiolite, which are in turn intruded by tonalitic to granodioritic stocks. The prospect was initially identified by stream sediment and soil anomalies. In 2018 and 2019, exploration activities identified potassic-altered porphyry intrusive outcrops covering an area of approximately 800 m x 500 m, with a phyllic alteration halo around the potassic zone of 4.3 km x 0.6 km. Seven DD holes totalling 1,461.5 m were drilled from 2007 through 2013 at the margin of the porphyry system, testing the elevated soil geochemistry. These holes intersected short intervals of copper–gold mineralisation.

7.5.3.4 Geology – Fındıklıdere

The Findiklidere porphyry copper–gold prospect is covered by massive Jurassic to Cretaceous limestone, which has been over-thrusted by ophiolites on the eastern flank. These units were intruded by fine to medium-grained tonalitic to granodioritic intrusive stocks. The porphyry copper mineralisation is characterised by well-developed stockwork quartz–magnetite–pyrite veins with copper. Peripheral iron–copper–gold skarns are observed within the limestone. In 2018, the geology, structure, and alteration were re-mapped to better understand the porphyry potential of the prospect. Results of this field work indicated that the porphyry mineralisation was potentially continuing underneath the ophiolitic body to the south-west of the known porphyry mineralisation. In 2019, DD hole FD02 was drilled to test porphyry potential beneath the ophiolitic cover. The hole was mineralised over 234.4 m (down-hole) with some higher grade intervals such as 32.1 m at 0.84% Cu and 0.37 g/t Au from 13.4 m and 16.5 m at 1.27% Cu and 0.07 g/t Au from 139.5 m.

The abovementioned drilling results were announced within the exploration press release dated 14 February 2020.

8 DEPOSIT TYPES

Porphyry copper–gold systems host some of the most widely distributed mineralisation types at convergent plate boundaries, including porphyry deposits centred on intrusions; skarn, carbonate-replacement, and sediment hosted gold deposits in increasingly peripheral locations; and high to intermediate-sulfidation epithermal deposits.

The alteration and mineralisation in porphyry copper–gold systems are zoned outward from the stocks or dyke swarms, which typically comprise several generations of intermediate to felsic porphyry intrusions. Porphyry copper (± gold, ± molybdenum) deposits are centred on the intrusions, whereas carbonate wall rocks commonly host proximal copper–gold skarns, less common distal zinc–lead and/or gold skarns, and, beyond the skarn front, carbonate-replacement copper and/or zinc–lead–silver (± gold) deposits, and/or sediment-hosted (distal-disseminated) gold deposits. Peripheral mineralisation is less conspicuous in non-carbonate wall rocks but may include base metal-bearing or gold-bearing veins and mantos (Sillitoe, 2010). Skarn deposits are typically hosted in mineralogically simple fine-grained clastic and carbonate sedimentary rocks. Skarn mineralogy and metal content is largely dependent on the crystallisation history and genesis of associated plutons (Meinert et al., 2005).

The Çöpler district is located at the edge of a convergent plate boundary. It is characterised by a complex structural history and is associated with intermediate intrusive and carbonate-rich host lithologies. As such, porphyry copper–gold systems and related styles of mineralisation are appropriate models to be applied across the Çöpler district.

The Çöpler deposit consists of three major mineralisation types that are closely associated with each other: low-grade sub-economic porphyry copper–gold–molybdenum mineralisation characterised by well-developed alteration zones and stockwork quartz veins (Main Zone); intermediate sulfidation epithermal mineralisation observed in the Manganese Zone as clusters of bright pink, banded, colloform rhodochrosite base metal sulfide veins and breccia lodes; and iron–gold (± copper) skarn with related carbonate replacement gold mineralisation.

The setting, alteration mineralogy, and mineralisation characteristics of the Manganese Zone are somewhat consistent with an intermediate sulfidation epithermal system, as defined in Hedenquist et al., (2000).

Exploration programmes modelled on epithermal-style deposits have shown success in the Çöpler district. A multi-phase porphyry model with a barren trapping system and a possible mineralised porphyry underneath it is also applicable.

9 EXPLORATION

9.1 Exploration – Çöpler Deposit

Exploration of the Çöpler deposit has been conducted by Anagold and its predecessors since September 1998. Work completed has included:

- Geological and reconnaissance mapping.
- Rock chip, grab, soil, channel, and stream sediment geochemical sampling.
- Ground geophysical surveys including ground magnetic, complex resistivity / IP, time domain IP and controlled source audio-frequency magneto-telluric (CSAMT) surveys.
- A regional helicopter-borne geophysical survey.
- Reverse circulation (RC) and diamond core (DD) drilling programmes.
- Acquisition of satellite imagery.
- Mining technical studies.
- Geotechnical and hydrogeological studies.
- Environmental baseline studies.
- Studies in support of project permitting.
- Metallurgical testwork and studies.
- Condemnation evaluations.

The principal exploration technique used at Çöpler has been RC and DD drilling, conducted in multiple campaigns since 2000. Initially, exploration was directed at evaluating the economic potential of the near-surface oxide mineralisation for the recovery of gold by either heap leaching or conventional milling techniques.

In 2013, drilling occurred primarily in the western portion of the Main Zone and on the northern edge of the Çöpler deposit. Drilling during 2014 focused on verification of existing drilling results through a twin hole programme. Drilling in 2015 provided data coverage at depth in the Manganese Zone, infill drilling in the Main Zone, and testing of low-sulfur mineralisation below the oxidation boundary.

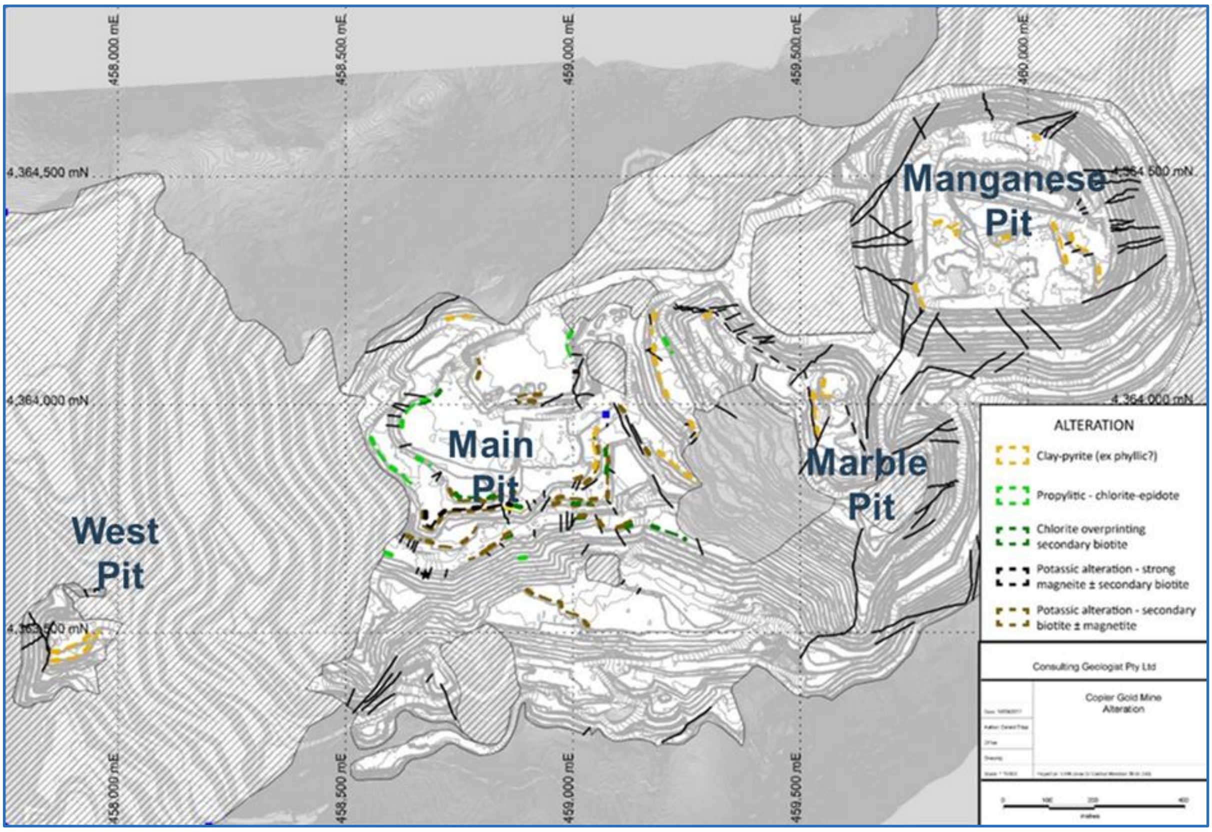
Drilling continues to better define both the oxide and sulfide portions of the Çöpler deposit.

9.1.1 Geological Mapping – Çöpler Deposit

Surface mapping and sampling has been undertaken over the life of the project, culminating in a detailed geological map of the Çöpler valley, shown in Figure 9.2.

Geological mapping is used in support of exploration vectoring, exploration activities, infrastructure locations, mine planning, and environmental monitoring. One of the aims of the mapping studies is to provide sufficient information to define mineralisation types and structural settings for the Çöpler deposits. Alteration zones, such as the high-temperature porphyry alteration preserved in the southern wall of the Main Zone (shown in Figure 9.1), were identified through detailed bench wall mapping during the target generation programmes.

Figure 9.1 Çöpler Deposit Map of Alteration Minerals



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9.1.2 Geochemical Sampling – Çöpler Deposit

Extensive sampling programmes have been, and continue to be, conducted within the Çöpler area, leading to the identification of significant gold anomalies including the near-mine discovery of the Çöpler Saddle on the western flank of the Çöpler mine.

9.1.3 Geophysics – Çöpler Deposit

Various ground and airborne geophysical surveys have been conducted at the Çöpler deposit as well as across the wider Çöpler district since mid-2000. Surveys carried out include ground magnetic, complex resistivity / Induced Polarisation (IP), time domain IP, and CSAMT surveys, as well as a regional helicopter-borne aeromagnetic survey that included the broader Çöpler district.

Physical property measurements are collected regularly on outcrops and DD core samples, including magnetic susceptibility, resistivity, and chargeability. Additionally, four samples from DD hole CDD067 were sent to Systems Exploration in Australia for a detailed physical property analysis.

9.2 Exploration – Çakmaktepe Deposit

The Çakmaktepe deposit and surrounding mineralised zones were identified by stream sediment samples with elevated gold geochemistry.

Drilling at Çakmaktepe started in 2012. The recent drilling (2019 onwards) has been designed to improve the known Mineral Resources identified at Çakmaktepe North. Data collected to date includes magnetic geophysical surveys, outcrop and bench wall mapping, rock and soil sampling, and both RC and DD drilling.

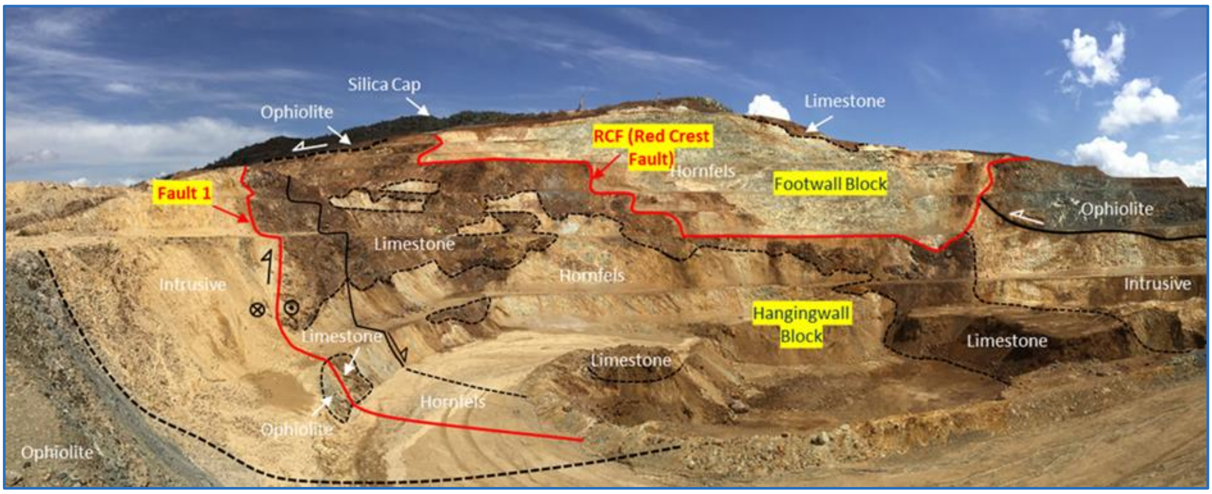
9.2.1 Geological Mapping – Çakmaktepe Deposit

The first geological mapping study in the area was conducted in 2000.

Mapping in 2014–2016 focused on deposit-wide surface geology definition at a scale of 1:1,000, reducing to 1:500 scale for the Çakmaktepe Mineral Resource area. The establishment of a network of drill tracks and pads on the sides of hills and ridges resulted in new rock exposures that have been subjected to detail geological mapping. Mapping included the collection of lithological, alteration, geochemical, and structural data.

An additional mapping study within the Çakmaktepe deposit was initiated as the Çakmaktepe operation advanced in late-2018. Details from the bench walls were collected and integrated into the drillhole dataset (mapping example shown in Figure 9.2). This has resulted in a more-accurate geological model for further pit extension exploration drilling.

Figure 9.2 Çakmaktepe Deposit – Example East Pit Geological and Structural Map



Anagold, 2020

9.2.2 Geochemical Sampling – Çakmaktepe Deposit

Geochemical sampling programmes at Çakmaktepe were initiated in 2014 and included rock chip and soil sampling (Table 9.1). Geochemical sampling was also used to define areas of alteration and mineralisation that inform additional detailed sampling surveys.

Table 9.1 Number of Geochemical Samples within the Çakmaktepe Deposit

Year	Rock Chip Samples	Soil Samples
2014	661	341
2015	3,527	–
2016	356	270
2017	63	1,638
2019	540	–
2020	13	–
Total	5,160	2,249

A total of 5,160 rock chip samples have been collected from the Çakmaktepe deposit since 2014. During 2019, rock chip sampling extended into bench wall and haulage roadsides to help define the extents of the deposit more accurately.

Soil sampling programmes were initiated during the 2010 exploration programme. The deposit has been fully covered with a 50 m x 50 m sampling grid totalling 2,249 samples.

Stream sediment sampling was carried out on a regional scale as part of target generation programmes since 2002. A total of 851 sediment samples have been collected.

9.3 Exploration – Ardich Deposit

Exploration activities across the Ardich deposit began in 2017 and included geological mapping, geochemical sampling, and DD drilling programmes.

9.3.1 Geological Mapping – Ardich Deposit

The Ardich deposit was discovered in 2017 during detailed geological mapping and rock sampling programmes. Results of the mapping study highlighted the potential of the Ardich deposit and its extension to the south. The mineralisation identified to date continues approximately 4 km on a north westerly trend.

9.3.2 Geochemical Sampling – Ardich Deposit

Geochemical sampling programmes at Ardich have included rock chip / channel and soil sampling, (Table 9.2). Most of the geochemical sampling campaigns across the Ardich deposit were designed based on findings from the geological mapping programmes.

Table 9.2 Number of Geochemical Samples within the Ardich Deposit

Year	Rock Chip/Channel Samples	Soil Samples
2017	175	125
2018	912	–
2019	880	1,718
2020	140	–
Total	2,107	1,843

A total of 2,107 rock chip / channel samples have been collected since 2017 from outcrops across the Ardich deposit. Rock chip / channel sampling has been the most representative surface sampling, collected directly from altered rock exposures. As the drilling programmes continue, newly opened drill tracks and pads give good access to new rock exposures that are subjected to rock sampling and geological mapping.

Soil sampling was completed in early-2000 as part of a regional geochemical reconnaissance programme, with early targets being potentially mineralised listwanite-capped faults. Anagold started regional systematic soil sampling on 200 m x 200 m grids to cover all tenements in 2011. At the Ardich deposit, a total of 1,843 soil samples were collected on a sampling grid of 50 m x 50 m, which was reduced to 25 m x 25 m in gold-anomalous areas in 2017–2019.

10 DRILLING

All drillhole counts in this section include holes drilled for resource definition, geotechnical, and metallurgical purposes.

10.1.1 Drilling – Çöpler Deposit

The Çöpler deposit continues to be tested by reverse circulation (RC) and diamond core (DD) drilling. The details of drillholes utilised in this Mineral Resource update for the Çöpler deposit are presented in Table 10.1. Typically, the drillhole spacing at surface is a nominal 50 m, however, in some areas the drill spacing has been reduced to 25 m (Figure 10.1).

Step-out drilling at the Çöpler deposit has defined most of the lateral boundaries of the mineralisation. There has been additional development drilling, as well as condemnation drilling of areas planned for infrastructure during the last few years. In order to improve confidence in the short-range mine planning, infill drilling programmes have been conducted since 2007.

Drilling in 2014 focused on confirmation of the mineralisation with a twin hole programme.

Development drilling continued in 2015 by improving sample coverage at depth in the Manganese Zone and along structural boundaries in the Main Zone. In addition to the drilling of in situ mineralisation, a stockpile drilling programme began in December 2015 to confirm sulfide stockpile ore grade, grade distribution, and mineralogy.

Drilling at Çöpler between 2016 and 2020 mainly concentrated on target generation to increase the amount of oxide material for the production portfolio. This was focused on the Main Zone, West Zone, and the Çöpler Saddle areas. More specifically, the programme aimed to test continuation of the main gold-bearing structures based on a re-interpretation of the Çöpler structural and mineralisation settings. In-pit drilling campaigns continue with extensive exploration programmes to define additional oxide gold potential.

Figure 10.1 Drillhole Collar Location Plan – Çöpler Deposit

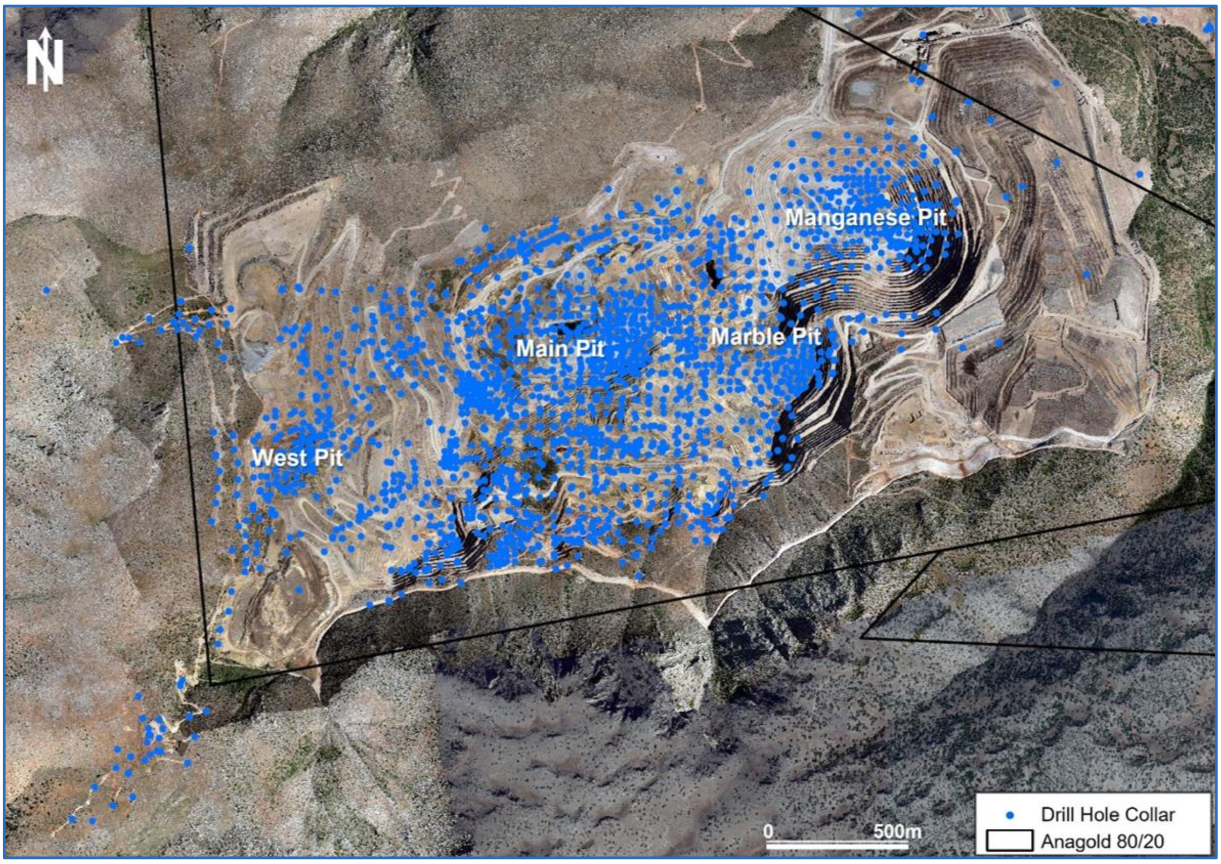


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Table 10.1 Drilling History – Çöpler Deposit

Year	Hole Type	Number of Holes	Metres Drilled	Total Metres / Year
2000	DD	4	971.5	971.5
2001	DD	10	2,254.4	6,320.3
	RC	32	4,065.9	
2002	DD	31	6,575.6	6,835.6
	RC	1	120.0	
	Other	2	140.0	
2003	DD	33	2,975.7	2,975.7
2004	DD	37	4,413.5	16,634.8
	RC	228	11,036.0	
	Other	16	1,185.3	
2005	DD	24	4,776.4	35,062.1
	RC	177	29,009.7	
	Other	16	1,276.0	
2006	DD	17	2,102.6	15,857.6
	RC	94	12,878.0	
	Other	24	877.0	
2007	DD	74	16,513.2	34,435.9
	RC	125	16,998.5	
	Other	40	924.2	
2008	DD	35	5,059.4	9,963.4
	RC	41	4,904.0	
2009	DD	23	5,789.5	10,135.5
	RC	34	4,346.0	
2010	DD	14	1,916.1	2,060.6
	RC	1	144.5	
2011	DD	115	29,359.0	47,342.0
	RC	150	17,983.0	
2012	DD	145	50,156.5	64,041.0
	RC	120	13,884.5	
2013	DD	126	33,040.9	37,585.9
	RC	53	4,545.0	
2014	DD	12	1,296.5	1,296.5
2015	DD	59	6,214.1	12,778.1
	RC	69	6,564.0	
2016	DD	148	3,826.5	6,020.5
	RC	94	2,194.0	
2017	DD	41	3,370.5	3,370.5
2018	DD	109	10,745.0	10,745.0
2019	DD	62	7,607.7	7,607.7
2020	DD	131	23,029.90	23,029.90
2021	DD	68	18,491.80	18,491.80
Total	RC	1,219	128,673.1	
	DD	1,318	240,486.3	
	Other	98	4,402.5	
	All Types	2,635	373,561.9	

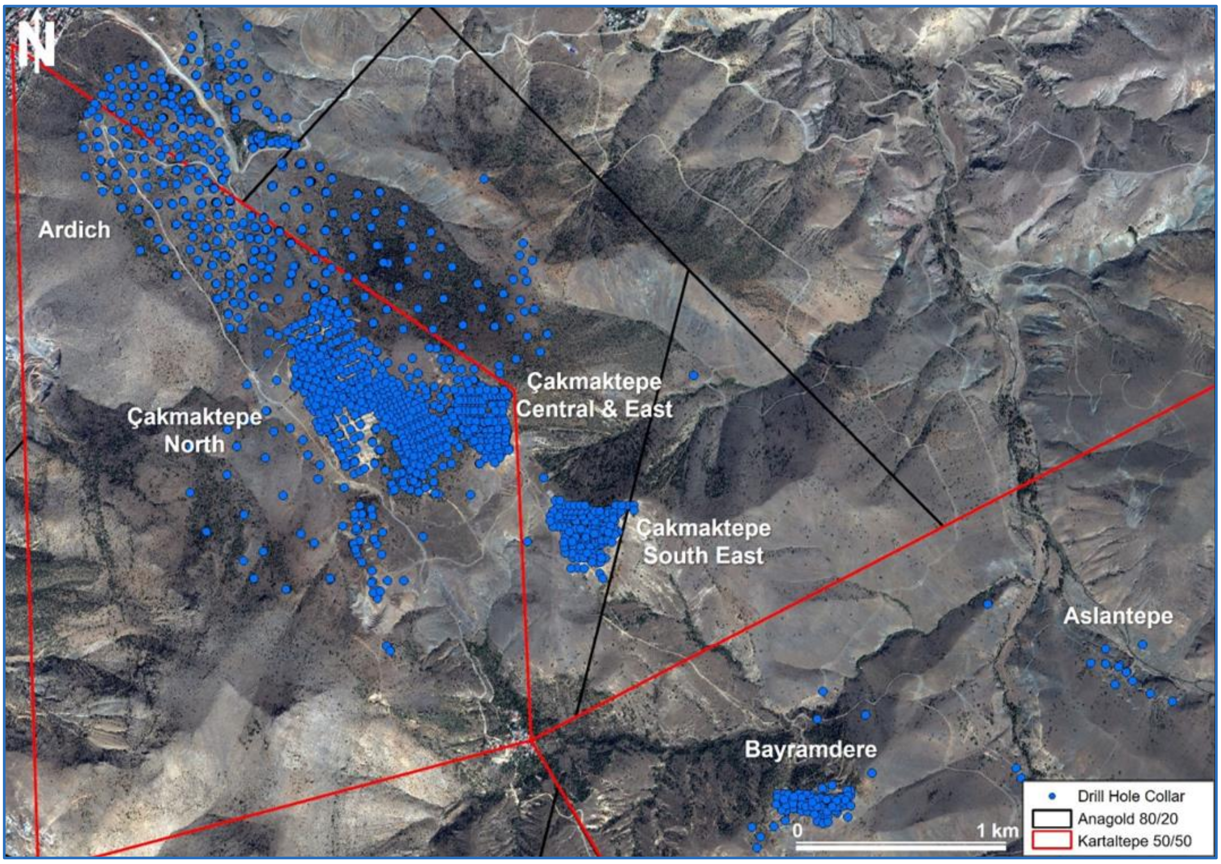
10.1.2 Drilling – Çakmaktepe Deposit

A total of 1,183 drillholes have been drilled at the Çakmaktepe deposit since 2012. This included 528 RC holes, 570 DD holes, and the remainder a mixture of RC and DD. As production proceeded within the Çakmaktepe Central and Çakmaktepe East pits, additional targets were generated to provide push-back options within the pit design. A total of 136 DD holes have been completed since 2019 to test for continuation of the Çakmaktepe deposit, Figure 10.2 and Table 10.2.

Table 10.2 Drilling History – Çakmaktepe Deposit

Year	Number of Drillholes	Drilled Metres
2012	21	2,287.5
2013	7	962.0
2014	162	15,976.7
2015	256	21,463.2
2016	485	64,108.6
2017	116	9,366.2
2019	75	5,919.4
2020	61	8,702.3
Total	1,183	128,785.9

Figure 10.2 Drillhole Collar Location Plan – Ardich, Çakmaktepe, and Bayramdere Deposits



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10.1.3 Drilling – Ardich Deposit

A total of 531 DD holes have been drilled at the Ardich deposit since late-2017, Figure 10.3 and Table 10.3. After the initial discovery of the Ardich deposit, DD drilling programmes have continued to better-define the mineralisation and to improve the Mineral Resource estimates. Drilling to obtain samples for metallurgical testing and hydrogeological studies has also been undertaken at Ardich.

A total of 233 drillholes were included in the previously-announced Ardich Mineral Resource (CDMP20TR, drillholes AR1–AR233). Since the data cut-off date for the CDMP20TR Mineral Resource update, data has been obtained for an additional 129 drillholes (AR233–AR531).

A drillhole collar plot is shown in Figure 10.3, indicating the various generations of drilling.

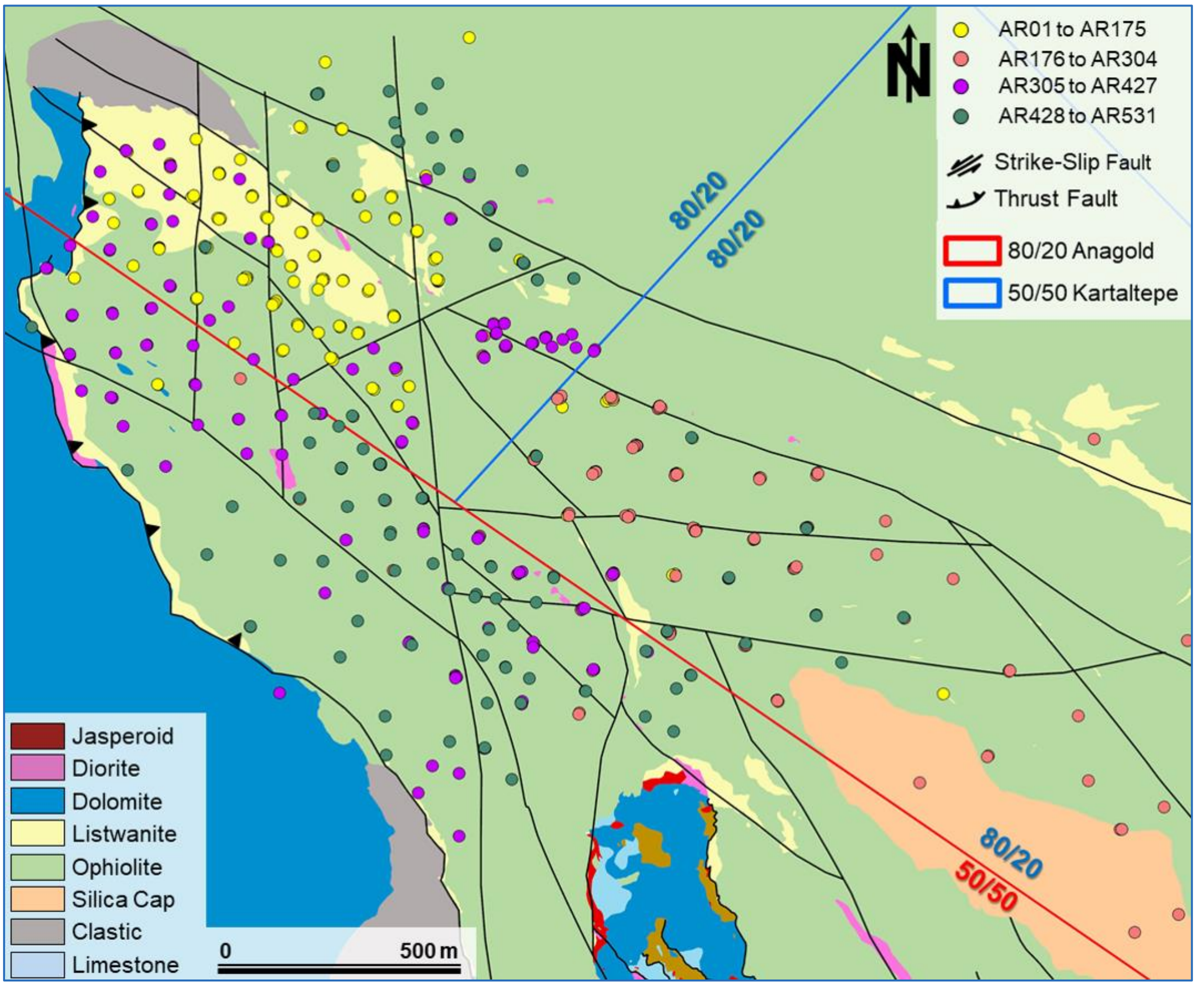
The target of the post-2020 drilling has been two-fold:

- Infill drilling within the bounds of the 2020 resource model area.
- Step-out drilling to the west, south, and south-west of the 2020 resource area.

Table 10.3 Drilling History – Ardich Deposit

Year	Number of Drillholes	Drilled Metres
2017	9	1,374.10
2018	91	14,216.40
2019	133	27,821.20
2020	147	35,146.65
2021	151	32,586.00
Total	531	111,004.35

Figure 10.3 Drillhole Collar Location Plan – Ardich



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Drillholes AR1 through AR427 have contributed to updated (2021) resource modelling for Ardich, which is discussed in Section 14.3. The 2021 update resulted not only in a larger inventory than that previously-announced but is also a higher confidence inventory. The data cut-off date for updated Ardich resource model was 31 May 2021.

10.1.4 Drilling – Mavialtin Porphyry Belt Prospects

Drilling within the Mavialtin Porphyry Belt first started in early-2000. Re-interpretation of historical drillholes and detailed mapping programmes resulted in the definition of new drill targets in subsequent years. A total of 353 holes have been completed between 2001–2020 at various targets within the Mavialtin Porphyry Belt, Figure 10.4 and Table 10.4.

10.1.5 Grid Coordinate Systems

The Çöpler project uses the European 1950 (E1950) datum coordinate system – this is a Turkish Government requirement.

The Çöpler project is in UTM6 zone 37N of the E1950 coordinate system. Until 2014, drillhole collars were surveyed by the mine surveyors in the E1950 UTM3 coordinate system and then converted to E1950 UTM6 before making them available to other personnel. The conversion from UTM3 to UTM6 was achieved by subtracting 1,746 m (–1,746 m) from the UTM3 northing coordinate and adding 17 m (+17 m) to the UTM3 easting coordinate. There is no rotation, scaling, or change in elevation between the E1950 UTM3 and E1950 UTM6 systems. Since March 2014, collar coordinates have been and are being collected in the ED1950 UTM6 coordinate system.

10.1.6 Collar and Down-hole Surveys

Up until 2014, drillhole collars were surveyed by Anagold surveyors using a Topcon differential global positioning system (DGPS) instrument. Approximately 4% of the drillholes up to 2014 have planned collar locations, rather than surveyed collar data. After 2014, the exploration department managed the collection of collar survey coordinates with the use of a differential GPS (DGPS). All collar survey data is checked prior to being stored within the corporate drillhole database.

Down-hole surveys are collected for all drillholes. Prior to 2009, down-hole surveys were undertaken using a Reflex Instruments Limited (Reflex) single shot down-hole camera. In 2009, a Reflex multi-shot down-hole camera was introduced on the project. Drilling contractors upgraded to a Reflex EZ Trac tool for down-hole survey data collection through to the end-of-2017, thereafter, to date the majority of the drillholes have been down-hole surveyed using Reflex S Process V2.5.0650 and Devico PeeWee. Survey measurements were taken every 10 m down-hole, and data provided with raw files to record quality assurance and quality control (QA/QC) for each survey.

The depth of the surveys varies between drillholes and is dependent on the depth and angle of the drillhole.

Figure 10.4 Drillhole Collar Location Plan – Mavialtin Porphyry Belt Prospects

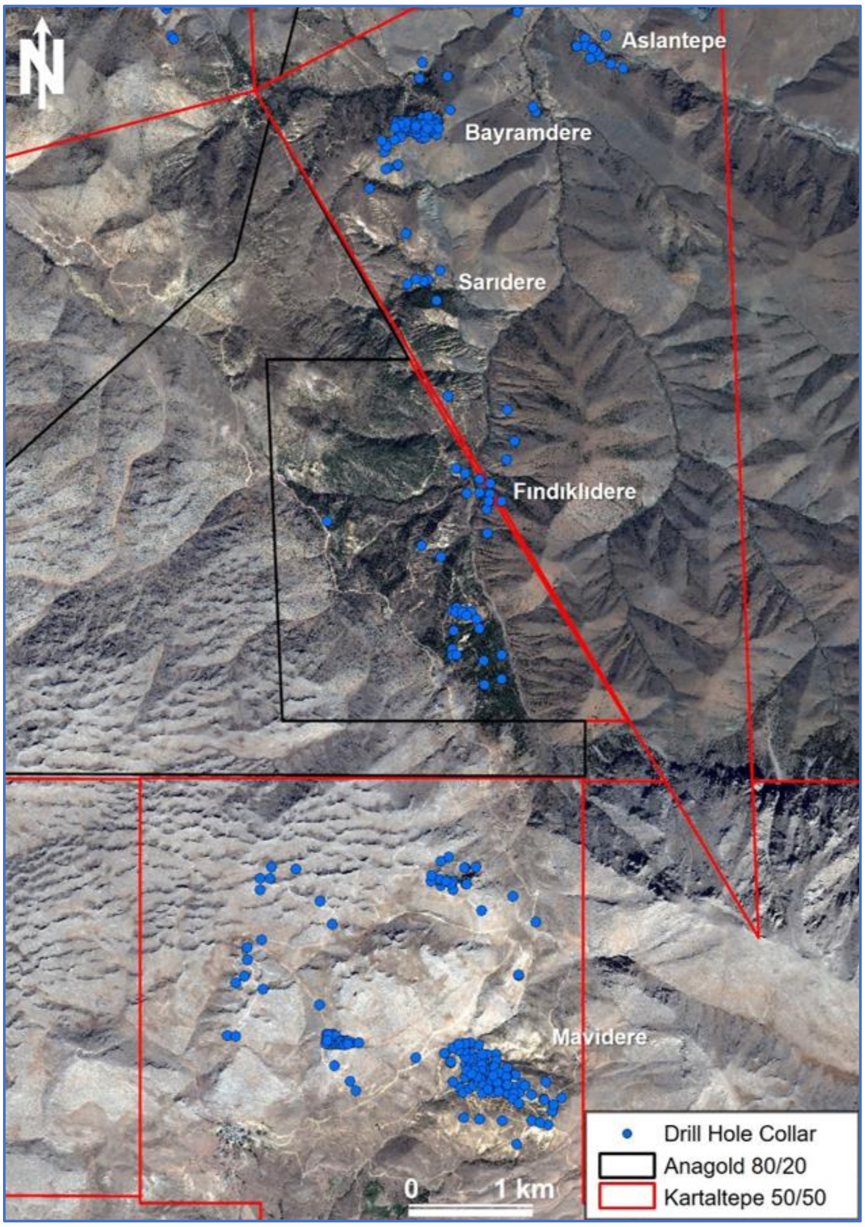

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Table 10.4 Drilling History – Mavialtin Porphyry Belt Prospects

Project	Year	Number of Drillholes	Drilled Metres
Aslantepe	2014	15	2,278.7
	2018	2	440.3
	2020	1	400.8
	Aslantepe Total	**18**	**3,119.8**
Bayramdere	2007	4	763.5
	2013	28	4,024.0
	2014	68	4,698.3
	2015	17	669.9
	2016	1	98.0
	2020	2	480.5
	Bayramdere Total	**120**	**10,734.2**
Fındıklıdere	2008	4	1,085.3
	2012	15	5,132.0
	2013	4	1,091.2
	2014	3	825.5
	2019	5	2,501.5
	2020	5	2,121.8
	Fındıklıdere Total	**36**	**12,757.3**
Sarıdere	2007	6	1,160.5
	2013	1	301.0
	2020	3	1,384.0
	Sarıdere Total	**3**	**2,845.5**
Mavidere	2001	8	1,780.3
	2008	22	7,761.1
	2011	22	3,806.2
	2012	37	10,479.5
	2013	78	11,171.6
	2018	5	2,119.8
	2019	4	1,567.1
	Mavidere Total	176	38,685.6

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

From 2004 to late-2012, samples were prepared at ALS İzmir, Turkey (ALS İzmir) and analysed at ALS Vancouver, Canada (ALS Vancouver), (collectively ALS Global). From late-2012 through 2014, samples were prepared and analysed at ALS İzmir. Samples in 2015 were prepared and analysed at the SGS laboratory in Ankara, Turkey (SGS). From 2015 to current, ALS İzmir is being used as the main laboratory and samples are being prepared and analysed there. Umpire analysis was completed by ACME Mineral Laboratories (ACME) in Ankara, Turkey.

SGS is certified to ISO 9001:2008 and OHSAS 18001, ALS İzmir has ISO 9001:2008 certification, and ALS Vancouver is ISO/IEC 17025:2005 accredited for precious and base metal assay methods. ACME is part of the Bureau Veritas (BV) group, globally certified to ISO9001:2008.

ALS Global and SGS are specialist analytical testing service companies, both independent of SSR.

Samples from the 2000–2003 drilling programme were submitted to OMAC Laboratories Limited (OMAC) in Loughrea, Ireland. ALS Global assumed ownership of OMAC in 2011.

Detailed sampling and quality assurance and quality control (QA/QC) procedures for reverse circulation (RC) and diamond core (DD) drilling were instigated and have been in use since the first drill programme. The QA/QC procedures have been retained by Anagold, although the insertion rates have been modified for some of the later programmes.

Anagold operates an on-site laboratory for assay of production samples. The on-site laboratory is certified to ISO 17025:2017 but is not independent. It is primarily used for the analysis of grade control samples.

11.1 Sample Collection

11.1.1 Reverse Circulation Drilling Sample Collection

Historically, RC drilling was completed with a 4.5–4.75 inch (11.4–12.0 cm) diameter down-the-hole hammer drill rig. RC cuttings were passed through a cyclone with a 10 inch (25.4 cm) port for sample collection. RC drill intervals were 1 m in length and cuttings for the entire 1 m sample interval were collected from the cyclone underflow in large, reinforced plastic bags. Prior to 2015, RC samples were split using a Jones splitter.

Since 2015, RC drilling has been completed with a nominal 5.25 inch face sampling hammer with centre sample return to a rig-side mounted sampling system. The sampling system included of a cyclone, sending 1 m samples to a rotary cone splitter. The rotary cone sample splitter was adjusted to maintain a representative sample volume. RC chip samples, to a weight of 3–5 kg, were collected in calico bags for analysis. All sample bags are clearly numbered and labelled with the drillhole name and sample number. Residual samples were collected in PVC bags and stored in a bag farm for six months in case re-logging, duplicate sampling, metallurgical sampling, or follow-up QA/QC was required.

The rig sampler sieves a small portion of the residual sample from the large plastic bag and places the sieved portion in a plastic chip tray to provide a sample for logging and as an enduring geological record. The plastic chip trays are photographed.

RC drilling is generally only used above the water table. The water table is closer to the surface in the northern region of the Main Zone, and for that reason, diamond drilling is the preferred method in this zone.

The following QA/QC samples are collected during the RC sampling process:

- Certified Reference Materials (CRMs) are inserted into each sample batch at a rate of two CRMs in every 40 samples (1-in-20 insertion rate).
- Prior to 2015, blank samples were inserted into each batch at a rate of one blank in every 60 samples (1-in-60 insertion rate). Since 2015, this has been changed to a 1-in-30 insertion rate.
- Field duplicate samples are collected by splitting an RC sample twice to collect two independently numbered samples of the same interval or selecting a quarter of the remnant core. Historically, field duplicates were collected and inserted into the sample job at a rate of 1-in-40 samples. In 2015, field duplicate insertion rates were increased to 1-in-20.

11.1.2 Diamond Drilling Sample Collection

Up until 2017, the diamond drilling undertaken on the project has generally been HQ or NQ diameter. HQ core has a nominal diameter of 63.5 mm while NQ has a nominal size of 47.6 mm. Approximately 90% of the DD core drilled at Çöpler and Çakmaktepe is HQ. Some drillholes are started with HQ and then reduced in size to NQ further down the hole.

Of the more recent drilling at Ardich, approximately 60% was completed with HQ core, and the remainder was mostly PQ sized core (very few holes were NQ core). PQ core has a nominal diameter of 85 mm.

Drill core is boxed at the rig by the driller and transported to the sample preparation facility on-site for logging by Anagold exploration staff.

Logging includes the collection of lithological, alteration, and structural information. Since 2017, drill core has also undergone a detailed geotechnical logging process including a detailed 'mining rock mass rating' to 'rock mass rating' system. In addition, core samples are collected every 10 m to undertake point load IS50 testing for uniaxial compressive strength (UCS).

Diamond core that is competent is sawn in half longitudinally with a diamond saw at the core yard. Core that is broken or rubbly is sampled using a spatula to take approximately half the sample. Half the core is placed in a sample bag and the remaining half is returned to the correct position in the core tray. Sample numbers are assigned, and sample tags are placed in the sample bags and recorded in the master sample list. Sample intervals are typically 1 m down-hole.

Prior to 2015, QA/QC samples were collected routinely during the sampling process. CRMs were inserted into each sample job at a rate of 1-in-20. Blank samples were inserted into each sample job at a rate of 1-in-60. Field duplicate samples were collected by cutting the remaining half core portion into two and selecting one quarter of the remaining sample to be submitted as the field duplicate. Field duplicates are collected and inserted into the sample job at a rate of 1-in-40 samples. From 2015 onwards, the field duplicate insertion rate was increased to 1-in-20.

11.1.3 Drillhole Logging and Data Collection

RC chip samples are collected by field staff for review by the logging geologist. Similarly, core samples are metre marked by field staff in preparation for the logging geologist.

Drill core is subjected to detail logging using Anagold geological codes and logging formats. Information captured includes lithology, structure, alteration, mineralisation, and geotechnical data on veining, joint frequency, and joint sets.

Until September 2019, all geological data was recorded onto hard-copy logs and then transcribed into text files, using data-loading templates, ready for loading into the corporate relational SQL database. Since September 2019, hard copy logs have been replaced with data loading templates on touchpads with direct links to the company server. Files located on the server are uploaded into the corporate database regularly following appropriate checking of the data entry.

Until 2017, the SQL drilling database was managed by the Anagold exploration team located at the Çöpler mine site. Thereafter, the exploration database is controlled and managed by the Anagold exploration team located at the head office in Ankara.

11.2 Sample Preparation

11.2.1 Reverse Circulation Sample Preparation

The majority of historical RC sample preparation was completed at ALS İzmir. From late-2012 through to the end-of-2013, pulp samples weighing approximately 150 g were sent to ALS Vancouver. All samples in 2014 were generated and analysed by ALS İzmir. In 2015, samples were sent to SGS for preparation and assay. Since 2015, ALS Global is being used as the main laboratory.

11.2.2 Diamond Drilling Sample Preparation

The majority of historical DD sample preparation was completed at ALS İzmir. From late-2012 through to the end-of-2013, pulp samples weighing approximately 150 g were sent to ALS Vancouver. All samples in 2014 were generated and analysed by ALS İzmir. In 2015, samples were sent to SGS for preparation and assay. Since 2015, ALS Global is being used as the main laboratory.

11.3 Sample Analysis

In the period 2004–2014, samples analysed for Au at ALS Vancouver used method Au-AA25, which is a fire assay of a 30 g sample followed by atomic absorption spectroscopy (AAS). The lower and upper detection limits are 0.01 g/t Au and 100 g/t Au respectively. Samples that returned Au grades above the upper detection limit were re-analysed using the gravimetric method Au-GRA21.

Analysis of an additional 33 elements was performed using the ALS Global method ME-ICP61, which involves a four-acid (perchloric, nitric, hydrofluoric, and hydrochloric acid) digestion (four-acid digest), followed by inductively coupled plasma-atomic emission spectroscopy (ICP-AES). Ag, Cu, Pb, Zn, and Mn are among the 33 elements analysed by this method.

In 2015, samples sent to SGS were analysed using the Au fire assay method FAA303, which also uses a 30 g sample and ICP-AES. Detection limits are 0.01 g/t Au. When content was detected above 3 g/t Au, method FAG303 using a gravimetric finish was added.

A 36 element analysis was performed at SGS with ICP40B method, which involves a four-acid digest (4A) followed by analysis via inductively coupled plasma-optical emission spectroscopy (ICP-OES).

From 2016 to recent, samples have been sent to ALS İzmir. Until 2019, Au-AA23 method was used, involving a fire assay of a 30 g sample followed by AAS with the lower and upper detection limits being 0.01 g/t Au and 10 g/t Au respectively. Samples that returned grades above the upper Au detection limit were re-analysed using the gravimetric method Au-GRA21. Since 2019, Au-AA24 method with a 50 g sample and lower detection limit of 0.005 g/t Au has been used. For Au grades above the upper detection limit, gravimetric method Au-GRA22 with a 50 g sample is used.

11.4 Sample Security

Drill core and RC chips are transported to the core storage facility by either the drilling company personnel or Anagold geological staff. Once at the facility, the samples are kept in a secure location while logging and sampling is conducted. The DD core storage facility is enclosed by a fence and gate that is locked at night and when the geology staff are absent. When samples are transported off-site a commercial carrier is used.

11.5 QA/QC Procedures

The QA/QC programme has historically consisted of a combination of QA/QC sample types that are designed to monitor different aspects of the sample preparation and assaying process.

Blanks consist of non-mineralised samples that are submitted in order to identify the presence of contamination through the sample preparation process. Prior to 2015, blank samples comprised of commercially available pulp samples. As pulp blanks require neither crushing nor pulverising, they are of limited value in terms of identifying contamination through those aspects of the sample preparation process. Therefore, commencing in 2015, the pulp samples were switched to a coarse quartz material that would allow for better monitoring of sample contamination. Blank samples have been inserted routinely into all sample batches. If a blank sample returns an assay grade above an acceptable limit, contamination from a previous mineralised sample is assumed to have occurred at either the crushing or pulverisation stage. The first sample in a drillhole is typically a blank, after which blanks are inserted into the sample batch at a nominal rate of 1-in-60 samples. The insertion rate was updated and for the period 2015–2020 to approximately 1-in-30 for diamond drillholes.

CRM samples are inserted into sample submissions in order to monitor and measure the accuracy of the assay laboratory results over time. CRMs have been inserted into sample submissions at a nominal rate of 1-in-30. The frequency was increased from 3% to 5% in 2015. Several different CRMs have been selected for use at varying Au and Cu grades over the life of the project. Pulp blanks have been used to determine the accuracy of assay results at very low-grades, and as such are inserted using the same logic as CRMs. The combined insertion rate of pulp blanks and CRMs is a nominal 1-in-20 samples. For the period 2015–2020, the combined rate is approximately 1-in-25.

Field duplicates are used as a means of monitoring and assessing sample homogeneity and inherent grade variability and enable the determination of bias and precision between sample pairs. Field duplicates have been routinely inserted into both RC and DD sample submissions since drilling began. DD field duplicates are generated by cutting the residual half core sample into halves again and submitting one of the resultant quarters of core as the field duplicate. RC field duplicates are generated by splitting the RC sample twice to create two samples from the same interval. Field duplicates have historically been submitted at a nominal rate of 1-in-40 samples. In 2015, the field duplicate insertion rate was increased to 1-in-20. Since 2017 for DD samples, duplicate samples are being collected as laboratory duplicates instead of quarter core field duplicate samples.

12 DATA VERIFICATION

Independent detailed quality assurance and quality control (QA/QC) analysis is undertaken routinely on data from the Çöpler project.

This work was discussed in detail in the CDMP20TR. The reader is referred to that report for all QA/QC of data used to develop resource models prior to November 2020.

The QA/QC pertaining to data used in the updated Ardich resource model is described here.

12.1 Çöpler Deposit Data Verification

The independent quality assurance and quality control (QA/QC) review presented in the CDMP20TR confirms that the Çöpler drillhole data sampling and assaying is of a good standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results. This is especially true for gold, which is the primary metal of economic interest. The confidence in the silver, copper, sulfur, and carbon analyses is at a level that at minimum supports modelling for geometallurgical and by-product metal characterisation.

In 2014, an independent database audit and review of available QA/QC data was undertaken to ensure the data are of sufficient quality to support resource estimation (the 2014 audit). The database audit covered data collected from 2000 to December 2013.

A further independent audit of the Çöpler deposit database as of 15 July 2015 was completed that year to verify the data are of sufficient quality to support Mineral Resource estimation of gold, copper, and silver for the Çöpler deposit (the 2015 audit). The 2015 audit focused on the 121 drillholes (12,959.8 m) completed since the 2014 audit. Available QA/QC data were evaluated to ensure the assay data are suitable to support resource estimation.

A data audit covering new data obtained from 2015 through 2020 was completed in June 2020 (the 2020 audit):

- Yetkin, E., 2020 (2020a). Çöpler Project Drill Data Validation, Verification & QA/QC Review. 30 June 2020.

The 2020 audit discusses some minor inconsistencies and outliers but overall confirms the previous findings that the Çöpler drillhole data sampling and assaying is of a good standard and suitable for the purpose of Mineral Resource estimation and the reporting of exploration results.

12.2 Çakmaktepe Deposit Data Verification

The independent QA/QC review presented in the CDMP20TR confirms that the Çakmaktepe drillhole data sampling and assaying is of a good standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results. This is especially true for gold, which is the primary metal of economic interest. The confidence in the silver, copper, sulfur, and carbon analyses is at a level that at minimum supports modelling for geometallurgical and by-product metal characterisation.

12.3 Ardich Deposit Data Verification

12.3.1 Data Verification – Ardich

Independent data verification was conducted on the Ardich drilling databases and available QA/QC sample data for drilling completed from the first Ardich hole drilled on 1 August 2017 to the established data cut-off date for the Mineral Resource modelling of 29 May 2021.

This verification was completed in stages as drill programmes progressed, and is reported in nine reports:

- Mineral Consultancy, 2018. Ardich Project Drill Data QA/QC Review. 28 February 2018.
- Yetkin, E., 2018 (2018a). Ardich Project Drill Data QA/QC Review. 29 July 2018.
- Yetkin, E., 2018 (2018b). Ardich Project Drill Data QA/QC Review. 29 October 2018.
- Yetkin, E., 2019 (2019a). Ardich Project Drill Data Validation, Verification & QA/QC Review. 8 March 2019.
- Yetkin, E., 2019 (2019b). Ardich Project Drill Data Validation, Verification & QA/QC Review. 31 October 2019.
- Yetkin, E., 2020 (2020c). Ardich Project Drill Data Validation, Verification & QA/QC Review. 30 March 2020.
- Yetkin, E., 2020 (2020d). Ardich Project Drill Data Validation, Verification & QA/QC Review. 30 November 2020.
- Yetkin, E., 2020 (2020e). Ardich Project Drill Data Validation, Verification & QA/QC Review. 30 January 2021.
- Yetkin, E., 2020 (2020f). Ardich Project Drill Data Validation, Verification & QA/QC Review. 30 July 2021.

It is concluded that the Ardich drillhole data sampling and assaying is of a high-standard and suitable for the purpose of Mineral Resource estimation and the reporting of exploration results.

12.3.2 Collar Location – Ardich

Collar positions were verified against the pre-mine topographic surface DTM to check for inconsistencies in elevation. The threshold difference between the DTM and the drillhole collar elevation used for validation was a ±4 m difference in data up to 2020, at which time the tolerance was decreased to ±3 m.

One hole was found to have a difference outside the tolerance limits – AR214, with 7.12 m difference. All other differences were < 3 m.

As Ardich has not been mined to date, this discrepancy can be resolved by re-surveying the collar location.

12.3.3 Down-hole Surveys – Ardich

All the Ardich drillholes were downhole surveyed using a multi-shot (Devico or Reflex) with readings spaced at 10 m on average (range of 4–110 m).

Seven Ardich holes were found to have no down-hole survey data and three successive survey intervals were found to have large gaps between readings (95–170 m).

A comparison of successive down-hole survey readings for a given drillhole was undertaken using a maximum 5° variation over 30 m (0.17°/m) in either inclination or azimuth to flag records with excessive deviations. A total of three spurious readings were deemed to be out of acceptable limits, and those data were removed from the resource database.

A recommended magnetic declination correction discussed in the CDMP20TR has been implemented for Ardich data.

12.3.4 Geology, Density, and Geotechnical Logs – Ardich

The drillhole database lithology table was checked for alphanumeric categorical code validity and interval reporting consistency with the log key sheets. No mis-matches were identified, and all entries were found to be identical to the codes provided in log key sheets. One lithology interval was shown to have an overlapping FROM–TO and there were three intervals that were missing lithology records.

Some minor discrepancies were identified in other coding in the database, such as lower-case ‘fault’ codes used instead of upper-case, or a lithology of ‘CLASTICS’ rather than ‘CLASTIC’, in both cases causing two different unique categories to be created, and some new codes created in the ‘redox’ and ‘alteration’ tables that do not appear in the log key sheet.

Several logged intervals were re-logged following the identification of incompatible geochemistry, and all of these were updated in the lithology logs used in the 2021 resource modelling dataset.

Density data were reviewed during six of the nine Ardich verification campaigns. Density measurements are collected using the same process described in Section 12.1.4 for the Çöpler deposit. A systematic truncation from four decimal places to three decimal places was observed, and several transcription errors in FROM–TO records were identified. Manually calculated spot check values were within ~2% of the density reading supplied in the resource database. The density samples are representative in a spatial and geological context. On a total project basis, there are no obvious density outliers.

12.3.5 Assays – Ardich

There were two different independent laboratories used for assays and geochemical analyses for the entire Ardich database to date, these were:

- ALS Global
- BV (ACME)

The variety of laboratories resulted in a variety of method codes for fire assay, four-acid digestion, multi-element, and Leco analyses.

In consistency checks on the ‘tblVWDHAssays_ALL’ assay table, four samples were found to have missing assay entries. The highest 1% of assays were checked for transcription errors, with no major errors identified.

12.3.6 Witness Samples – Ardich

No witness samples are known of for Ardich.

12.3.7 Quality Assurance / Quality Control (QA/QC) Results – Ardich

Ardich QA/QC data was independently reviewed on a campaign basis at milestone times in the evolution of the exploration programme. There are currently nine individual reports describing the results. The collective results are reported in this section.

The Ardich QA/QC programme follows suggested guidelines for QC sample insertion rates:

- 3%–5% CRMs and blanks
- 5%–10% field duplicates
- 3%–5% pulp duplicates
- 5% of coarse rejects/pulps to a third-party external laboratory

12.3.7.1 Screen Analyses – Ardich

No screen analysis has been undertaken to date on Ardich material.

12.3.7.2 Certified Reference Material (CRM) Samples – Ardich

The principal assay laboratory for drill samples was ALS İzmir, with umpire samples principally submitted to the BV (ACME) laboratory.

Au CRMs were submitted across the entire Ardich database, plus S (Leco) and C (Leco) CRMs in the later programmes. The average insertion rate was of the order of 3.5%, which meets the guideline.

The performance of the CRM sample data was assessed by plotting the laboratory assay values for Au (FA and CL), Ag (4A), Cu (4A), S (4A and Leco), and C (Leco) of the CRMs against time on control charts.

A review of the CRM results from the samples submitted indicated that both ALS İzmir and BV (ACME) had acceptable overall performances for the listed Au CRMs used during the programme, although ALS consistently had issues with isolated ±2SDs as well as failed cases of ±3SDs. In general, the ALS shows high bias in almost all Au CRMs at varying levels, being more evident in low Au and cut-off Au grades, which are also responsible for the most of the +2SD and +3SD occurrences. Few of these failed cases appeared to be as a result of mis-labelling. No unexplained extreme outliers were identified. Several CRMs had insufficient data to identify any change in performance over time.

The performance of Ag, Cu, S, and C CRMs was also reviewed, showing ALS had an acceptable overall performance with isolated cases to be followed up for Ag, Cu and S.

ALS and BV (ACME) performance both for S (4A) and S (Leco) are generally acceptable other than calibration-related bias noted for low-grade CRMs. C (Leco) performance of OREAS20A (ALS and BV (ACME)) and OREAS25A (ALS and BV (ACME)) returned acceptable results both for low-grade and cut-off grade.

Timely monitoring of the CRM performance will ensure that the replicate assays stay within range, that systematic analytical drift is promptly corrected, and that mis-labelled samples are promptly identified. The extreme-outlier cases need to be investigated and if these are found to be mis-labelling then the organisational procedures should be reviewed and updated. If it transpires that these are not mis-labelled samples and the errors are found to be laboratory-related, then re-assay procedures needed to confirm the assays for the relevant batches.

12.3.7.3 Blank Samples – Ardich

Blanks were inserted into the sample stream as a check for cross-contamination during sample preparation. The insertion rate was of the order of 3%, which meets the guideline.

For ALS İzmir, Au assays for blanks were assessed by charting the laboratory assay values and assessing performance versus the maximum accepted threshold value of 0.05 g/t Au, which is 10 times the lower DL. All blank assays were below 3DL except for one sample, however it was noted that there were several occurrences where consecutive blanks assayed close to the threshold.

For Ag, all blank assays were below the maximum threshold value of 0.5 g/t Ag. For Cu, the threshold level is 10 ppm Cu and there were several samples that assayed slightly above, at, or close to the threshold value. For sulfur (both for 4A and Leco) the threshold value is 0.1% S and all blank sample assays were below this value. For carbon (Leco) the threshold value is 0.1% C and there are 92 assays above the threshold. These results show that the blank material used to monitor Au and other elements may not be suitable for C analysis, or that the samples are contaminated during the sample preparation. Other than these no obvious contamination issues are apparent within the assay database.

For BV (ACME) blanks returned all below threshold values for Au, Ag, Cu, S (4A), S (Leco), and C (Leco) analyses. Only nine blanks were submitted to BV (ACME), which does not meet the 3%–5% insertion rate guideline.

12.3.7.4 Duplicate Samples – Ardich

Duplicate sample data was analysed to determine the reproducibility of assays according to the combination of geological, sampling, and analytic variances. The insertion rate was of the order of 5%, which meets the guideline.

The duplicates in each of the six QA/QC reviews have an average absolute relative difference of between 0.029–0.140 for Au, with a sample-weighted average of approximately 0.053, which falls within or below the rule-of-thumb of 0.10–0.20 absolute relative difference range for acceptable laboratory duplicate samples for each campaign.

The Au assay variance in each campaign, given by the average percent difference, is within the range of –2.6%–3.8%, with an average of approximately –0.25%, which falls within the rule-of-thumb of ±10% precision window.

The absolute relative difference and average percent difference results were equally encouraging for S (4A) where data was obtained (from drillhole AR56 onwards).

The high-precision of the duplicates reflects the inherent sample homogeneity of laboratory-prepared duplicate samples from coarse rejects, which allows more representative sampling of the grade population.

12.3.7.5 Check Assays – Ardich

All six QA/QC campaigns report the results of umpire assays with pulp duplicates submitted to BV (ACME) for independent analysis.

The rate of check assay was lower for the earlier campaigns, as low as 2% in the first campaign, but the overall average is 4.7%, which is approaching the guideline.

Generally, the results show low-level artefacts due to differing DLs between the two laboratories, and the occasional outlier result, but overall, the scatter plots demonstrate strong linear correlation.

The check assays in each of the six QA/QC reviews have an average absolute relative difference of between 0.042–0.078 for Au, with a sample-weighted average of approximately 0.064, which falls below the rule-of-thumb of 0.10–0.20 absolute relative difference range for acceptable laboratory duplicate samples in every campaign.

Two of the earlier campaigns showed questionable performance for Ag and S; a result that is considered to be moderated by the small number of samples submitted in these early campaigns.

12.3.8 Discussion – Ardich

The independent QA/QC reviews confirm that the Ardich drillhole data sampling and assaying is of a high-standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results. This is especially true for gold, which is the primary metal of economic interest. The confidence in the silver, copper, sulfur, and carbon analyses is at a level that at minimum supports modelling for geometallurgical and by-product metal characterisation.

12.4 Bayramdere Deposit Data Verification

The Bayramdere sampling project was part of the near-mine programme that also included the Yakuplu East and Yakuplu South-east areas.

Independent data verification was conducted during and immediately following the 2015 drilling programme on the project, and a data audit for Bayramdere drilling was completed in January 2016 (Cube Consulting, 2016b).

The independent data verification concluded that the sample data is considered to be of an acceptable standard and appropriate for the purpose of Mineral Resource estimation and the reporting of exploration results.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Oxide Ore for Heap Leaching

13.1.1 Testwork – Çöpler Oxide

Metallurgical testwork for Çöpler oxide ore for heap leaching commenced in September 2004. Much of this testing was carried out by Resource Development Inc. (RDi) of Wheat Ridge Colorado, with oversight from Ausenco Limited of Brisbane, Australia, and Pennstrom Consulting of Highlands Ranch, Colorado. Additional follow-up metallurgical testwork was conducted by AMMTEC Limited (AMMTEC) of Perth, Australia in 2009 and by McClelland Laboratories, and supervised by Metallurgium more recently.

The heap leaching facilities were commissioned at the Çöpler project in late-2010 and have operated continuously since that time. Operations are currently ongoing.

13.1.2 Testwork – Çakmaktepe Oxide

Metallurgical testwork on Çakmaktepe oxide ore for heap leaching was undertaken at the on-site metallurgical laboratory, initially under the supervision of Kappes, Cassiday & Associates. The initial testwork in 2015 undertook bottle roll and column leach tests. The results compare to the Çöpler oxide ore, with similar behaviour and leach kinetics. Subsequently, Çakmaktepe oxide ore was heap leached together with Çöpler oxide ore.

13.1.3 Testwork – Ardich Oxide

Metallurgical testwork on Ardich oxide for heap leaching has been undertaken at McClelland laboratories and supervised by Metallurgium. An initial testwork programme including bottle roll and column leach was carried out in 2019. This initial programme identified two distinct domains with respect to gold recovery based on sulfide sulfur (SS) content of <1% and 1%–2%. The column test results indicated that the listwanite, dolomite, and jasperoid lithologies have physical properties amenable to heap leaching. The column tests were undertaken at a crush size of P80 of 12.5 mm.

This initial test programme has been followed up in 2020 and 2021 with further testwork, with final results yet to be released.

13.1.3.1 Ardich Crushing Testwork

Crushing testwork on six Ardich composite samples was performed as part of the 2019 McClelland testwork programme, Crushing Work index (CWi), and Abrasion index (Ai). The CWi values ranged from 4.0–6.9 kWh/t, indicating that the material was very soft. The jasperoid was the hardest material, with a CWi of 6.9 kWh/t. The Ai values ranged from 0.12–0.90. The jasperoid was the most abrasive (0.90, Very Abrasive), whereas all other lithology types ranged from 0.12–0.26 (Abrasive to Moderately Abrasive).

13.1.4 Testwork – Bayramdere Oxide

Metallurgical testwork has been completed to characterise the Bayramdere oxide mineralisation and determine its suitability for potential heap leaching. In total, five PQ (85 mm diameter) DD (diamond core drilling) holes were completed in 2014 for this purpose and 91 m of half-core have been provided for intermittent bottle roll leach (IBRL) test and column leach testing.

In the IBRL tests, the gold extraction ranges from 54% to 97% at the end-of-11 tests with the consumption of 0.85 kg/t NaCN.

In the column test, gold extraction is 84% in the two duplicate columns.

Final gold extraction in column testing is approximately 84% with reasonable leach kinetics. The extracted gold quantity will be economic for heap leach processing if haul costs are not excessive.

13.1.5 Heap Leach Gold Recovery

The heap leaching process gold recovery assumptions have been updated to reflect actual performance of the operation. The gold recovery assumptions are summarised for Çöpler oxide in Table 13.1, Çakmaktepe oxide in Table 13.2 (including Bayramdere), and Ardich oxide in Table 13.3.

Table 13.1 Çöpler Gold Recovery Assumptions for Heap Leaching of Oxide

Oxide Ore Type	Çöpler Zone					
	Manganese	**Marble**	**Main**	**Main East**	**Main West**	**West**
Diorite	71.2	62.3	71.2	71.2	62.3	62.3
Metasediment	66.8	66.8	66.8	66.8	66.8	66.8
Limestone/Marble	78.4	75.7	68.6	78.4	75.7	75.7
Gossan	71.2	65.1	71.2	71.2	65.1	65.1
Manganese Diorite	71.2	62.3	71.2	71.2	62.3	62.3

The original gold recovery assumptions for Çöpler ores were developed in 2008, based on the results of column leach and bottle roll testing performed by RDi between 2005–2008. These recovery assumptions are reviewed and updated annually based on the following information:

- An analysis of the results of additional column leach and bottle roll tests performed on monthly composite samples of heap leach feed material conducted at the Çöpler project from July 2011 through December 2019.
- Use of a MS Excel-based heap leach production model that is calibrated against actual gold production data at the Çöpler mine from start-up.

The recovery values listed in Table 13.1, Table 13.2, and Table 13.3 consider heap leaching of ore crushed to 80% passing 12.5 mm, agglomerated, and placed on a lined heap leach pad for treatment.

Table 13.2 Çakmaktepe Gold Recovery Assumptions for Heap Leaching of Oxide (incl. Bayramdere)

Oxide Ore Type	Çakmaktepe Zone				
	Central	North	East	South-east	Bayramdere
Limestone/Marble	70.0	59.0	67.0	–	75.0
Metasediment	80.0	14.0	–	–	–
Gossan	–	59.0	67.0	75.0	75.0
Jasperoid	73.0	59.0	–	–	–
Diorite	61.0	38.0	–	–	–
Ophiolite	70.0	63.0	67.0	75.0	75.0

Table 13.3 Ardich Gold Recovery Assumptions for Heap Leaching of Oxide

Ore Type	Ardich Zone	
	Main	East
Sulfur <1%		
Jasperoid	50.0	50.0
Listwanite	73.0	55.0
Dolomite	73.0	55.0
Sulfur 1%–2%		
Jasperoid	40.0	40.0
Listwanite	58.0	45.0
Dolomite	58.0	45.0

13.2 Sulfide Ores

Sulfide material (i.e., material with >2% sulfur content) is not suitable for treatment by the heap leaching process.

13.2.1 Historical Testwork – Çöpler Sulfide

Historical testing was conducted on samples from the sulfide material in several phases. RDi performed several sulfide processing scoping-level investigations from 2006–2009. A two-phase programme on sulfide samples was conducted at SGS laboratory in Ankara, Turkey (SGS) in 2009 and 2010 to support a Pre-feasibility Study (PFS) completed in 2011, (Samuel, 2011). A QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy) mineralogy study on three sulfide (and six oxide) samples was performed by AMMTEC in December 2008.

The historical work completed at both RDi and SGS concentrated on evaluating sulfide processing options, including direct cyanidation, flotation, cyanidation of flotation concentrates, pressure oxidation (POX) coupled with cyanidation, and roasting coupled with cyanidation. The evaluation of the historical data in the PFS resulted in the selection of POX coupled with cyanidation as the process to further evaluate with testing and a FS.

Initial metallurgical testwork carried out by RDi indicated that 11%–30% of the gold content in the Çöpler sulfide material may be amenable to whole-ore cyanidation, as demonstrated by diagnostic leaching. Between 60%–80% of the gold content was found to be associated with sulfide minerals and would require some type of oxidation step to liberate the gold for cyanidation.

The RDi scoping studies indicated that pre-treatment using POX was the most effective treatment and displayed the potential to achieve greater than 90% gold extractions. Flotation tests indicated that gold could be recovered by flotation, but the concentrates were low-grade with relatively high-mass pulls, and relatively low-gold recovery. Testwork indicated that flotation concentrate, and tailings did not leach well using cyanide, even after being finely ground.

13.2.2 Sulfide Mineralogy

In December 2008, Anagold commissioned AMMTEC to complete a QEMSCAN precious metals search (PMS), trace mineral search (TMS), and energy dispersive spectra signal (EDS) mineralogy analyses performed on three sulfide mineralisation samples. Analyses were performed on samples of diorite, metasediment, and massive pyrite rock types.

The findings from the 2008 QEMSCAN analyses indicated that the gangue mineralisation in the sulfide mineralisation is composed mainly of quartz, micas / clays, and feldspars, (displaying relative abundances of approximately 31%, 27%, and 21%, respectively). The sulfide mineralisation consists of pyrite, arsenopyrite, chalcopyrite, and sphalerite.

A gold deportment study was performed by AMTEL Ltd. (AMTEL) on samples of MC4 composite after flotation separation. Although flotation was not part of the flow sheet, it is a useful method of concentrating the sulfides (the main gold carriers) to improve analysis statistics.

The combined concentrate represented 18.5% of the feed mass and assayed 9.8 g/t Au and 23% SS. Recoveries of gold and sulfur to concentrate were 72.7% and 90% respectively. Flotation tailings assayed 0.68 g/t Au and 0.48% SS.

The detailed mineralogical analysis confirms that the gold is primarily carried by sulfide minerals. In the calculated head, 83% of all gold is in sulfides (free or locked) and only 2.4% was held in rock. The remainder of the gold (14%) was present as free gold, and this correlates well with a direct cyanidation recovery of only 17% when the ore was ground to a P_{80} of 90 µm.

Of the gold that is in sulfides, the majority (78%) is in sub-microscopic form. This confirms the refractory nature of the ore and explains why oxidation of the sulfides is necessary to make the gold available for leaching. Arsenopyrite was the sulfide mineral found to have the highest contained gold, averaging 123 g/t Au by one measure and 182 g/t Au by a second. Gold in pyrite was more than an order of magnitude lower than arsenopyrite and averaged 7.0 g/t Au. Marcasite, a mineral chemically similar to pyrite, carried an average of 17.8 g/t Au. Of the gold contained in sulfides, 50% was found to be in arsenopyrite, 25% in pyrite, and 20% in marcasite.

In summary, the AMTEL gold department study is consistent with previous mineralogy studies and confirms that a large portion of the gold is present as sub-microscopic particles, primarily in sulfides, largely arsenopyrite. The study also concluded that whole-ore oxidation would be required as a pre-treatment to cyanidation to liberate the majority of the gold contained in the sulfide materials.

13.2.3 Direct Cyanidation

Hazen performed direct cyanidation carbon-in-leach (CIL) tests at various grind sizes with no pre-treatment on the individual sulfide rock type composites to establish baseline gold extractions. The goal of these tests was to examine gold extraction variability with grind size. These samples were subsequently used to prepare feed composites used in the Hazen pilot plant programme.

The testwork demonstrated that the bulk of the Çöpler sulfide samples are refractory to direct cyanidation, and that extractions do not improve significantly with finer grinding.

Table 13.4 Gold Deportment in Flotation Separated Streams

Form and Carrier of Gold	Concentrate (g/t)	Tails (g/t)
Assayed Grade	10.187 ± 0.167	0.837 ± 0.028
Free / Liberated Gold Grains		
>40 µm	0.106	0.004 *
5–4 µm	0.346	0.003
<5 µm	0.871	0.146
Exposed Associated Gold Grains		
Free Sulfides +5 µm	0.350	0.018
–5 µm	–	–
Rock-Sulfide Composites	0.125	0.052
Rock Particles	0.021	0.035
Enclosed Associated Gold Grains		
Free Sulfides +5 µm	0.977	0.007
–5 µm	0.292	0.029
Rock-Sulfide Composites	0.338	0.023
Rock Particles	0.014	0.031
Sub-microscopic Gold		
Free Sulfides +5 µm	4.156	0.020
–5 µm	1.244	0.157
Associated Sulfides	1.605	0.304
Total (mineralogically counted)	10.444(102.5%)	0.829(99.0%)

* From a very small number of grains (1 free grain, from ~2 kg of material)

13.2.4 Flotation Testwork

Flotation testwork has been undertaken on Çöpler sulfide samples since before 2006 with a series of testwork programmes and studies undertaken by RDi, FL Smidth, and the on-site metallurgical laboratory.

Initially the testwork was focused on development of a viable flowsheet to recover gold to enable subsequent recovery as doré. This work was unsuccessful due to a generally poor flotation response, resulting in the adoption of the current POX and CIP gold recovery flowsheet.

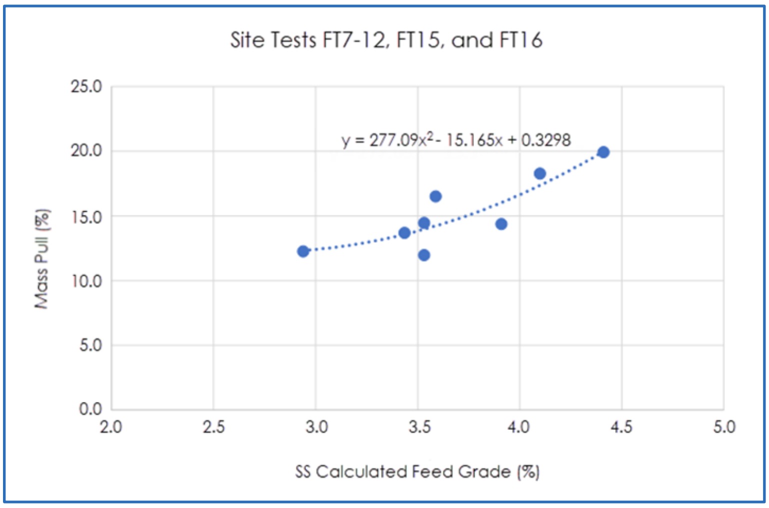
In 2019, flotation was again considered for incorporation into the POX / CIP circuit to improve both sulfur and gold recovery and enable the POX circuit to operate at optimum conditions.

Testwork was conducted on fresh material from the existing sulfide circuit. A total of 20 tests were conducted as part of this programme.

The key variables considered in determining throughput for flotation are SS flotation recovery and flotation mass pull. Gold recovery to concentrate and gold recovery of the flotation tails are also determined. Of the 20 tests undertaken, a total of eight flotation testwork tests are considered representative due to their relative commonality of flotation conditions, and the SS feed grade is within the range that the flotation plant is expected to operate. The results ranged from 65% to 81% SS recovery, and 43% to 55% Au recovery to concentrate.

The mass pull for sulfide flotation is typically related to SS grade. Figure 13.1, shows the relationship of mass pull to SS feed grade.

Figure 13.1 Feed SS% – Mass Pull Relationship



Anagold, 2020

Float Concentrate Mass Pull = 277.09 x Feed SS%^{2} – 15.165 x Feed SS% + 0.3298

13.2.5 Testwork – Comminution

The comminution properties for the three major ore domains (metasediment, diorite, and manganese diorite) have been measured during all testwork stages. Rock competence drives semi-autogenous grind (SAG) mill selection, Bond Work index (BWi) drives ball mill selection, and Ai is used to estimate media and mill liner consumption rates. The major domains exhibit moderate comminution characteristics.

As part of the flotation circuit sizing, the throughput capacity of the installed crushing and grinding circuit was determined on review of testwork and plant actual performance.

The review of the grinding circuit determined that the throughput has exceeded design expectations since commissioning due to the processing of ore that is softer than the design comminution testwork identified. The design maximum feed rate of 306 t/h was achievable with close to full milling power being consumed. However, an average throughput rate of 370 t/h was achieved in the period late-2019 through early-2020 with the SAG and ball mills drawing approximately half of their design power.

A simulation model of the comminution circuit was prepared (in JKSimMet) and calibrated to this actual plant performance. This calibrated simulation was then used to estimate plant performance with future harder ores, having properties approximating design expectations. Wood's simulation showed that the plant is expected to be able to process the target rate of 400 t/h of design-hardness ore with the mills at full design operating power.

13.2.6 Testwork – POX

Three continuous pilot plant programmes have been conducted for the POX sulfide plant; the first two programmes at Hazen Research, Inc. (Hazen) comprising a total of four test campaigns, and the third programme at SGS Lakefield Oretest, Perth (SGS Perth). Three campaigns were completed during the first pilot plant programme, with the first campaign commencing in February 2012. The second pilot programme incorporating one campaign, was conducted in December 2012. The third pilot programme, conducted in August 2015, included a single campaign that tested multiple lithologies at high and low-acidulation extents.

The pilot plant facility for the first pilot programme included the following continuous circuits: acidulation, POX autoclave, hot cure (HC), primary neutralisation (PN), six-stage counter current decantation (CCD), and mixed sulfide precipitation (MSP). Ore preparation (grinding), cyanidation, activated carbon gold recovery, cyanide destruction, tailings neutralisation, and final tailings production were all completed on a batch basis.

In 2015, Anagold performed confirmatory pilot testing on a range of ore-types and composite blends treated at 'high' and 'low' acidulation conditions. This programme comprised a single pilot plant campaign, Campaign 5, which was conducted at SGS Perth during August and September. Apart from testing the impact of acidulation chemistry, one of the key purposes of the campaign was to produce samples for repeat thickener vendor testing. This was prompted by the inconsistent vendor data generated during Campaigns 1–4.

13.2.7 Testwork: Pyrite Recovery from Copper-Rich Ores

Preliminary metallurgical testwork has been undertaken to investigate the potential to produce a copper–gold concentrate for sale and a pyrite concentrate to supplement POX operations utilising a copper-rich portion of the Çöpler resource.

The testwork was conducted at ALS using drillhole samples to produce a master composite and eight individual composites representing copper-bearing zones of the Çöpler mine. The composites copper grades were between 0.05%–0.43% Cu, and 0.16–1.54 g/t Au. Silver grades were between 1.0–3.0 g/t. An elevated arsenic content was measured for Composite 5 at approximately 0.16% As. Sulfide sulphur to copper ratios for many of the composites indicated potentially high pyrite to copper sulfide ratios, which would require chemical conditions to control pyrite flotation.

Results of the mineralogical analysis indicated that chalcopyrite was the predominant copper sulfide mineral. Approximately 4% of the copper was measured as bornite, and 1% as secondary copper sulfide minerals and arsenic sulphosalts tennantite and enargite.

A flotation flowsheet was developed that included a copper and pyrite circuit with primary grinding to a nominal 150µm K_{80}. To control pyrite recovery, lime was used to elevate both the copper roughers and copper cleaner pH to 10, and a dithiophosphate collector 3477 was used as the copper collector. The copper rougher concentrate was reground to approximately 25 µm K_{80} to produce high grade concentrates from three cleaner stages for most of the composites. The copper circuit tailings fed a pyrite circuit where potassium amyl xanthate (PAX) was used as the pyrite collector. In a locked-cycle test, approximately 83% copper and 50% gold was recovered to a copper concentrate, which measured approximately 27% Cu, 46 g/t Au and 0.3% As. Approximately 17% gold was recovered to the pyrite concentrate, which measured 4 g/t Au.

Gold recoveries did not trend with sulfur recoveries, therefore a strong association of gold with pyrite does not appear to be evident. However, for Composite 5, which measured the highest arsenic content of the individual composites, gold recoveries trended closely with arsenic recoveries to the product streams, thereby potentially indicating a close association of gold with arsenopyrite for this particular feed type.

Comminution testing was completed with unique comminution composites, representing the same feed material as that used for the flotation test but using drill core and crushed rock samples. The composites were characterised as soft-to-medium hardness with respect to ball milling, and Bond ball mill work indices ranged between 11.0–15.7 kWh/t when using a closing screen of 150µm. Axb values derived from SMC tests ranged between 56 and 124. The Bond Crusher work index for six composites tested ranged between 3–7 kWh/t, which indicated very soft material in terms of crushing.

Further flotation testing is suggested to determine whether improvements to the copper and gold performance in the copper circuit is possible with changes to collector type and dosage, and copper regrind discharge sizing. Additionally, once a flowsheet is optimised, variability testwork is recommended

13.2.8 Overall Circuit Performance

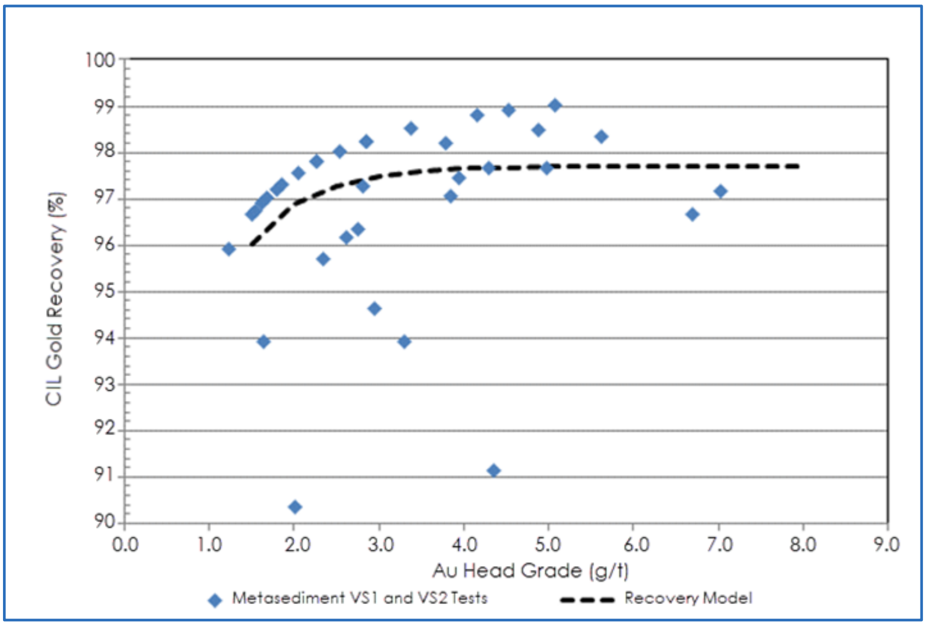
The recovery of gold across a laboratory carbon-in-pulp (CIP) circuit was measured for a number of variability samples representing each of the three major ore types.

In addition to the testwork, the commercial sulfide POX plant commenced commissioning in December 2018, with actual results reviewed to validate the recovery.

13.2.8.1 POX Gold Recovery

The gold recovery results of the acceptable tests are plotted in Figure 13.2, Figure 13.3, and Figure 13.4, together with an appropriate recovery model curve in each instance.

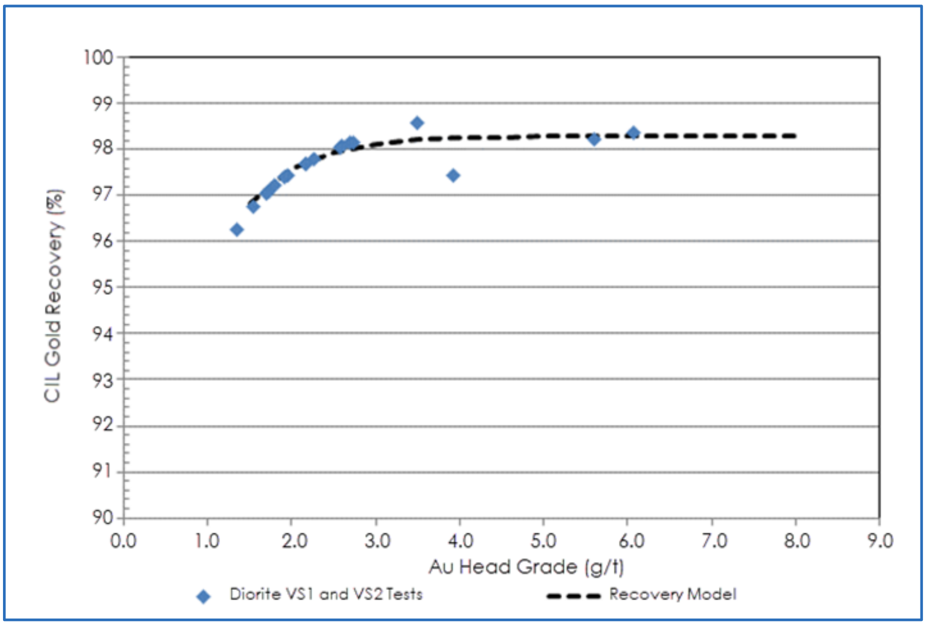
Figure 13.2 Metasediment Gold Recovery Results and Model



Anagold, 2016

The results are plotted in terms of feed grade so that predictions of recovery during operations can be made by knowing the feed grade.

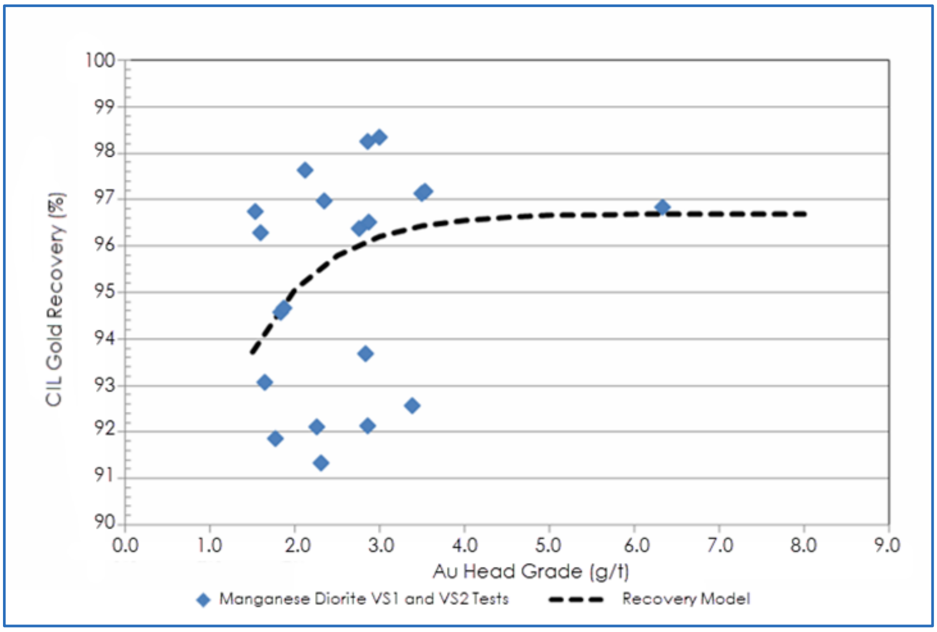
Figure 13.3 Diorite Gold Recovery and Model



Anagold, 2016

Note that Figure 13.2 and Figure 13.3 show a number of results that tend to form a regular curve at the top of the datasets. In each instance, where the results are on this curve the solid tails Au grade was below the limit of detection and an assigned tails grade, equal to half the limit of detection, was set for calculation purposes.

Figure 13.4 Manganese Diorite Gold Recovery and Model



Anagold, 2016

The recovery model is represented by the equation:

Gold Recovery (%) = a x (1 – exp (–b x (Au head grade in g/t – c))) + d

Parameter 'a' is the only one of the four that has a direct process meaning, representing the maximum recovery the equation can generate. The parameter 'd' represents circuit losses in a commercial operation.

The parameters used to generate the curves in Figure 13.2, Figure 13.3, and Figure 13.4 are shown in Table 13.5, and include an allowance for operational losses of 1%.

Table 13.5 Gold POX Recovery Model Parameters

Material Type	a	b	c	d
Metasediment	97.7	1.4	−1.4	−1.0
Diorite	98.3	1.4	−1.5	−1.0
Manganese Diorite	96.7	1.2	−1.4	−1.0

The POX commissioning and ramp-up allowances in Table 13.6 have been made on top of the base recoveries.

Table 13.6 Commissioning and Ramp-up Allowances

Recovery Corrections	Gold Recovery Deduction (%)
Commissioning to June 2019	–3.30
Ramp-up July 2019 to June 2020	–2.30
Flotation Commissioning	–0.75

13.2.8.2 POX Silver Recovery

The silver recovery pattern is much less clear than gold because silver is not released by the oxidation process. Silver recovery is determined from actual plant recovery over the period January 2019–February 2020.

The silver recovery calculates to 3.0%.

13.2.8.3 Flotation Gold Recovery

From the testwork, it is estimated that the flotation concentrate reporting to the POX circuit will achieve the same overall recovery as the ore directly reporting to POX. Gold recovery to the flotation concentrate is estimated to be 55%.

The flotation tails reporting directly to the leach circuit is estimated to have a gold recovery of 43%, based on testwork.

An allowance of 0.75% reduced gold recovery during commissioning and ramp-up of the flotation circuit (Year-1 of flotation operation) has been included.

13.3 Mineral Processing and Metallurgical Discussion

A large amount of POX testwork has been performed on Çöpler sulfide ore across several pilot plant campaigns. The processes used have been shown to be robust, as demonstrated through operational performance during commissioning, ramp-up, and operations.

The addition of a flotation circuit to the sulfide plant is estimated to provide stability and flexibility to the POX circuit operation to maximise throughput and oxygen utilisation by maintaining optimum sulfur grade to the autoclaves.

The final construction and commencement of commissioning of the flotation circuit in January 2022 is expected to confirm the assumptions developed in the design.

Ongoing testwork and analysis is also recommended on POX oxidation and leach recovery to improve and optimise circuit performance. This should include detailed assessment of gold deportment in final tailings.

Further metallurgical testing of Ardich material types, both oxide and sulfide, is recommended to optimise the feeds to the heap leach and POX and flotation circuits, respectively.

Further flotation testing on the copper–pyrite flowsheet is suggested to determine whether improvements to the copper and gold performance in the copper circuit is possible with changes to collector type and dosage, and copper regrind discharge sizing. Additionally, once a flowsheet is optimised, variability testwork is recommended.

14 MINERAL RESOURCES ESTIMATES

Mineral Resources for the project have been estimated using industry best practices and conform to the requirements of NI 43-101.

The resource model for Ardich has been updated in 2021 and is reported in detail here.

All other resource models are unchanged since the CDMP20TR, and the reader is directed to that report for the more detail on those resource models, with only summaries included here.

14.1 Çöpler Deposit

At Çöpler, a resource model was constructed to define the geometry of the gold mineralisation. Grades were estimated using exploration drilling data and then calibrated against the production grade control data. Steps for the gold modelling process included:

- Creation of wireframes that constrain gold mineralisation.

  This step incorporated structural trends to guide the shape of the wireframes along known geological features within the deposit. Mineralised trends commonly followed lithological contacts, such as the diorite / marble contact, and structural features identified by surface mapping.

- Gold mineralisation was then estimated using a method termed probability assigned constrained kriging (PACK) and then trimmed using the gold mineralisation shell.

  PACK first uses a probabilistic model or envelope (indicator envelope) to define the limits of the potentially economic mineralisation. The model cells and drillhole composites within these indicator envelopes were then used for grade estimations. The PACK process was designed to prevent economic grades inside the indicator envelope from being smeared into the waste and restricts low-grade material outside the indicator envelopes from diluting the mineralised material inside the envelope.

- The parameters used to construct the indicator envelopes were calibrated such that the estimated tonnes and grades approximated the historical production data.

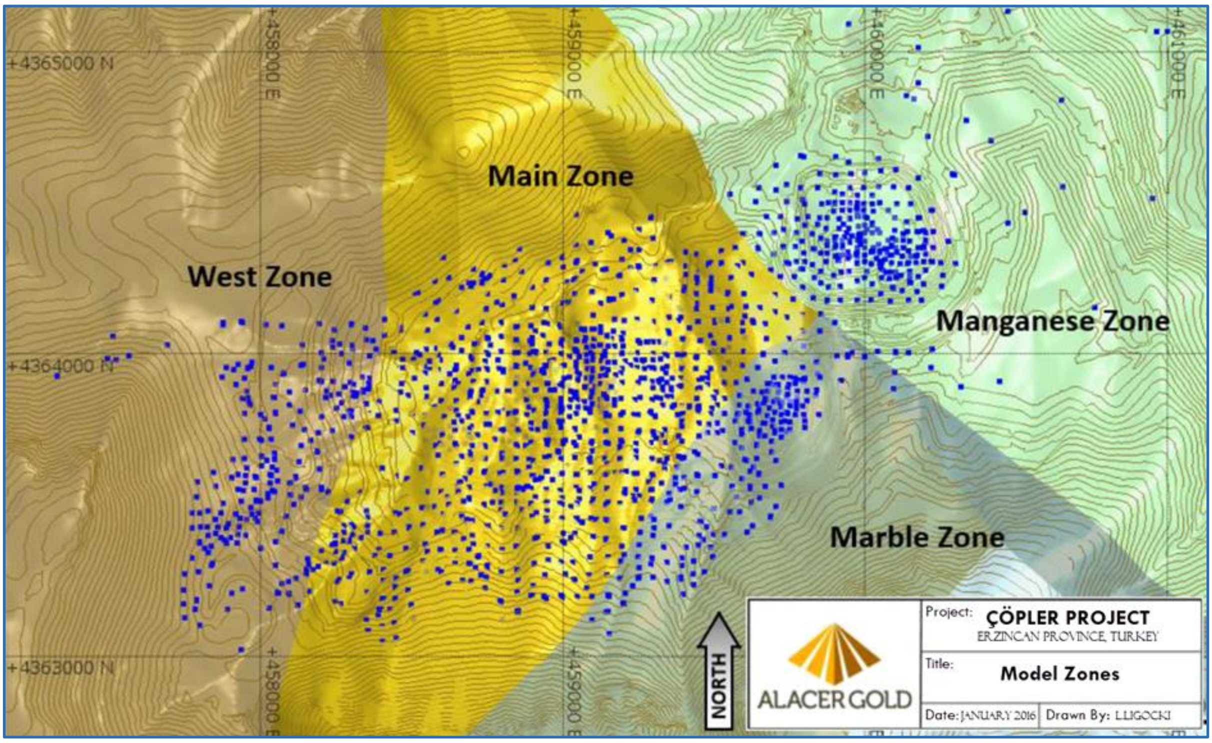
Au, Ag, and Cu were interpolated into the parent cells using ordinary kriging (OK), while As, Mn, Fe, and Zn were interpolated using inverse distance method, weighted to the power of two (ID2).

14.1.1 Çöpler Mineral Resource Estimate – Key Assumptions

The estimation methods at Çöpler were designed to address the variable nature of the epithermal, structural, and disseminated styles of gold mineralisation, while honouring the bi-modal distribution of the sulfur mineralisation and the oxide / sulfide boundary.

No obvious correlations were observed between Au and total sulfur; they were therefore domained and estimated separately. Au also showed little correlation with lithology and was therefore domained simply according to model zone (Manganese, Main, Marble, and West), to reflect the different trends of the mineralisation that commonly follow structures and lithological contacts (see Figure 14.1).

Figure 14.1 Çöpler Model Zones



Anagold, 2016

The percentage of total sulfur is the main criterion used to delineate between 'oxide' and 'sulfide' material types:

- Oxide material (S <2%) is processed using a heap leach method and has a cut-off grade of approximately 0.3 g/t Au.
- Sulfide material (S ⩾ 2%) is processed in the sulfide plant and has a cut-off grade of approximately 1.5 g/t Au.

Total sulfur assay data exhibits a bi-modal distribution with a distinct inflection point at 2% S, and also shows a good correlation with logged lithology. The 2% S inflection point also agrees well with a 1% pyrite break point in the drillhole logs.

As a result, sulfur was modelled using oxide and sulfide sub-domains within each lithology, and gold PACK models were constructed separately for oxide and sulfide within each lithology using the respective Au cut-offs.

The gold models were then reconciled to historical production data and the resource modelling parameters were adjusted to best match the historical data. Mineral Resource categories were applied to each model cell based on a combination of parameters including drillhole density and data quality.

14.1.2 Çöpler Base Indicator Model

A parent cell size of 10 m x 10 m x 5 m was selected, with the 10 m easting and northing dimensions representing approximately one half the average drillhole spacing, and the 5 m height of the cells representing the mining bench height. Cell model prototype parameters are provided in Table 14.1. The Mineral Resource model has an implicit selective mining unit (SMU) size of 5 m x 10 m x 5 m. The cell model is not rotated.

Table 14.1 Çöpler Block Model Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
East	457,100	461,100	4,000	10	400
North	4,362,500	4,365,100	2,600	10	260
RL	400	1,750	1,350	5	270

Drillhole intervals were composited to 10 m down-hole lengths and then assigned Au indicator values based on their composited Au grade. The sulfur indicator values were assigned to 5 m composites. Composites below the threshold were assigned '0' and composites at or above the threshold were assigned '1'.

Gold and sulfur indicator values were then interpolated into the parent cell model. The interpolated indicators represent a distance-weighted average of the composite indicators that occur within the search neighbourhood and therefore have values anywhere in the range 0–1. The interpolated indicator was used to create an envelope encapsulating the mineralisation above 0.3 g/t Au (the indicator envelope).

Exploratory data analyses (EDA) and capping studies were performed on samples within the indicator envelope.

14.1.3 Çöpler Domains

The model cells within the indicator envelope were assigned into four zones that represent the four geologically distinct zones (Manganese, Main, Marble, and West) using wireframe solids.

The position of the boundary between the Manganese Zone and the Main Zone was selected between discrete diorite intrusive events. The boundaries for the Marble Zone were selected along one limb of a diorite intrusion associated with a region of higher grade gold mineralisation. The boundary direction then follows the north-easterly trend of the mineralisation. The extension of this boundary includes a larger discrete diorite intrusion that carries minor gold mineralisation along its contact with the metasediment.

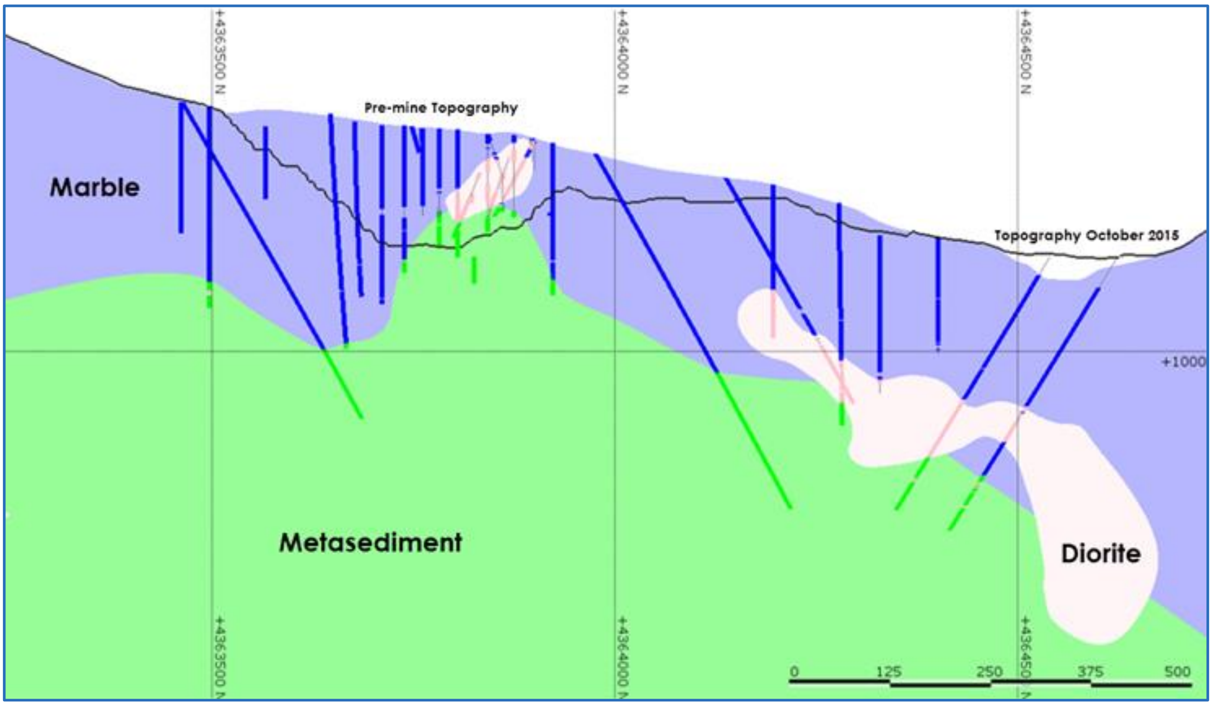
The tops of the model zone boundaries wireframe solids were trimmed to the original (pre-mining) topography.

14.1.4 Çöpler Geological Model

Exploration drillhole data and surface mapping were used to create 3D solid interpretation wireframes for the four main geological units: marble, diorite, metasediment, and manganese diorite. Surface mapping was used to provide indicative contact locations in areas of sparse drilling. In areas where the two datasets did not match, priority was given to the drillhole data. Blasthole data were not used to generate the lithology interpretations but were referenced to provide guidance in zones of wide-spaced drilling and in areas with missing drillhole data. The interpretation was adjusted in the Manganese Zone after referencing the blasthole data.

A typical cross-section illustrating the lithology interpretation at Çöpler is shown in Figure 14.2.

Figure 14.2 Çöpler Lithology – Marble Zone, Cross-Section 459,700 mE (looking west)



Anagold, 2016

14.1.5 Çöpler Data Summary

The cut-off date for the export of the drillholes from the database to be used in the resource modelling was 15 July 2015. The extract contained 1,957 drillholes with a total of 297,798.2 m of drilling. Of this, a total of 1,880 drillholes have collar coordinates within the extents used to construct the resource model. In general, the drillhole spacing ranged from 5–60 m, averaging approximately 20 m. Most drillholes are either vertical or inclined at 60°. Approximately 2% of the drillholes had missing assays; these were set to a null value and not used in the statistics or mineral resource estimation.

14.1.6 Çöpler Exploratory Data Analysis

Detailed exploratory data analysis (EDA) was conducted on the Çöpler resource modelling dataset. This is discussed in detail in the CDMP20TR. A summary of findings of the statistical analyses follow.

14.1.6.1 Çöpler Statistical Summary

Detailed statistical analyses were undertaken to assist with the understanding of the mineralisation distribution in the various domains. The statistical review included typical univariate statistics (tabulations, histograms, box plots) and bivariate statistics (scatter plots, correlations).

A summary of key findings follows:

- A histogram of sulfur grade in the 1 m composites shows a bi-modal distribution, with the lowest mode at or near trace S (27% of the dataset), and the second mode at approximately 3.25% S (7% of the dataset).
- Mean Au grade statistics are similar for diorite, metasediment, and marble but higher in the manganese diorite. When reviewing the data spatially, however, the higher grade Au mineralisation commonly occurs along the lithological contacts.
- Mean Ag grades are similar for diorite and metasediment, but lower in marble and higher in manganese diorite.
- Mean Cu grades varied between lithologies, but in general are higher in the diorite and metasediment.
- Mean Au grades in diorite, metasediment, and the marble are higher within the sulfide material. Manganese diorite carries a higher mean Au grade within the oxide material relative to the sulfide material.
- Distinctively different sulfur populations were observed for each lithology (although each lithology hosts both low and high-sulfur mineralisation) suggesting that sulfur should be domained by lithology for estimation. This approach was taken on the current model.
- The diorite, metasediment, and manganese diorite showed similar As grades, but the marble As was lower.
- There is moderate correlation between:
  - Au and As
  - Cu and Fe
- Minor correlations occur between:
  - Au and Ag
  - Ag and As
  - Ag and Mn

- While correlation probably exists between gold and sulfur on a mineralogical level, as suggested by the correlation between gold and arsenic, and the observed presence of arsenopyrite (FeAsS), this correlation is probably masked by the much larger episode of non-auriferous sulfide mineralisation. This suggests that it is reasonable to model silver, copper, zinc, arsenic, and manganese using the gold statistical model.
- Regarding core recovery:
  - No correlation was identified between any of the elemental grades and core recovery, and
  - There is no obvious increase or decrease in Au grade with lower core recovery.
- In a twinned hole analysis, RC and DD showed good agreement:
  - the average RC Au grade was slightly higher than the average DD hole grade.
  - No significant changes in grades were noted for the RC holes above or below the water table.
  - For sulfur, little difference in grade was noted between DD holes and RC holes.
  - For Cu, little difference was noted between the DD holes and RC holes, but the grades were very low.
- Contact plots were constructed for the different combinations of lithological contacts and categorised by material located within the oxide or sulfide portion of the deposit. In general, no hard contacts were observed for Au. The higher grade Au mineralisation commonly occurs along the lithological contacts, which indicated that the gold mineralisation should not be modelled separately for each of the lithological domains.

14.1.7 Çöpler Top Cutting

In mineral deposits with skewed distributions, it is not uncommon for a small number of the highest assays to account for a significant and disproportionate quantity of the total metal content in the model estimates. Although these assays are real and reproducible, they commonly show little continuity, and can add a significant amount of uncertainty to a mineral resource estimate.

One method of constraining the influence of these samples is to apply a top cut to the assays before compositing and grade estimation. Top cutting was performed on the 1 m composites prior to compositing into the 5 m composites used for the grade estimations. Au was studied and capped by domain and low and high-sulfur category. Top cut thresholds for Ag, Cu, S, As, Fe, Mn, and Zn were applied globally. The top cut thresholds applied before compositing are summarised in Table 14.2 and Table 14.3.

Table 14.2 Çöpler Top Cuts for Au

Domain	Top Cut Au (g/t)
Oxide (S <2%)	
Manganese Zone	18
Main Zone	16
Marble Zone	30
West Zone	16
Sulfide (S ⩾2%)	
Manganese Zone	18
Main Zone	14
Marble Zone	25
West Zone	14

Table 14.3 Çöpler Top Cuts for Non-Au Elements, Applied Globally

Element	Unit	Top Cut
Ag	g/t	300
Cu	%	5
S	%	20
C	%	13
As	ppm	30,000
Fe	%	50
Mn	ppm	100,000
Zn	ppm	60,000

14.1.8 Çöpler Drillhole Compositing

Samples used for grade estimation were prepared by first compositing the raw sample lengths to 1 m down-hole intervals. Au composites were capped globally at 40 g/t Au for the EDA. The 1 m composites were subsequently top cut at the relevant threshold according to the statistics of each model zone and oxide / sulfide domain. These 1 m composites were then composited into 5 m down-hole for additional statistical analysis and grade estimation.

The 5 m composite interval for grade estimation was selected as it was considered to notionally match the mining bench height. The 5 m composites were not truncated at lithological contacts, nor domain boundaries.

14.1.9 Çöpler Variography

The EDA showed that the trends of the Au mineralisation followed lithological contacts and structures that vary by domain. As a result, variograms (correlograms) were calculated for Au, Ag, and Cu composites for each domain categorised by oxide (S <2%) and sulfide (S ⩾2%).

The directions of the anisotropy axes were determined by creating multi-directional variograms, variogram models, and visual observation of the tabular shaped trends of the mineralisation. After the anisotropy had been determined, three directional variograms were calculated and modelled in each of the three primary directions of anisotropy. Given the low and high-sulfur domain variograms showed similar structures, albeit with the low-sulfur domain variogram structures better defined, the low-sulfur domain variograms were used for the grade estimation.

14.1.10 Çöpler Resource Model Estimation

14.1.11 Çöpler Sulfur Model

The total sulfur model was designed to emulate the hard 2% S threshold used during ore control to delineate material to be processed on the heap leach pad or sent to the pressure oxidation (POX) plant.

EDA showed that sulfur should be modelled separately in each of the four main lithological units (diorite, metasediment, marble, and manganese diorite). The sulfur estimate proved to be very sensitive. Minor changes in the estimation parameters causes the reclassification of material from high to low-sulfur and vice versa. The change in the sulfur categorisation has an impact on what cut-off grade is used and what mining and processing cost is applied.

To match the proportion of material greater than and less than 2% sulfur in each lithological domain, a sulfur indicator was generated using a discriminator of 2% sulfur. To accomplish this, a sulfur indicator field was created in the drillhole data, and populated as follows:

- S Indicator = 0 where S < 2%
- S Indicator = 1 where S ⩾ 2%

The S indicator was then interpolated into the cell model using nearest-neighbour (NN) and inverse distance method, weighted to the power of two (ID2) methods. The ID2 interpolated indicators represent a distance-weighted average of the composite indicators and therefore have values anywhere in the range 0–1. In contrast, the NN interpolated indicators represent only the closest composite indicator and therefore can only have the value '0' or '1'.

The number of cells above and below 2% sulfur was initially defined using the NN result (Indicator 0 = S <2% and Indicator 1 = S ⩾2%). The ID2 indicator estimate was calibrated against the NN model to make the proportion of low and high-sulfur material honour the NN proportions.

Sulfur indicator ID2 estimate thresholds that honoured the results of the NN estimation for low and high-sulfur proportions were:

- Diorite = 0.50
- Metasediments = 0.51
- Marble = 0.26
- Manganese diorite = 0.36

A soft boundary approach was achieved at lithological contacts by slightly raising the maximum indicator estimate for the oxide estimate and lowering the minimum indicator estimate for the sulfide estimate.

The sulfur model was not constrained by the mineralisation envelope. This means sulfur was also estimated into the waste rock cells; this was for the purpose of waste rock characterisation.

14.1.12 Çöpler Gold and Other Metal Models

A total of nine elements, Au, Ag, Cu, S, C, Zn, Fe, As, and Mn were estimated. Au, Cu, and Ag were estimated using OK and the remaining elements were interpolated using the ID2 method. Zn, Fe, As, and Mn, which are only used for material-type classification, were restricted to within the mineralisation envelope. All cells were estimated using a discretisation matrix of 3 x 3 x 1.

The volume of the mineralisation envelope was calibrated to past production by:

1. Creating a production cell model:
   - Constructing a 3 m x 3 m x 5 m cell model in the areas that had already been mined.
   - Populating the 3 m x 3 m x 5 m cells with the ore control tonnes and grades estimated from blasthole assays.
   - Tabulating ore control tonnes and grade from January 2014–October 2015.
2. Building an indicator model and estimation of gold grade:
   - The low-grade estimates were achieved using an indicator approach defined by an 0.3 g/t Au discriminator. First a low-grade Au indicator field was established in the drillhole 5 m composite file: if the composite grade was <0.3 g/t Au, the low-grade indicator field was set to zero (IND1=0); if the composite grade was ⩾0.3 g/t Au, the low-grade indicator was set to one (IND1=1). The low-grade indicator was then interpolated into all cells using ID2, and those cells with an estimated low-grade indicator of greater than 0.3 (i.e., IND1 > 0.3) were selected to define the indicator envelope. Only composites within the indicator envelope were used to estimate the Au grade.

– Similarly, a high-grade gold estimate was developed using a high-sulfur indicator model with a discriminator of 1.5 g/t Au to reflect the higher cut-off required for processing the material through the POX plant. The high-grade gold estimate uses the same indicator estimate threshold of 0.3 (i.e., IND2 > 0.3) to define the boundary limits.

– The low-grade gold estimates were applied to those cells with estimated sulfur grades <2%, and the high-grade gold estimates were applied to those cells with estimated sulfur grades ⩾2% S.

3. Calibrating the PACK model:

– The PACK model parameters were then adjusted so that the gold ounces in the PACK model approximates the gold ounces reported from the ore control model.

– After the gold ounces were calibrated by zone and material type, cells with estimated Au grades below the selected indicator threshold were set to waste.

14.1.13 Çöpler Density Model

Density measurements were performed on representative DD core by the site exploration geologists. Measurements were taken using the wax-coated water displacement method (Archimedes method). Density data were reviewed spatially and statistically. Density values that fell outside expected upper and lower density limits (shown in Table 14.4) were considered to be outliers and removed.

Table 14.4 Çöpler Upper and Lower Density Limits by Lithology

Lithology	Density Lower Limit (t/m^3)	Density Upper Limit (t/m^3)
Diorite	1.7	3.5
Metasediment	1.7	3.5
Marble	1.7	3.5
Manganese Diorite	–	–

Density values were assigned to the cell model based on rock type and depth below the surface. The density samples were first flagged by lithological code. Since lithological codes were not available for many of the density samples, Lithology was assigned using the lithological wireframes for all density values.

Densities used in the resource model are summarised in Table 14.5.

Table 14.5 Density Values Assigned to the Çöpler Cell Model by Lithology and Vertical Depth Below Surface

Lithology	Depth (m)	No. Density Data	Assigned Density (t/m^3)
Diorite	0–20	111	2.22
	20–40	173	2.42
	40–60	155	2.44
	60+	1,653	2.50
Metasediment	0–20	86	2.38
	20–40	209	2.51
	40–60	219	2.54
	60+	1,769	2.63
Marble	all	1,099	2.57
Manganese Diorite	all	23	2.63

14.1.14 Çöpler Oxidation Model

The oxidation model reflects oxidation due to surficial weathering and/or oxidation resulting from the manganese alteration. Oxide (low-sulfur material (S <2%)) can be processed by heap leaching while sulfide (high-sulfur material (S ⩾2%)) is processed through the POX plant.

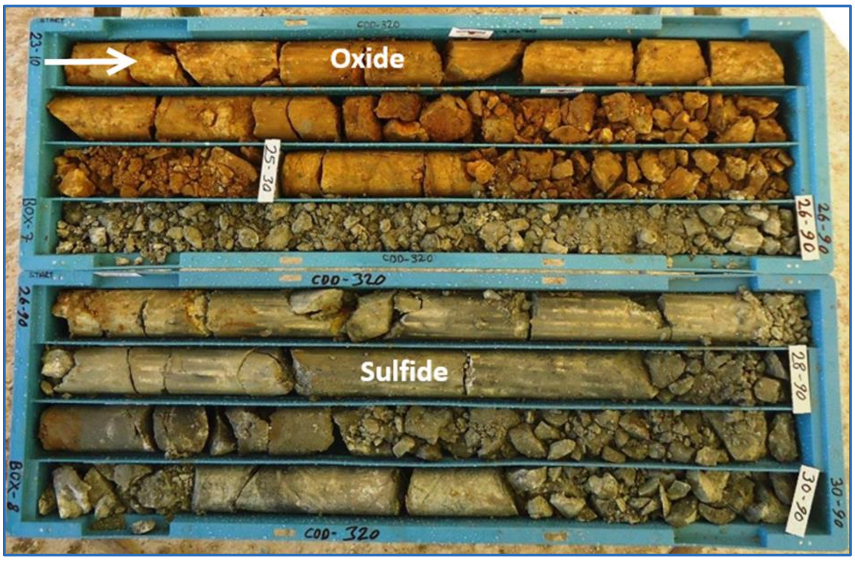
The low/high-sulfur criteria were further finessed using the logged colour codes and pyrite percentages recorded in the drillhole logs. Review of the logs showed a generally relatively sharp colour change from orange–brown tones to grey–black tones (Figure 14.3). A wireframe was constructed to represent this logged colour change. The wireframe was further refined using the logged visual estimates of pyrite. Near-surface material is highly oxidised and usually does not include visually identifiable sulfides, while visual sulfide percentage increases with depth to a point (pyrite ⩾1%) where the percent pyrite can be estimated and recorded in the drill logs. In general, the 1% visual pyrite boundary matched the red - grey colour boundary within approximately 5 m, but locally deviated up to 10 m. The 5 m variance is considered to be within the accuracy of the data, as it reflects the composite sample length and the mining bench height.

The resulting oxide-sulfide wireframe boundary was compared to the sulfur-estimates model. This comparison showed that the S <2% and S ⩾2% domains matched the oxide-sulfide boundary reasonably well, although there are local areas of material with S <2% below the oxide-sulfide surface which are due in part to deeper weathering along structures. As a result, the oxide boundary surface is considered to be somewhat conservative locally in estimating the amount of oxide material.

Blasthole data from Main Zone that contains both Au fire assays (AuFA) and cyanide leach assays (AuCN) show that the gold recovery significantly decreases below the oxide / sulfide boundary. This implies there is low-sulfur material below the oxide / sulfide boundary that has not oxidised, and hence lower recoveries are obtained by cyanide leaching. As a result, the oxide / sulfide boundary is used in the Main Zone to delineate material types. In the Manganese Zone and Marble Zone, however, the estimated sulfur content is used to delineate material.

In the eastern portion of the Çöpler deposit, the oxidation profile is better-developed and follows the diorite intrusion. This contrasts with the much shallower oxidation profile in the western portion of the mining operation.

Figure 14.3 Çöpler Drill Core Showing Colour Change from Oxide to Sulfide



Anagold, 2016

14.1.15 Çöpler Model Validation

Model validation was approached in several ways:

- The estimated Au grades in the model were compared to the composite grades by visual inspection in plan views, north–south cross-sections, and east–west cross-sections. In general, the model and composite grades compared well visually.
- The cell model was checked for global bias by comparing the mean Au, Ag, Cu, and S grades (with no cut-off) from the model (OK/ID2 grades) with means from NN estimates for cells of Indicated classification. The NN estimator produces a theoretically unbiased (de clustered) estimate of the mean value when no cut-off grade is imposed and provides a reasonable basis for checking the performance of different estimation methods. In general, an estimate is considered acceptable if the bias is at or below 5% (relative difference).

- Local trends in the grade estimates (also known as drift analysis) were assessed by plotting the mean values from the NN estimate versus the kriged results for Indicated model cells in east–west, north–south and vertical directions (swath plots). The global comparisons agree well, however the swath plots do illustrate the existence of slight local differences between the NN and kriged model grades.

14.1.16 Çöpler Mineral Resource Classification

Grade estimates were classified using the following Anagold guidelines:

- Indicated Mineral Resource should be quantified within relative ±15% with 90% confidence on an annual basis, and
- Measured Mineral Resources should be known within ±15% with 90% confidence on a quarterly basis.

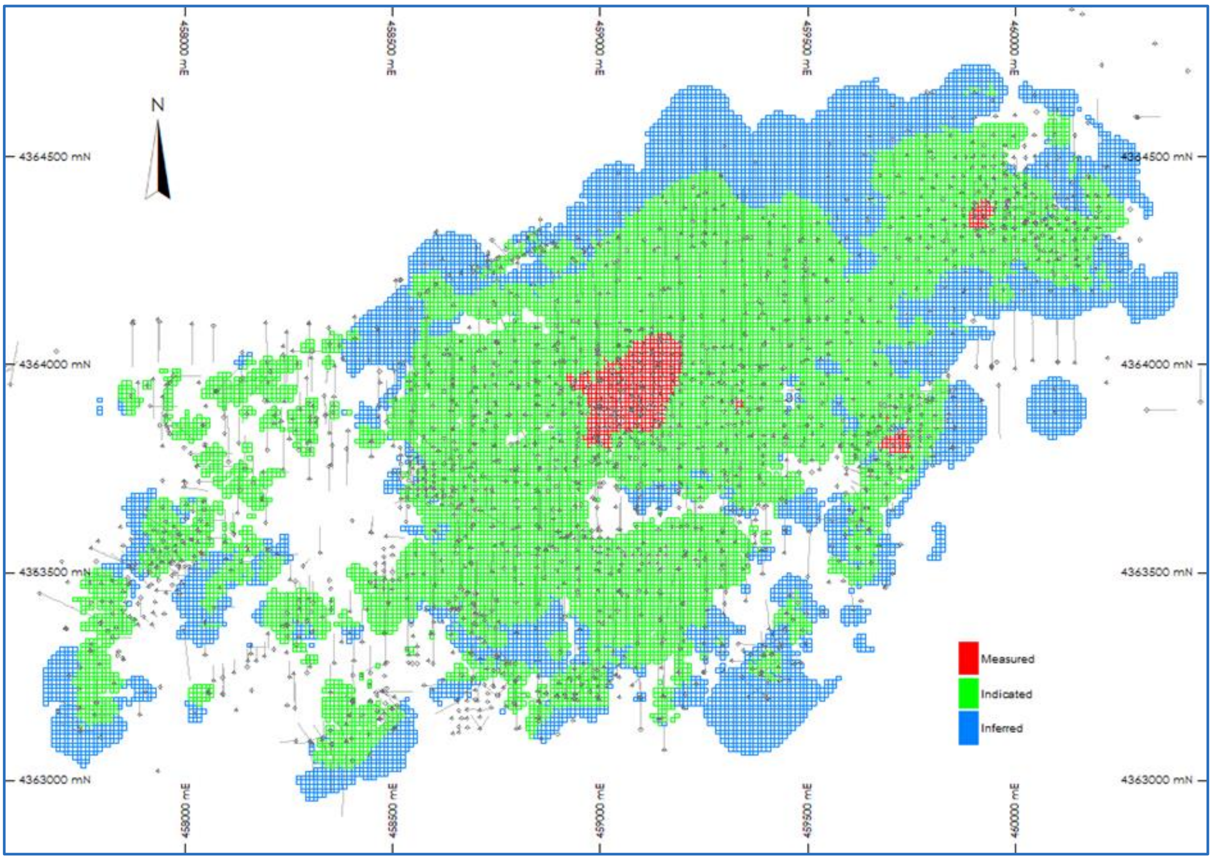
Based on these guidelines, the drilling is generally sufficiently close-spaced enough to permit confirmation of or assumption of continuity (Measured vs. Indicated, respectively) between data points. For the Çöpler model, a drillhole spacing study was performed to determine the nominal drillhole spacing required to classify material as Indicated.

Confidence limits were calculated on a single block that represents one month of POX production (based on 1.9 Mtpa). The confidence limits, a review of continuity on sections and plans, and an assessment of data quality were used to determine minimum drillhole spacing by domain. A spacing of 40 m x 40 m in the Marble Zone, 50 m x 50 m in the Manganese Zone and West Zones, and 60 m x 60 m in the Main Zone was required to meet the requirements for Indicated. An 80 m x 80 m spacing was required for Inferred in all domains. Model Cells with a drillhole spacing that was greater than 80 m were not classified as Mineral Resource.

The resultant classification was then 'smoothed' to remove the isolated cells that are not of the same classification tenor as the proximal surrounding cells.

The resulting classification shows that much of the deposit can be classified as Indicated, with Inferred cells forming a halo around the Indicated mineralisation Figure 14.4. A small quantity of cells classified as Measured.

Figure 14.4 Projected Plan View of Çöpler Resource Classification



OreWin, 2020
Only model cells with Au >0.3 g/t shown

14.1.17 Çöpler Model Validation

Model validation was approached in several ways:

- The estimated Au grades in the model were compared to the composite grades by visual inspection in plan views, north–south cross-sections, and east–west cross-sections. In general, the model and composite grades compared well visually.
- The cell model was checked for global bias by comparing the mean Au, Ag, Cu, and S grades (with no cut-off) from the model (OK/ID2 grades) with means from NN estimates for cells of Indicated classification. In general, an estimate is considered acceptable if the bias is at or below 5% (relative difference).
- Local trends in the grade estimates (also known as drift analysis) were assessed by plotting the mean values from the NN estimate versus the kriged results for Indicated model cells in east–west, north–south and vertical directions (swath plots). The global comparisons agree well, however swath plots illustrate the existence of slight local differences between the NN and kriged model grades.

14.1.18 Çöpler Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,750/oz for gold and the parameters summarised in Table 14.6. These parameters, with the exception of the gold price, are the same parameters as those used to define the Mineral Reserve pit.

Table 14.6 Summary of Key Parameters Used in 2021 Conceptual Pit Shell at Çöpler

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	62.3	78.4
POX Gold Recovery	%	91.0	91.0
Mining Cost per tonne mined	$/t	1.49	2.78
Process Costs Heap Leach	$/t	9.30	9.30
Process Costs POX	$/t	34.88	34.88
Site Support and G&A	$Mpa	15	15
Internal Au Cut-off – Heap Leach	g/t	0.19	0.24
Internal Au Cut-off – POX	$/t NSR	34.88	34.88
Internal Au Cut-off – Cu Conc.	$/t NSR	7.68 + 34.88 x Pyrite Mass Pull	
Royalty	%	2.0	2.0

14.1.19 Çöpler Deposit Mineral Resource Tabulation

Mineral Resources are reported exclusive of Mineral Reserves in Table 14.30 according to resource classification and material type. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability. The pit shell used to constrain the resource has been updated to reflect the increase in gold price. Depletion from mining has been included.

14.2 Çakmaktepe

The Çöpler district hosts various styles of mineralisation, mainly epithermal, skarn and contact style gold and gold–copper mineralisation. The Çakmaktepe North zone of the Çakmaktepe deposit is a strongly sheared zone with strong epithermal characteristics and grade associations with intrusive diorite dykes. As with the other prospects the mineral association is dominantly Au–Cu–Ag. Other mineralised zones belonging to the Çakmaktepe deposit are generally contact styles of mineralisation where Au–Cu–Ag have been emplaced along thrust surfaces next to ophiolite, limestone, and metasediment. Epithermal veining and replacement alteration textures are prevalent.

Oxide mining began in the Çakmaktepe Central and East pits in November 2018. Mining continued through September 2019 within the same two pits. Oxide ore material was transported to the Çöpler oxide processing facility for inclusion on the heap leach pad.

A geological model was constructed along with a cell model estimating grades for Au, Cu, Ag, S, and C. Estimated grades were constrained by mineralised envelopes.

14.2.1 Çakmaktepe Domains

At Çakmaktepe, mineralisation follows structural controls and designated lithological contact orientations. Grades trends and element associations were investigated, and several separate domains were identified and are shown in Figure 14.5.

Mineralisation at Çakmaktepe often overlaps multiple lithological units along its boundary, rather than being hosted within a single rock type. For this reason, grade shells were constructed for gold and copper to constrain estimates within mineralised zones. The mineralised shapes for gold and copper are lenticular with thicknesses ranging from 5–40 m, the average thicknesses being approximately 6 m.

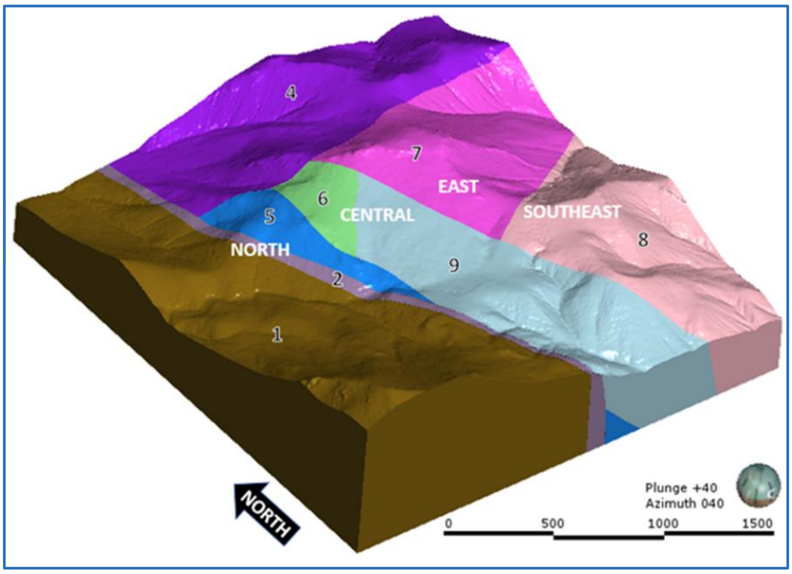
Grade shells were also developed for silver. However, because silver mineralisation tends to be more dispersed and more difficult to follow across the deposit than gold, different methods were used for silver grade shells depending on which area was being modelled.

Sulfur grades follow lithological units. Higher S values are seen in diorite and metasediment, with decreased S in gossan, jasperoid, ophiolite, and marble.

The key points in relation to Çakmaktepe mineralisation domains are:

- Çakmaktepe North is located on a vertical shear structure with elevated metal grades within jasperoid unit. Several low-angle structures dipping to the north-east carry grades along the marble to metasediment contact. Intrusive diorite/s, orientated vertically, cross-cut all other lithological units. Mineralisation within/around the diorite is limited in Çakmaktepe North.
- Çakmaktepe Central mineralisation follows the marble contact, which dips gradually to the north-east. The marble unit is approximately 15 m thick and located between the ophiolite and metasediment units.
- Mineralisation in the Çakmaktepe East area is near-surface and within the gossan unit, which is relatively flat lying and localised.
- The South-east area seems to be controlled by a massive diorite body with gossan at the surface. Mineralisation is weak and near-surface.

Figure 14.5 Çakmaktepe Model Domains (oblique view)



Anagold, 2020

Contacts for lithological shapes used the raw logged interval depth in 3D space. Surfaces were generated through implicit modelling of contact locations in the drillholes.

Construction of the lithological shapes assumed the following:

- Diorites are intrusive units that can exist as large bodies or thin sills cross-cutting other units.
- Jasperoid is an alteration product but treated here as a lithological unit. Jasperoids occur along shear zones and are high in pyrite. Jasperoid can exist in pods and can be discordant to surrounding stratigraphy.
- Gossan is primarily the result of surficial oxidation, with the shape influenced by the local topographic elevation.
- In most areas, marble overlays metasediment, with ophiolite above marble.
- Offsets in lithological units help to define fault locations and structural boundaries.

A series of fault surface wireframes were developed to represent the structural knowledge at Çakmaktepe. These structures extended beyond the Çakmaktepe model area to take into consideration the spatial relationships between Çakmaktepe and Ardich. The incorporation of modelled 3D faults into the geological model highlighted a discrepancy between the Ardich lithological concept and the Çakmaktepe geological units. Given the correlation of the two deposits was not clearly defined at the time of this model, interpreted faults were excluded from the Çakmaktepe geological model.

14.2.2 Çakmaktepe Data Summary

The cut-off date for the export of the drillholes from the database to be used in the resource modelling was 31 October 2019. The extract contained 1,109 drillholes with a drilling date range of September 2007–October 2019, totalling of 119,001 m of drilling.

14.2.2.1 Çakmaktepe Drillhole Compositing

The original sample lengths in the Çakmaktepe dataset are predominately 1 m, with some 2 m sampling through zones presumed at the time of drilling to be waste. The average sample length is 1.02 m. The shortest interval was 0.1 m and the maximum length was 3.1 m.

Samples were composited to 5 m lengths for use in statistical analysis and construction of mineralisation boundaries. Often, composites along lithological boundaries were selected to match geological control with mineralisation.

Composites were then flagged within the mineralisation shapes. Lithology is also coded into the composite file based on the interpreted shapes.

14.2.3 Çakmaktepe Exploratory Data Analysis

Detailed exploratory data analysis (EDA) was conducted on the Çakmaktepe resource modelling dataset. This is discussed in detail in the CDMP20TR. A summary of findings of the statistical analyses follows.

14.2.3.1 Çakmaktepe Statistical Summary

Detailed statistical analyses were undertaken to assist with the understanding of the mineralisation distribution in the various domains. The statistical review included typical univariate statistics (tabulations, histograms, box plots) and bivariate statistics (scatter plots, correlations).

A summary of key findings follows:

- Box plots confirm observations made from histograms and probability plots that gossan and jasperoid contain significantly higher Au grades and the remaining units (diorite, metasediment, ophiolite, and marble) have lower Au grades.
- Box plots of sulfur show higher sulfur content in diorite and metasediment with moderate sulfur grades in gossan and jasperoid. Low-sulfur is consistently seen in ophiolite and marble. For this reason, the sulfur estimate uses lithologic contacts as domain boundaries.
- Mineralisation tends to spatially follow lithological contacts.
- For Çakmaktepe Central, the probability plot is relatively straight, indicating only one population is present in the distribution.
- Core recoveries are between 80–90%, reflecting strongly sheared, brecciated, altered and in areas of limestone, karstic ground (cavities) being drilled at Çakmaktepe.

- Contact plots were created to show grades change across geological boundaries. Jasperoid and gossan are favourable mineralisation hosts and show abrupt grade changes when compared to the other lithologies (marble, metasediment, diorite, ophiolite).

Grade shell boundaries were constructed to follow lithological contacts and were used as hard domains in the grade estimation process.

14.2.4 Çakmaktepe Top Cutting

Top cuts were selected based on the log probability plot, supported by the projection of the data trend to the expected upper grade (y-axis value) using the top sample value curve (Table 14.7). Top cutting occurred after compositing to 5 m. A spatial review of top values by domain shows randomly spaced samples rather than a localised body of higher grades.

High-yield limits were included outside of the grade shells to restrict the extrapolation of higher grades within the applied search distance. For Au, sample values above 4 g/t Au were restricted to a distance of 10 m x 10 m x 5 m in the East and South-east areas. For Central, a high-yield limit of 8 g/t Au was used. The high-yield limit was increased to 12 g/t Au in the Çakmaktepe North area. For copper, samples above 2% Cu were restricted to 10 m x 10 m x 5 m in Central and 3% Cu in North and East.

Table 14.7 Çakmaktepe Top Cuts for Au, Cu, and Ag

Element	Çakmaktepe Area	Top Cut Grade	No. Samples Cut
Au (g/t)	North	15.0	2
	Central	9.0	7
	East	5.5	1
	South-east	5.0	4
Cu (%)	North	4.0	2
	Central	3.0	2
	East	4.0	2
	South-east	1.0	2
Ag (g/t)	North	180	2
	Central	130	3
	East	150	5
	South-east	60	5

14.2.5 Çakmaktepe Resource Model Estimation

14.2.5.1 Çakmaktepe Cell Model

A cell model was constructed by first coding the interpreted lithology shapes into the cells. These were then flagged by each of the grade shells and model domains. A project-wide solid was used to trim out distant cells at model edges.

The cell model limits are shown in Table 14.8.

The model was not rotated, and no sub-celling was used.

Table 14.8 Çakmaktepe Cell Model Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
East	463,400	465,700	2,300	5	460
North	4,364,800	4,366,700	1,900	5	380
RL	1,050	1,850	800	5	160

14.2.5.2 Çakmaktepe Estimation Method

Au, Ag, Cu, S, and C were interpolated using ID3 and NN methods. Au, Cu, and Ag were estimated according to grade shell constraints. S and C were estimated by modelled lithological units. All grade shell boundaries were treated as hard. Mineralisation domains were treated as soft boundaries allowing the selection of samples from nearby domains.

A single search distance was used within the Au, Cu, and Ag grade shells. A two-pass method was used to estimate cells outside of the grade shells. Search ranges and sample requirements varied by estimation pass. Search orientations were selected to match the mineralised dip and dip-direction.

Au was interpolated within each gold grade shell using only composite samples inside the shell. Au grade was then interpolated into cells outside the grade shell using domain-specific parameters. Cu was estimated using the same method as Au, by first interpolating grade within the copper grade shells and then interpolating outside the grade shells in two-passes. Ag estimation followed the same technique as Au and Cu by interpolating within the silver grade shell and then interpolating outside the grade shell by domain.

Sulfur and carbon content is linked to lithology. Lithological shapes were used as hard boundaries to interpolate S and C grades. No preferred orientation of S or C grades was observed; therefore, a spherical search was used.

A NN estimate was completed for all variables using the same composites, same domains, same search ranges and same top cut values as the ID3 estimates. The resulting NN model was used for estimation validation to detect potential estimation bias by domain.

14.2.6 Çakmaktepe Density Model

Density measurements were collected on DD core samples spaced nominally 3 m apart down-hole. Density values were statistically analysed by lithology with outliers and non-representative values excluded from the analysis.

A review of histograms of density within each rock type aided in the selection of bottom and top cut values.

Selected lower and upper cut density values by lithology are shown in Table 14.9.

Table 14.9 Çakmaktepe Upper and Lower Density Limits by Lithology

Lithology	No. of Density Data	Bottom Cut	Top Cut
Cataclasite	33	2.60	2.62
Diorite	1,496	2.00	3.00
Gossan	407	2.00	2.90
Jasperoid	1,972	2.00	3.20
Listwanite	29	2.28	2.80
Marble	4,041	1.91	3.50
Metasediment	4,114	2.00	3.30
Ophiolite	3,400	2.00	3.00

Densities used in the resource model are summarised in Table 14.10.

Table 14.10 Density Values Assigned to the Çakmaktepe Cell Model by Lithology

Lithology	No. Density Data	Assigned Density (t/m^3)
Gossan	389	2.48
Jasperoid	1,755	2.60
Diorite	1,186	2.54
Ophiolite	2,870	2.41
Metasediment	3,666	2.65
Marble	3,533	2.64

14.2.7 Çakmaktepe Model Validation

Validation of the 5 m x 5 m x 5 m model estimates included visual inspection of grade estimates, comparisons of cell grades to drillhole data, checks for global bias, check of local bias (swath plot), metal reduction calculation, and comparison of estimates within the Central and East areas using grade tonnage curves.

Visual inspection of plans and sections and 3D visualisation confirmed that the cell model estimates honour the drillhole data and grade shell boundaries.

14.2.8 Çakmaktepe Comparison to Production Data

Mining occurred in the Çakmaktepe Central and East pits, primarily during 2019. Blasthole data from these two pits were used to construct an Au grade estimate for comparison of the production model to the Mineral Resource model.

The production model, using blasthole assay data, was set up to follow the same parent cell size used in the resource model of 5 m x 5 m x 5 m. This generates cell centroids with the same centroid coordinates as the resource model for relational comparisons by cell.

Plotting the grade / tonnage curve for Au shows the number of tonnes to be similar in both models, with a crossover of the resource to production model tonnes occurring between the 0.8–2.8 g/t Au cut-offs. A large variance is seen when comparing Au grades between the two models. The increased grade in the production model results in more gold ounces. The largest positive and negative variances between the two models were investigated. The following observations were made:

- Estimate variances exist throughout the two cell models. An overall bias towards higher grade blastholes results in higher cell grades in the production model.
- Comparison of cut-off grades shows a larger variance in gold ounces between the two models as the cut-off grade is increased. Variances were plotted on a grade / tonnage curve by pit for a comparison of gold ounces by area.
- Variances were not limited to specific locations. Positive and negative variances were mixed throughout the Central and East pit. This suggests the selected modelling method for the resource grade estimation is not bias high or low, but likely producing a gold model more generalised than the variability seen within the deposit.
- When using the ID3 interpolation method, cell grades closely match drillhole composite values. Investigation of areas where exploration drilling crosses cells shows lower estimated grades in the resource model and higher estimated grades in the production model. This illustrates the variance in the two drillhole datasets – exploration to blasthole data.

These observations indicate that the variances between the two datasets are likely greater than the software tools available to match the deposit grade distribution and short range variability to the resource model. To compensate for the model variances, increasing the exploration drill density to the deposit variability is preferred. However, increasing the exploration drill density is probably not feasible due to the high inherent variance seen in the deposit. This presents a risk that mining may not match the predictive abilities of the resource model using the available exploration data.

14.2.9 Çakmaktepe Mineral Resource Classification

In summary, assignment of model classification followed these steps:

- Sample spacing was calculated based on samples from drillholes containing assay values. The calculation of sample spacing did not use limiting boundaries such as domains or lithological shapes.
- Inferred and indicated classification was assigned based on drill sample distances (20 m and 35 m).
- Indicated classification was then restricted to those cells within the modelled mineral grade shells for gold, copper, and silver.
- South-east estimates were downgraded to Inferred.

Grade estimates within the grade shells were visually confirmed by comparing the grade of the cell with the grade shell boundary. Higher grades exist inside the grade shell with a drop in grade tenor evident when crossing the grade shell boundary. Grade shells follow geological features such as lithological contacts and the Çakmaktepe North shear structure. Estimates outside of the grade shells were set to generalised orientations honouring the trends of the low-grade mineralisation and orientations of the major lithological units.

Hard grade boundaries were used for gold, silver, and copper. The sharp changes in grade are expected, rather than being an artefact of the estimate, due to the close relationship between mineralisation and structural features. This relationship is supported by close-spaced drilling throughout Çakmaktepe and crossing holes in areas such as the shear zone in Çakmaktepe North.

14.2.10 Çakmaktepe Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,750/oz for gold with the parameters summarised in Table 14.11. These parameters, with the exception of the gold price, are the same parameters as those used to define the Çakmaktepe Mineral Reserve pit.

Table 14.11 Summary of Key Parameters Used in Conceptual Pit Shell at Çakmaktepe

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	38.0	80.0
Mining Cost per tonne mined	$/t	1.59	1.59
Process Costs Heap Leach	$/t	14.16	14.16
Site Support per tonne processed	$/t	3.17	3.17
Internal Au Cut-off – Heap Leach	g/t	0.36	0.76
Royalty	%	4.0	4.0

14.2.11 Çakmaktepe Mineral Resource Tabulation

Çakmaktepe Mineral Resources are reported inclusive of Mineral Reserves and have been tabulated by resource classification and oxidation state in Table 14.30. Mineral Resources are presented on a 100% basis.

14.3 Ardich

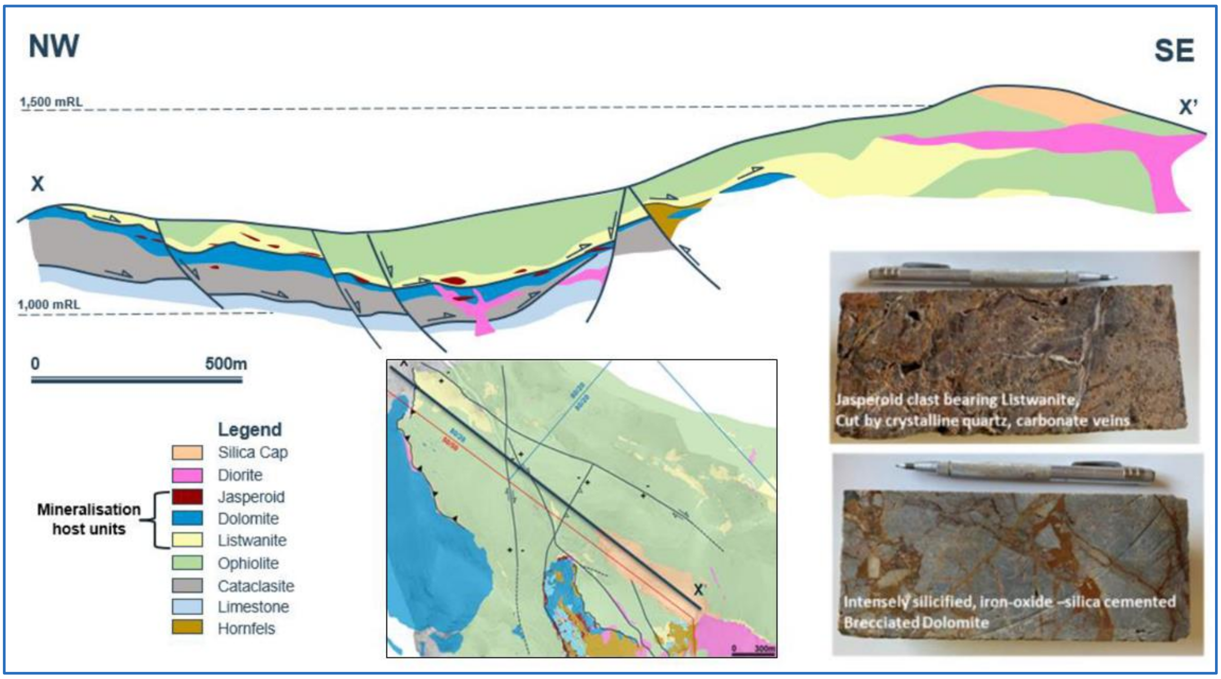
The latest mineral asset to be intensively studied in the Çöpler district suite of mineralised zones is Ardich, which is located approximately 6 km east of the current Çöpler pit and 1 km north of the Çakmaktepe pits. The Ardich deposit is accessed by the İliç-Yakuplu village road, which is open throughout the year.

Ardich mineralisation was discovered in August 2017. Ardich does not appear to have hosted historical mining or trenching in the way that Çöpler and Çakmaktepe have.

The local geology at Ardich is dominated by ophiolites, listwanites, dolomites, cataclasites and limestones, with lesser amounts of jasperoid and diorite Figure 14.6. This lithology assemblage occurs within a complex north-west trending structural zone that is cut by multiple high-angle faults, which together result in multiple rotated fault blocks and mineralised zones.

The jasperoids and diorites form a volumetrically minor part of the lithology sequence, however, they appear to be the most important controls of the mineralisation at Ardich.

Figure 14.6 Ardich Geology Schematic (long-section)



Anagold, 2021

14.3.1 Ardich Geological Model

The 2021 Ardich model update is underpinned by the structural and lithological interpretations developed for the previous geological modelling at Ardich.

Overall, the lithological and structural framework for the geological model remains conceptually similar to the previous interpretations. However, the July 2021 update includes six additional faults, which dissect the deposit into more structural domains than in previous modelling.

Standalone lithological interpretations were developed for each structural domain. This was necessary to enable the offsets / discontinuities and changes in behaviour of the lithological units in each structural domain to be represented appropriately.

14.3.2 Ardich Structural Interpretation

There are two dominant structural trends interpreted at Ardich:

1. North-west / south-east trending faults form the primary structural features.
2. A series of smaller, less pervasive secondary faults cross-cut the primary structures in a north-east / south-west trend.

The combination of these faults creates unique structural blocks that have moved vertically relative to each other as well as pivoted / rotated. The nomenclature for, and characteristics of, each of the interpreted faults is listed in Table 14.12.

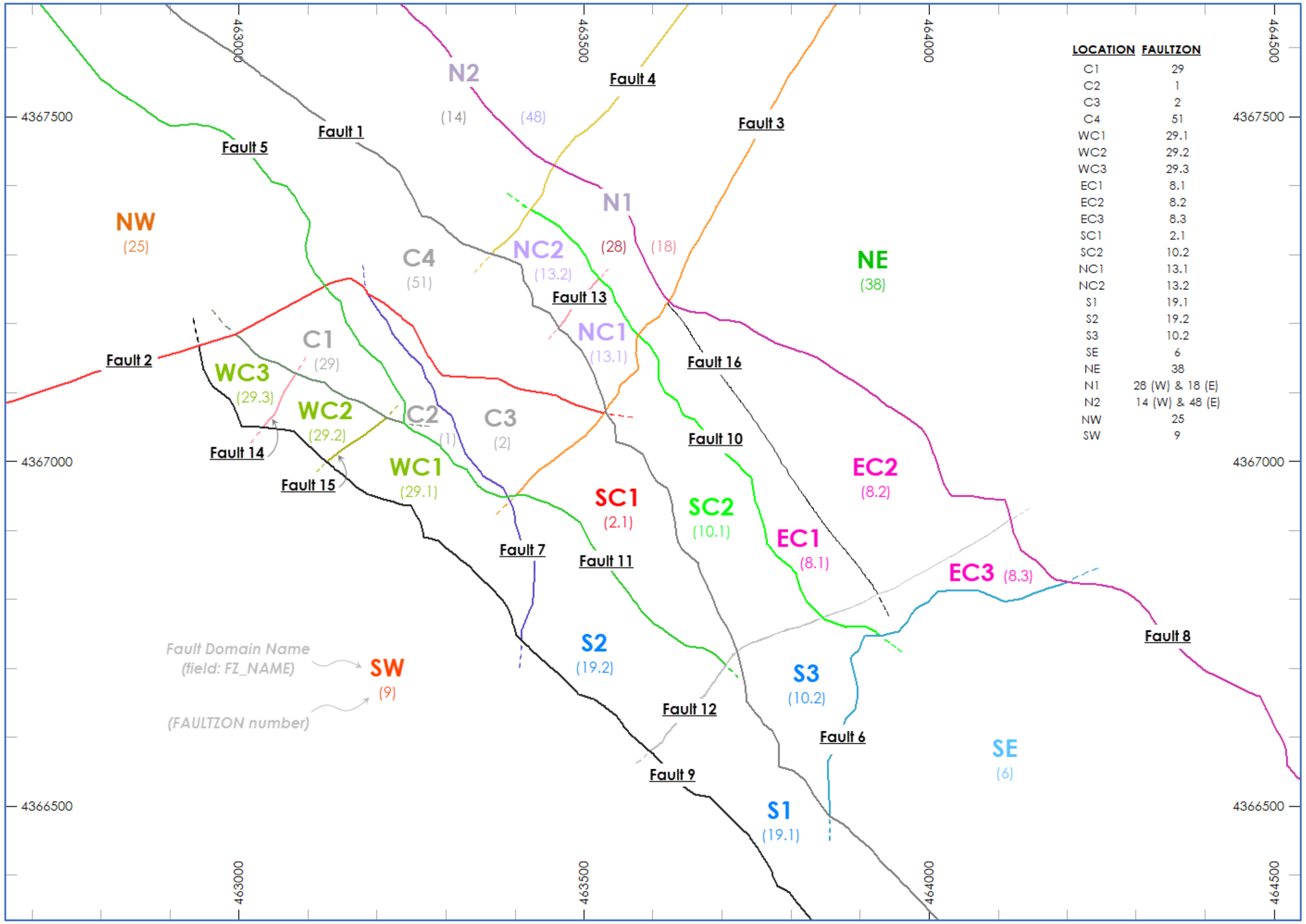
The location and attitude of the faults was largely determined from incongruous lithological occurrences observed down drillholes, further enhanced at times by logged commentary regarding evidence for faulting in the lithological dataset.

Table 14.12 Interpreted Fault Trends

Fault Number	Trending Direction	Fault Dominance
1, 5, 8, 9	North-west / South-east	Primary
7, 10, 11, 16	North-west / South-east	Secondary
2	East / West	Primary
3, 6	North-east / South-west	Primary
4, 12, 13, 14, 15	North-east / South-west	Secondary

The delineation of the deposit by the fault wireframes allowed for the creation of a domain field named 'FAULTZON' in each drillhole sample and model cell to identify its location within the structural framework. The interactions of the faults resulted in 25 unique FAULTZON domains. The location and inter-relationships of the faults and the FAULTZONs are shown in plan view in Figure 14.7.

Figure 14.7 Ardich Structural Framework: Interpreted Faults and Resultant FAULTZONs



OreWin, 2021

14.3.3 Ardich Lithological Interpretation

The typical stratigraphic sequence encountered at Ardich is (from surface down): ophiolite, listwanite, dolomite, cataclasite, and limestone, with jasperoid and diorite variably present within any of these strata. An example cross section is shown in Figure 14.8.

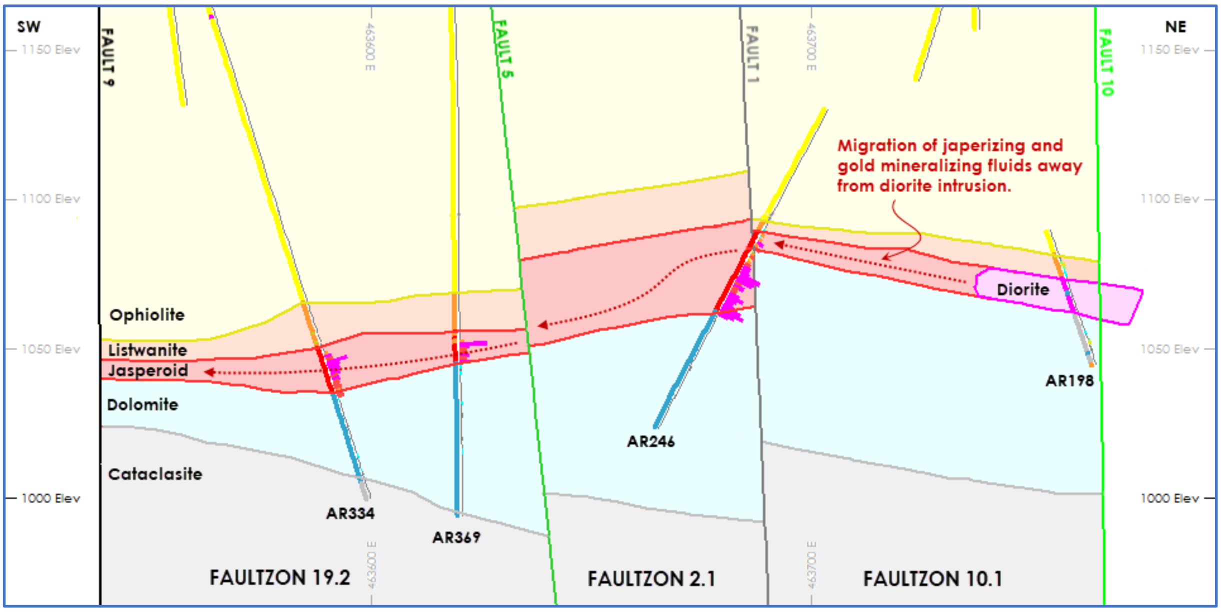
In addition to this typical sequence, secondary repeat occurrences of ophiolite, listwanite, dolomite, and cataclasite can be present in the stratigraphic sequence in some FAULTZONs. A silica cap unit is present in FAULTZON 6 (SE).

The jasperoid and diorite lithologies have a relatively small volumetric presence and were formed by different geological mechanisms compared to the more extensive (meta)sedimentary stratigraphic host lithologies (ophiolite, dolomite, cataclasite, hornfels, and limestone).

Wireframe surfaces and solids were created to represent the interpreted nature of the main lithologies. The field 'MODLITH' was created in the sample and model files to identify the lithology domain that the sample or cell represents.

An additional field 'LITHNUM' was also created in the sample and model files to provide a numerical identifier for each lithology to differentiate between the repeat occurrences of the same lithological units within a FAULTZON.

Figure 14.8 Schematic Example Cross-Section of Ardich Lithology Model (looking north-west)



OreWin, 2021

14.3.4 Ardich Mineralisation

The mineralisation at Ardich is related to crystalline and chalcedonic quartz veins within the brecciated and silicified jasperoid, listwanite, and dolomite zones. The mineralisation is predominantly in the form of oxide, with sulfide mineralisation confined to limited pyrite-rich jasperoid zones.

The mineralisation is considered to be related to fluids associated with diorite intrusions. This manifests as either direct contact mineralisation surrounding the diorites or as diorite-derived mineralising fluids that have migrated along lithological or fault contacts.

Gold grades increase at jasperoid / dolomite / listwanite contacts and within the silica-rich listwanites, which act as horizontal traps for higher grade gold-bearing mineralisation. Increases in gold grade often occur along the lithological contacts. A rapid down-hole change in gold grade tenor, notionally from mineralised to unmineralised material, can be seen in many drillholes, indicating that the mineralisation is tightly constrained within the controlling features rather than generally disseminated across the deposit.

Four distinct styles of mineralisation have been observed:

- Jasperoid-related mineralisation

  In this most-prevalent style of mineralisation at Ardich, gold mineralisation can either occur throughout the entirety of a jasperoid unit or display a concentration towards the middle-to-lower part of the unit. The formation of the jasperoid, and the gold mineralisation more broadly, is interpreted to be related to the fluids associated with diorite intrusions.

  The diorites from which jasperising and mineralising fluids have been interpreted to originate can be located either within a single FAUTLZON or in an adjacent FAULTZON to the mineralised jasperoid, with the listwanite / dolomite contact providing a low-resistance fluid pathway between the diorite and the jasperoid.

- Diorite contact-related mineralisation

  The second most common style of mineralisation is related directly to the contact of a diorite intrusion with the host strata; most-commonly within the listwanite and in the absence of jasperoid.

  This style of mineralisation tends to be parallel to the diorite intrusion, but generally limited to within approximately 20 m of the diorite contact.

- Contact-parallel mineralisation

  Both the well-mineralised jasperoid / listwanite domains and the less well-mineralised dolomites show contact-parallel mineralisation.

  In the jasperoids, this often manifests as higher grade results (within an overall well-mineralised zone) occurring at a similar distance from the contact (i.e., contact-parallel).

  The mineralisation is likely related to a sedimentary / depositional feature such as bedding within the unit. Like the lithological contact being preferentially used as a low resistance fluid pathway for the mineralising fluids, internal variation within the units may offer additional low resistant pathways for the fluids.

- Complex domains mineralisation

  There are a small number of FAUTLZONs that are more structurally and lithologically complex than the majority of the FAULTZONs in the deposit. The mineralisation within these complex domains does still appear to adhere to the jasperoid-related, contact-parallel, and diorite contact-related mineralisation models. However, the number of lithology contacts and the geometry of these units appears to be more complex than in other parts of the deposit.

14.3.5 Ardich Database Extract

A MS Access database 'Ardich_12072021.mdb' (the July database) was supplied to OreWin on 13 July 2021. The database contained collar, survey, assay, lithology, and density data for drilling completed up to 29 May 2021. The tables contained within the database and used in this study are:

- tblDHColl
- tblDHSurv
- tblDHLithology2017
- tblVWDHAssays_ALL
- tblDHSpecGrav

The July database tables were exported to comma delimited text format (csv) then imported in to Datamine software for validation and to create the working drillhole files.

The tables were reviewed extensively to identify any erroneous or unusual data. A comparison was also made between the July database and the previous database extract.

14.3.6 Ardich 2021 Resource Modelling Dataset Summary

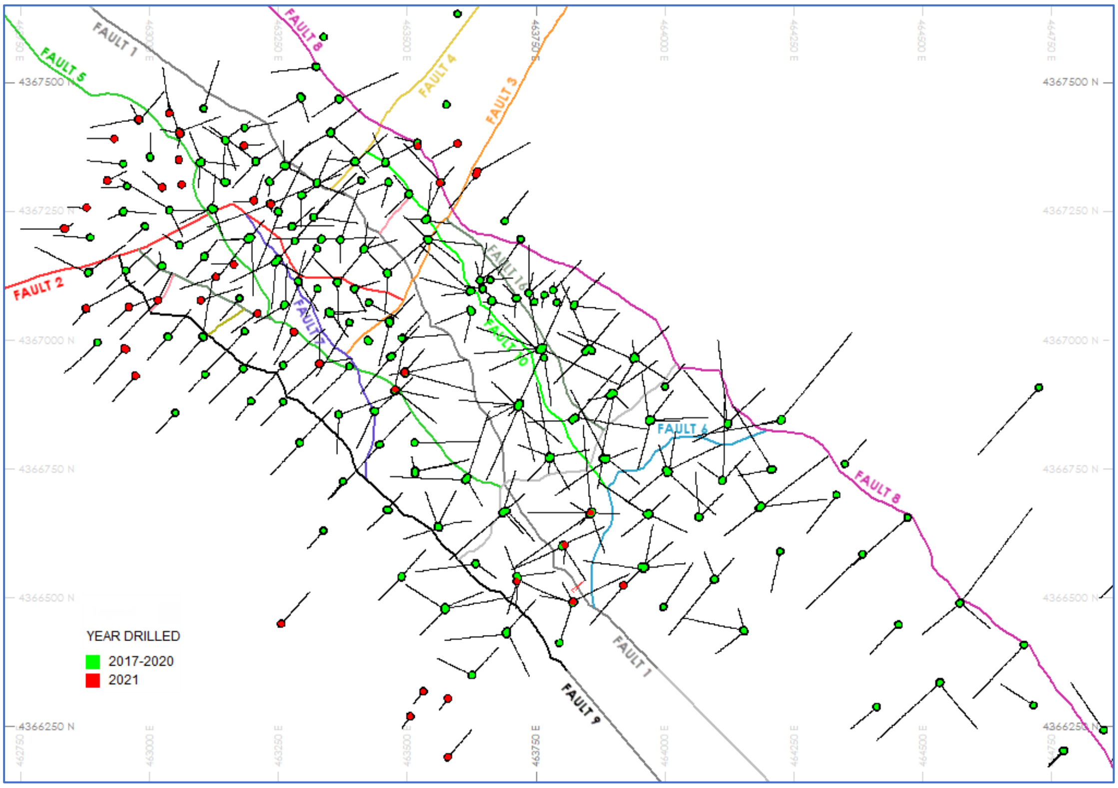
A total of 427 diamond drillholes (DD) have been drilled at Ardich since late-2017, (see Table 14.13 and Figure 14.9). Exploration drilling at Ardich utilised surface PQ and HQ triple-tube diamond core drilling. No RC drilling has occurred to date at Ardich.

After the initial discovery of the Ardich deposit, DD drilling programmes have continued to improve confidence in the interpretation.

Table 14.13 Ardich Drilling History in 2021 Resource Modelling Dataset

Year	Number of Drillholes	Drilled Metres
2017	9	1,374.10
2018	91	14,216.40
2019	133	27,821.20
2020	147	35,146.65
to 29 May 2021	45	8,479.90
Total	**427**	**87,038.25**

Figure 14.9 Ardich Drill Collar Location Plan



OreWin, 2021

14.3.7 Ardich Exploratory Data Analysis

In-depth analysis of the grade distribution and continuity of the assay data was undertaken both visually and statistically.

14.3.7.1 Ardich Summary Statistics

The raw statistics of the key elements in the global dataset are shown in Table 14.14.

Table 14.14 Key Element Statistics of Uncomposited Drillhole Data (length weighted)

ELEMENT	No. Samples	Min.	Max.	Mean	Std. Dev.	Variance	CV
Au (ppm)	72,432	0.0025	44.7	0.26	1.12	1.26	4.25
Ag (ppm)	72,432	0.25	246	0.68	2.95	8.67	4.32
Cu (%)	72,432	0.00005	19.4	0.005	0.11	0.01	22.87
S (%)	72,432	0.01	32.5	0.52	1.23	1.50	2.34

Statistics were generated for all lithologies present at Ardich.

It can be observed from the summary statistics that the mean grades, the number of samples and the range of grades varies significantly across the lithologies and FAULTZONs.

Histograms of Au data at the domain level show that most domains have a lognormal grade distribution. Also seen in the histograms for many of the domains is evidence of mixed grade populations. The presence of higher grade samples proximal to the lithology contacts provides a geological explanation to the mixed grade distributions seen in the some of the histograms.

Sulfur content is more strongly related to the lithological unit than the gold mineralisation. While some increase in sulfur grades was observed proximal to the lithological contacts, this phenomenon was not of the same tenor or uniformity as is evident in the gold mineralisation.

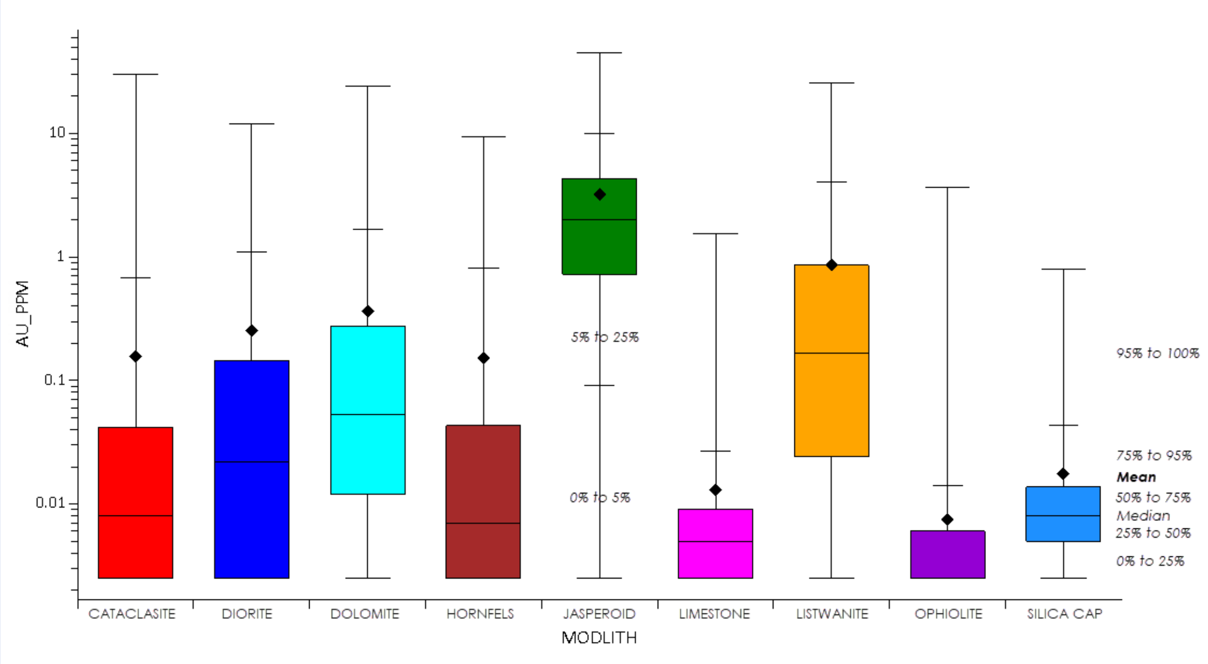
Gold Statistics

As can be seen in Table 14.15 and Figure 14.10, the highest mean Au grades are in the jasperoid and listwanite.

Table 14.15 Au Summary Statistics by Lithology based on Uncomposited Samples (length weighted)

Lithology	No. of Samples	% of all Samples	Au (ppm)		
			Min.	Max.	Mean
Ophiolite	29,425	41%	0.0025	3.7	0.01
Listwanite	8,723	12%	0.0025	25.4	0.86
Dolomite	15,070	21%	0.0025	24.1	0.36
Cataclasite	8,990	12%	0.0025	30.3	0.16
Jasperoid	2,058	3%	0.0025	44.7	3.21
Limestone	1,852	3%	0.0025	1.5	0.01
Diorite	2,203	3%	0.0025	11.9	0.25
Hornfels	3,447	5%	0.0025	9.3	0.15
Silica Cap	664	1%	0.0025	0.8	0.02
ALL	**72,432**	**100%**	**0.0025**	**44.7**	**0.31**

Figure 14.10 AU_PPM Box and Whisker Plot by Lithology (MODLITH)



OreWin, 2021

A detailed visual assessment of the Au grade distribution showed a preference for higher Au grades to be concentrated on or near a lithological contact. The following observations were made:

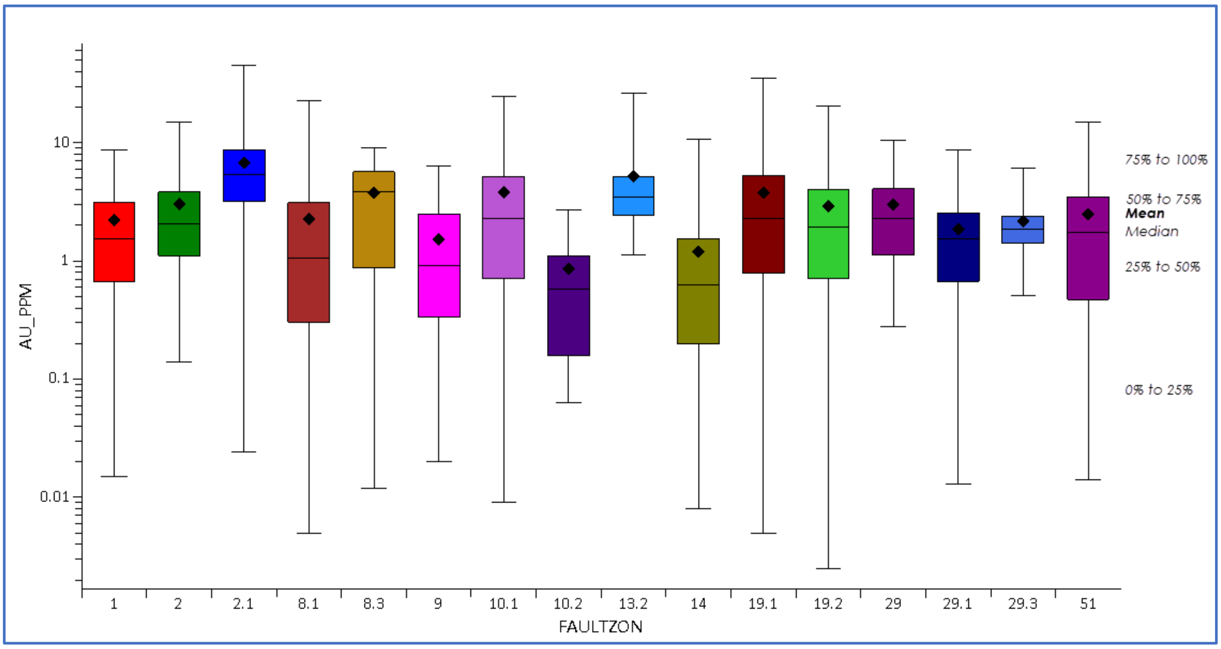
- For the jasperoid and listwanite, the higher Au grades were predominately located on or near the lower lithological contact. In addition to the preferential occurrence of grades proximal to contacts, a grade trend with a contact-parallel orientation was also noted.
- For the dolomite and cataclasite, higher Au grades were associated with the upper lithological contact. These lithologies also displayed areas of contact-parallel mineralisation within the main body of the unit.
- Higher Au grades were also noted in the host rock surrounding some, but not all, of the diorite intrusions and occasionally proximal to the faults. Consistent with the mineralisation observed on the main lithology contacts, there appears to be a contact-parallel nature to the mineralisation surrounding the diorites.
- Areas of strongest diorite intrusion often coincide with the areas of greatest jasperisation and gold mineralisation. The presence of the highest Au grades proximal to the lithological contacts suggests the preference of these contact boundaries as a low resistance pathway for the diorite derived mineralising fluids.
- Gold mineralisation in the listwanite is most-commonly located in the lower part of the unit, occurring on either the listwanite / jasperoid, or listwanite / dolomite contact.

- Gold mineralisation in the dolomite is most-commonly located in the upper part of the unit, close to the dolomite / jasperoid or dolomite / listwanite contact. Where the listwanite, jasperoid, and dolomite are all mineralised within a single FAULTZON, the Au grade tenor is often (but not always) highest within the jasperoid.

Differences in Au grade were observed between the different lithologies at the deposit scale. In addition to this, there were also noticeable differences in Au grade for the same lithologies in different FAULTZONs. As an example, Figure 14.11 shows the range and mean Au grades of the jasperoid for the various FAULTZONs.

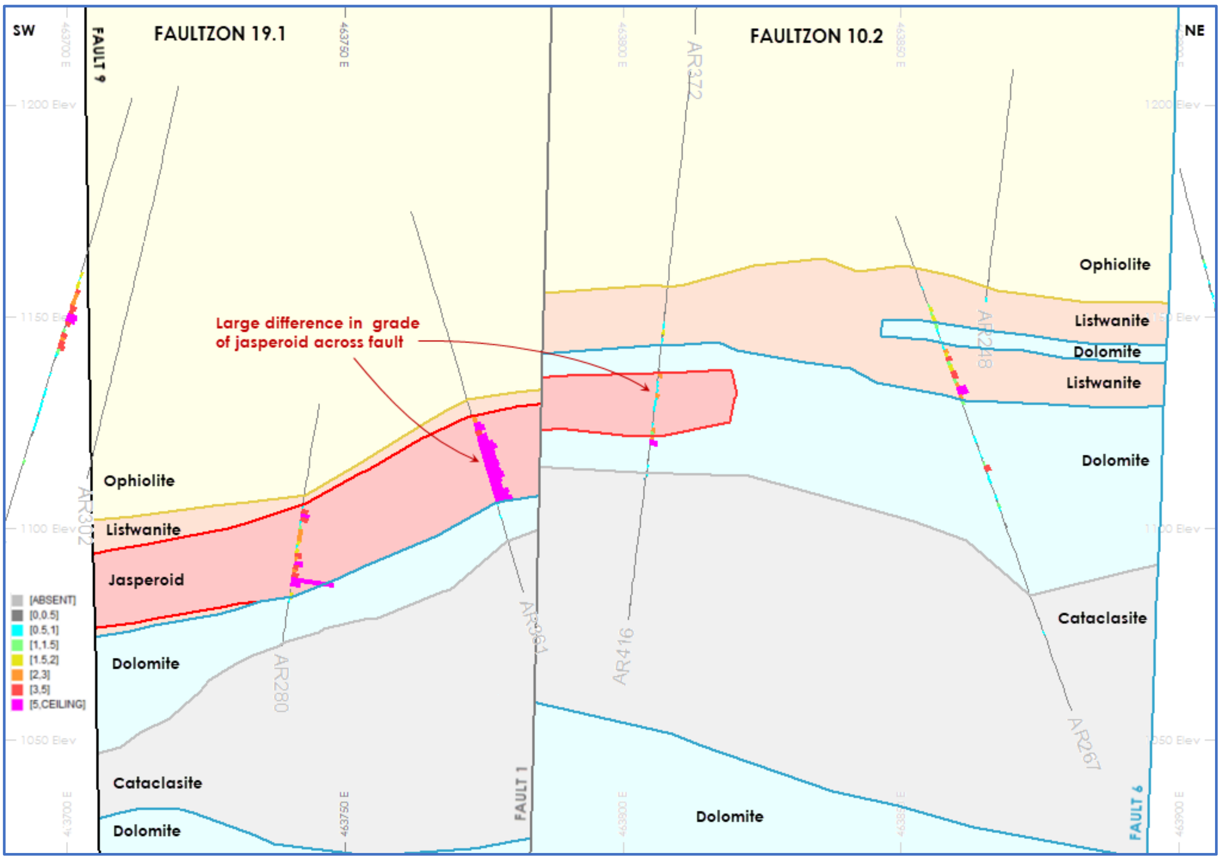
This variation in Au grade between FAULTZONs is illustrated in Figure 14.12, which shows a very large difference in grade tenor and location in the jasperoid in AR361 in FAULTZON 10.2 (S1) relative to the jasperoid in AR416 in FAULTZON 10.2 (S3). The jasperoid hosts only 3% of the total samples collected.

Figure 14.11 Jasperoid AU_PPM Box and Whisker Plot by FAULTZON



OreWin, 2021

Figure 14.12 Example Cross-Section (looking north-west) through FAULTZONs 19.1 and 10.2 showing AU_PPM Differences in Adjacent FAULTZONs



OreWin, 2021

Both the visual and statistical observations of the Au grade distribution in the different FAULTZON blocks informed the decision to estimate each lithology within each FAULTZON independently.

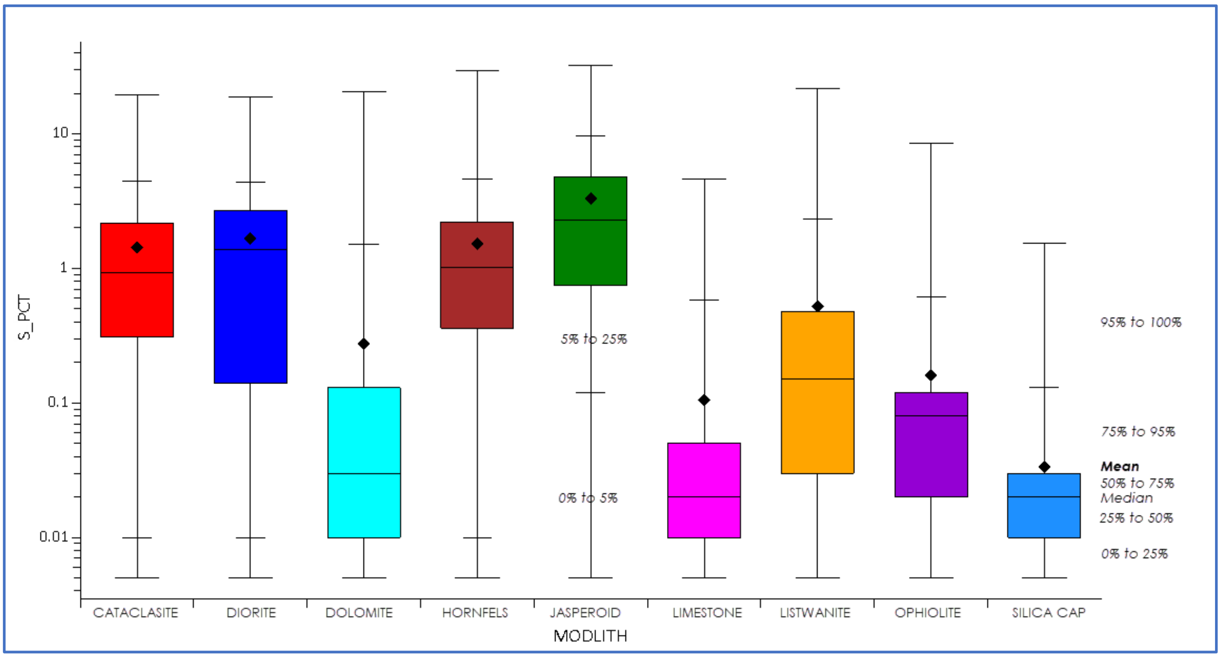
Sulfur Statistics

Table 14.16 and Figure 14.13 show the shows the basic statistics for sulfur in each lithology at the deposit scale. This shows that the lithology with the highest mean S grade is the jasperoid (3.3% S). There is a strong correlation between lithology and sulfur content.

Table 14.16 Sulfur Summary Statistics by Lithology based on Length Weighted Uncomposited Samples

MODLITH	No. of Samples	% of all Samples	S (%)		
			Min.	Max.	Mean
Ophiolite	29,425	41%	0.005	8.5	0.16
Listwanite	8,723	12%	0.005	21.9	0.52
Dolomite	15,070	21%	0.005	20.5	0.28
Cataclasite	8,990	12%	0.005	19.6	1.44
Jasperoid	2,058	3%	0.005	32.5	3.30
Limestone	1,852	3%	0.005	4.6	0.11
Diorite	2,203	3%	0.005	18.9	1.67
Hornfels	3,447	5%	0.005	29.3	1.52
Silica Cap	664	1%	0.005	1.6	0.03
ALL	**72,432**	**100%**	**0.005**	**32.5**	**0.52**

Figure 14.13 S_PCT Box and Whisker Plot by MODLITH



OreWin, 2021

14.3.8 Ardich Core Recovery

Exploration drilling at Ardich utilised surface PQ and HQ triple-tube diamond core drilling.

Overall, Ardich drill core recovery is good with a mean recovery over 92%.

Uncomposited data statistics were compared to core recovery collected during geotechnical logging. No correlation is seen between Au grade and core recovery.

14.3.9 Ardich Top Cutting

Statistical analysis, using log. probability plots, mean variance plots, log. histograms, and percentage change statistics, was undertaken for each of the domains. Once identified, apparently-outlying samples were reviewed visually in 3D to determine whether there was sufficient local support to allow the sample to retain its un-cut grade, or if grade cutting was required. Often, samples that appeared as statistical outliers at the domain scale transpired to be well-supported at the local scale and conformed to the mineralisation model of higher grades located proximal to lithological or fault contacts, indicating that top cutting may not be justified.

This review process resulted in top cuts being applied to only 41 samples from 24 domains. The domains where top cutting was applied are summarised in Table 14.17. This table shows the effect of the top cutting process on the statistics of the cut samples. Sulfur grades were also assessed for potential top cutting with no top cuts applied

.

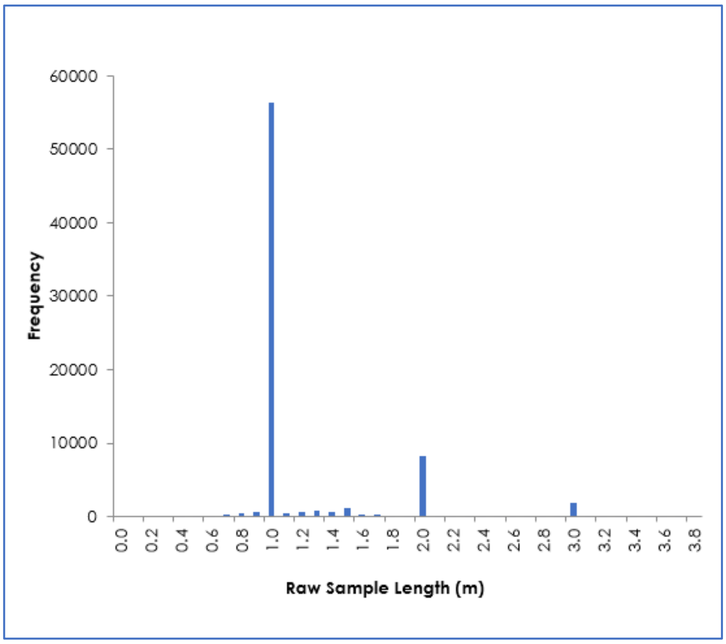
Table 14.17 Au Top Cut Values and Effect on Statistics

LITHNUM	Lithology	FAULTZON	No. Samples		Top Cut Value	Top Cut Percentile	Mean (Au ppm)		CV (Au)	
			Total	Cut			Un-cut	Cut	Un-cut	Cut
2	Listwanite	8.2	256	1	8.0	99.6%	0.88	0.86	1.73	1.57
		9	491	2	9.5	99.6%	0.61	0.58	2.36	2.01
		19.2	93	3	1.2	96.7%	0.22	0.14	3.20	2.19
3	Dolomite (upper)	8.2	1144	3	6.0	99.7%	0.28	0.27	2.87	2.26
		13.2	66	1	4.4	98.4%	0.69	0.62	2.01	1.65
		14	40	2	1.3	94.9%	0.34	0.21	2.54	1.63
		29.1	240	1	7.0	99.5%	0.74	0.68	2.19	1.41
3.1	Dolomite (lower)	51	292	1	2.0	99.7%	0.07	0.06	3.95	2.71
4	Cataclasite (upper)	2	503	2	12.0	99.6%	0.54	0.50	3.81	3.40
		10.1	713	1	5.0	99.8%	0.17	0.16	3.38	3.06
4.1	Cataclasite (lower)	13.1	269	2	0.3	99.2%	0.04	0.03	3.94	1.89
		29.2	33	2	0.4	93.8%	0.21	0.14	1.70	0.99
5.1	Jasperoid	2.1	296	3	24.0	99.0%	6.73	6.60	0.79	0.71
		8.1	221	3	12.0	99.0%	2.25	2.16	1.29	1.13
		13.2	23	2	9.0	91.0%	5.52	4.45	0.98	0.52
		14	107	2	6.5	98.1%	1.19	1.13	1.35	1.18
		19.1	9	1	1.0	88.3%	0.85	0.55	1.18	0.45
		29.3	15	1	3.5	92.9%	2.19	2.01	0.59	0.41
5.2	Jasperoid	19.1	197	2	18.0	98.9%	3.90	3.77	1.15	1.01
7.2	Diorite	25	28	1	2.0	97.2%	0.52	0.44	1.51	0.97
7.3	Diorite	8.1	64	1	2.0	98.7%	0.30	0.26	2.13	1.37
		13.1	11	1	2.0	90.0%	1.40	0.84	1.63	1.00
7.4	Diorite	8.2	48	1	0.7	98.1%	0.18	0.16	1.46	1.08
7.5	Diorite	51	73	2	7.0	97.3%	2.17	2.05	0.96	0.78

14.3.10 Ardich Drillhole Compositing

A 1 m sample interval was the most prevalent raw sample length in the assay database, accounting for 76.95% (55,733 samples) of all samples. The next most-prevalent sample length was 2 m, accounting for 11.17% (8,090 samples) of all samples. See Figure 14.14.

Figure 14.14 Histogram of Raw Sample Lengths



OreWin, 2021

Drillholes containing samples of 2 m length are located within almost every FAULTZON. However, the location of these samples was heavily biased to the ophiolite lithology, with 99.25% (8,029 of 8,090 samples) of the 2 m samples located in ophiolite.

In addition, there is a temporal bias in the collection of the 2 m sample lengths with all but three (3) of the 2 m samples being collected from hole AR194 and onwards. This indicates an understanding of the lithology / mineralisation relationship developed over time, for example, it became understood that the ophiolite contained little or no mineralisation and a 2 m sample interval could be used in this material without compromising the sample support of the mineralised areas.

After due consideration of the sample length analyses, compositing to a length of 1 m was determined to be optimal as this length would preserve the integrity of the majority of samples in the raw database, with minimal compromise to the modified samples.

During compositing, samples were not permitted to composite across any of the boundaries defined by the FAULTZON or LITHNUM attributes. Compositing to 1 m had a very little impact on the statistics of the data. Table 14.18 and Table 14.19 shows the effect of the compositing process for the jasperoid and listwanite within each FAULTZON area. The full summary of the effect of compositing across all domains can be found in Table 14.18.

Table 14.18 Drillhole Au Statistics for Uncomposited (Raw) and Composited (Comp.) Sample Data – Jasperoid

LITHNUM	FAULTZON	No. Samples		Max. Length (m)		Max. Grade (Au ppm)		Mean Grade (Au ppm)		CV (Au)	
		Raw	Comp.	Raw	Comp.	Raw	Comp.	Raw	Comp.	Raw	Comp.
5.1	1	74	75	1.5	1.0	8.69	8.69	2.04	2.05	0.98	0.95
	2	161	166	1.6	1.0	14.90	14.90	3.04	3.05	0.91	0.88
	2.1	292	296	1.8	1.0	44.70	42.00	6.74	6.73	0.82	0.79
	8.1	218	221	1.6	1.0	22.90	22.90	2.26	2.25	1.34	1.29
	8.3	23	23	1.3	1.0	8.99	8.99	3.75	3.75	0.72	0.73
	9	119	121	1.8	1.0	6.32	6.32	1.53	1.54	0.96	0.93
	10.1	92	94	1.5	1.0	24.50	23.15	3.90	3.86	1.11	1.09
	10.2	35	36	2.2	1.0	2.70	2.70	0.86	0.86	0.93	0.89
	13.2	21	23	1.7	1.0	26.20	26.20	5.26	5.52	1.04	0.98
	14	104	107	1.6	1.0	10.60	10.60	1.20	1.19	1.42	1.35
	19.1	9	9	1.0	1.0	3.63	3.63	0.85	0.85	1.19	1.19
	19.2	208	216	2.9	1.0	20.30	16.15	2.91	2.95	1.06	1.01
	29	82	84	1.5	1.0	10.50	10.50	3.00	3.01	0.80	0.78
	29.1	122	122	1.4	1.0	8.67	8.67	1.86	1.88	0.87	0.87
	29.3	16	15	1.6	1.0	6.15	6.15	2.18	2.19	0.58	0.59
	51	64	64	1.1	1.0	5.84	5.84	1.10	1.11	1.11	1.11
5.2	1	28	28	1.2	1.0	6.37	6.37	2.64	2.65	0.58	0.59
	10.1	13	13	1.4	1.0	9.40	9.40	3.18	3.08	0.82	0.86
	19.1	192	197	1.8	1.0	35.10	35.10	3.91	3.90	1.18	1.15
	51	168	169	1.5	1.0	14.80	14.08	3.01	3.05	0.96	0.92
5.3	51	17	17	1.2	1.0	10.35	10.35	2.82	2.84	0.74	0.74

Table 14.19 Drillhole Au Statistics for Uncomposited (Raw) and Composited (Comp.) Sample Data – Listwanite

FAULTZON	No. Samples		Max. Length (m)		Max. Grade (Au ppm)		Mean Grade (Au ppm)		CV (Au)	
	Raw	Comp.	Raw	Comp.	Raw	Comp.	Raw	Comp.	Raw	Comp.
1	203	208	1.6	1.0	8.01	8.01	0.86	0.87	1.70	1.65
2	229	234	1.5	1.0	15.10	15.10	1.12	1.13	1.87	1.83
2.1	130	132	3.0	1.0	4.99	4.99	0.47	0.45	2.14	2.17
6	944	959	1.7	1.0	25.40	24.80	1.08	1.09	2.56	2.47
8.1	142	150	1.9	1.0	11.40	8.31	0.86	0.85	1.85	1.72
8.2	253	256	1.6	1.0	14.55	14.55	0.87	0.88	1.78	1.73
8.3	198	202	1.7	1.0	11.10	9.13	0.66	0.64	2.20	2.17
9	480	491	2.5	1.0	16.55	16.55	0.59	0.61	2.33	2.36
10.1	318	330	1.8	1.0	19.30	17.79	1.46	1.45	1.64	1.55
10.2	198	204	1.7	1.0	12.25	12.25	1.42	1.44	1.68	1.65
13.1	279	289	2.3	1.0	8.84	7.89	0.70	0.70	1.90	1.79
13.2	58	59	1.5	1.0	6.41	6.41	1.56	1.55	1.10	1.09
14	66	67	1.6	1.0	8.60	8.60	0.87	0.83	2.05	2.08
18	8	11	2.7	1.0	0.05	0.05	0.02	0.02	1.25	1.08
19.1	117	117	1.4	1.0	5.73	5.73	0.51	0.51	1.74	1.72
19.2	92	93	1.6	1.0	4.81	4.81	0.21	0.22	3.23	3.20
25	459	475	2.0	1.0	16.95	16.95	1.03	1.03	1.64	1.61
28	783	802	2.0	1.0	25.20	25.20	1.69	1.69	1.82	1.74
29	281	283	1.8	1.0	14.35	14.35	1.02	1.01	1.67	1.62
29.1	68	69	1.7	1.0	2.63	2.63	0.27	0.25	1.85	1.85
29.2	59	62	3.0	1.0	9.88	5.61	0.96	0.96	1.79	1.46
29.3	25	25	1.6	1.0	2.24	2.24	0.28	0.30	1.73	1.65
38	88	89	1.6	1.0	7.40	7.40	0.49	0.50	2.48	2.31
48	27	29	1.5	1.0	0.60	0.54	0.12	0.12	1.43	1.38
51	3,028	3,079	2.0	1.0	17.15	17.15	0.58	0.59	1.75	1.75

14.3.11 Ardich Resource Model Estimation

A cell model with 10 m x 10 m x 5 m parent cells was constructed to cover the entire Ardich deposit. The cell model parameters are shown in Table 14.20.

Table 14.20 Ardich Cell Model Prototype Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
X (East)	461,800	466,000	4,200	10	420
Y (North)	4,365,500	4,368,500	3,000	10	300
Z (RL)	600	1,650	1,050	5	210

Sub-celling to 2.5 m x 2.5 m x 1.25 m was permitted to honour interpreted boundaries. Further sub-celling to a minimum of 0.25 m in the Z (RL) direction was permitted at the topographic surface.

The model was not rotated.

This model was populated with the same domain fields as the sample files; these being FAULTZON, MODLITH, LITHNUM, and for some domains the field 'SUBDOM', were used to generate the estimation flagging domains required for the subsequent grade estimation processes.

The combination of the interpretated lithological and fault block attributes resulted in the creation of 218 unique domains.

The concatenated field 'AUESTFLG' was used to define the estimation domains for the Au and other elements estimation, and 'SESTFLG' for the sulfur estimation. The unique estimation domain values were compiled using the attributes FAULTZON, LITHNUM, and SUBDOM (where relevant).

The AUESFLG and SESTFLG identifiers were calculated using the following formula:

AUESTFLG / SESTFLG = (FAULTZON x 10,000) + (LITHNUM x 100) + (SUBDOM)

A suite of 13 elements were estimated using ordinary kriging (OK). Au(cut) plus Au(uncut), Ag, As, C, Ca, Cu, Fe, Mg, Mn, Pb, Sb, and Zn (referred to as Au and other elements) were estimated using the estimation, search, and variogram parameter inputs developed for Au. In addition, a nearest neighbour Au(cut) estimation was undertaken, to be used for validation. A separate OK grade estimation for sulfur (S) was undertaken using a unique set of estimation, search, and variogram parameters developed specifically for that element. Sulfide sulfur (SS) was not estimated; rather, it was calculated using linear regressions based on the estimated sulfur grades.

Several methods were used to honour the observed styles of mineralisation and local variability. These included a dynamic search ellipse orientation for the mineralised domains, and the use of sub-domaining to implement of hard / soft boundaries proximal to contact mineralised zones (SUBDOM) and forced sub-celling with sub-cell estimation at lower contacts in Jasperoids.

14.3.11.1 Search Parameters

The search parameters used for the estimation were developed based on several criteria relating to the nature of the mineralisation and the composition of estimation domains.

Each of the 218 individual domains across the full spectrum of MODLITH and FAULTZON combinations were able to be categorised into two broad domain types. These were 'thick' and 'thin' domains. Thick domains are comprised mainly of whole parent cells and long drillhole intercepts. Thin domains comprise predominately sub-cells and have shorter drillhole intercepts.

For both the thick and thin domains the nature of the mineralisation is the same. The grades are more consistent laterally and change much quicker in the vertical direction. Lateral change in grade can still be significant, however, and the local grade conditions should be emphasised in the estimation. This was achieved by selecting search parameters that directed the estimation to select samples close to the cell and reduce the number of samples used from more distant locations.

The singular factor differentiating the two domain types (thick and thin) was the length of the drillhole intercept and the number of samples present in each drillhole. This factor was of primary concern for appropriately selecting the minimum and maximum sample attributes, to ensure that the goal of producing an estimation that represented the local grade conditions was met.

These concepts regarding the nature of the mineralisation and the sample data led to the following concepts being incorporated into the search volume criteria:

- The orientation of the search volume will be rotated to the local conditions by the dynamic anisotropy function. However, the Z axis search distance was to be reduced compared to the X and Y axes to honour both the variography and the visual observations of grade continuity being lower in the Z direction.
- A minimum of two drillholes are required for a cell to estimate in the first, second and third estimation passes. A single drillhole may be used for the fourth estimation pass.
- No more than six samples from any individual drillhole to be used in the thick domains and no more than three samples from any individual drillhole to be used in the thin domains. Due to the angle of the drillholes, 6 m of drillhole generally represents 5 m vertical metres, or a full parent cell of the model. Therefore, capping the samples from any individual drillhole at six ensures that a full 5 m high parent cell is using approximately five vertical metres of drilling data.
- The maximum number of samples used to estimate a cell is 14 for the thick domains and 12 samples for the thin domains. The selection of these values, in conjunction with the maximum samples per hole values was designed to limit the number of distant holes / samples used in the estimation in preference to local grade conditions. This, while still ensuring that two, but preferably three holes were used to estimate most cells.
- No octant search restrictions were used.

The search parameters for grades estimation are shown in Table 14.21.

Table 14.21 Ardich Search Volume Criteria for Grades Estimation in All Domains

LITHOLOGY	DOMAIN TYPE	Search Ellipse Rotation			Search Volume		Search Distance (m)			No. of Samples		Max. Samples Per Hole
		Rotation 1 Axis 3	Rotation 2 Axis 1	Rotation 3 Axis 3	Search Pass	Volume Factor	X	Y	Z	Min.	Max.	
Jasperoid, Listwanite, Dolomite, Cataclasite, Hornfels, Ophiolite, & Silica Cap	THICK	DA	DA	DA	1	1	40	40	10	8	14	6
					2	2	80	80	20	8	14	6
					3	3	120	120	30	8	14	6
					4	3	120	120	30	4	14	6
Cataclasite (FAULTZON 2)	THICK	DA	DA	DA	1	1	40	40	10	8	12	6
					2	2	80	80	20	8	12	6
					3	3	120	120	30	8	12	6
					4	3	120	120	30	4	12	6
Diorite	THICK	045	0	0	1	1	40	40	10	8	14	6
					2	2	80	80	20	8	14	6
					3	3	120	120	30	8	14	6
					4	3	120	120	30	4	14	6
Jasperoid & Listwanite	THIN	DA	DA	DA	1	1	40	40	10	4	12	3
					2	2	80	80	20	4	12	3
					3	3	120	120	30	4	12	3
					4	3	120	120	30	3	14	3
Diorite & Ophiolite	THIN	045	0	0	1	1	40	40	10	4	12	3
					2	2	80	80	20	4	12	3
					3	3	120	120	30	4	12	3
					4	3	120	120	30	3	14	3

14.3.11.2 Variography

Variograms were generated for Au and S using the 1 m composites (Table 14.22 and Table 14.23).

Due to the structural complexity of the deposit, many domains are relatively small and are often offset vertically from the same lithological unit in neighbouring domains. This often resulted in insufficient samples to generate a variogram at the individual domain level and the dislocation from domain-to-domain also hampered the development of a deposit-wide variogram model for each lithology.

FAULTZON 2.1 (SC1) was selected as a representative domain within the deposit that contained sufficient data in each of the main mineralised lithology units to generate robust variograms. Variograms were developed for the jasperoid, listwanite, and dolomite using the FAULTZON 2.1 (SC1) data, and these were applied across the deposit.

The use of the dynamic anisotropy function allowed the search volume orientation as well as the variogram to be rotated to the local conditions in the other domains.

The grade continuity identified in the gold variograms was consistent with the visual observations of the mineralisation with grade continuity highest perpendicular to the lithology surfaces and lowest in the Z direction. The sulfur variograms showed more grade continuity than the gold variograms with ranges often twice as long as the gold variograms. This higher continuity observed in the sulfur variogram data is supported by visual observations of the surface data.

Table 14.22 Gold Variogram Parameters

MODLITH	Rotation Angles			Nugget	Structure	Structure Variance	Range		
	Axis 3 (Z)	Axis 1 (X)	Axis 3 (Z)				Axis 1	Axis 2	Axis 3
Jasperoid and Diorite	–110	20	–25	0.38	1	0.62	85	106	10
Listwanite	–105	5	–160	0.06	1	0.94	150	60	10
All Other	–100	5	–5	0.13	1	0.87	123	84	12

Table 14.23 Sulfur Variogram Parameters

MODLITH	Rotation Angles			Nugget	Structure	Structure Variance	Range		
	Axis 3 (Z)	Axis 1 (X)	Axis 3 (Z)				Axis 1	Axis 2	Axis 3
Jasperoid	–105	10	–50	0.1	1	0.90	195	140	20
Listwanite	90	5	175	0.15	1	0.85	134	78	20
Dolomite	80	90	180	0.18	1	0.24	66	14	54
					2	0.58	179	23	89
Cataclasite	–140	170	0	0.04	1	0.96	153	96	12

14.3.12 Ardich Density Model

The density estimation was undertaken within a similar framework to the grade estimations, likewise, using ordinary kriging. The sub-domaining routines employed for the grade estimations were not used for the density estimation.

The model field 'SGESTFLG' was created using the LITHNUM and FAULTZON fields to control the estimation domaining.

Due to the lower number of samples present in the density dataset compared to the grade datasets (12,419 density samples vs. 72,432 Au assays), specific density estimation search parameters were developed. Density variogram models were also developed utilising samples from FAULTZONs SC1, C4, and EC2.

The density variogram models showed a longer range of continuity than was shown in the gold or sulfur variograms. The density variograms were different to the grade variogram models, however over-all the density variogram models could also be described as having greatest continuity parallel to the lithology contacts in the same way as the gold and sulfur variograms do. Therefore, the TRDIPDIR and TRDIP values estimated into the model and used for the grade estimation were also used for the density estimation.

Minimum and maximum sample numbers were developed based on the density dataset. In a similar manner to the grade search parameters a 'thick' and 'thin' domain type designation was applied depending on the domain.

There are a few notable differences between the density and grade estimations. First is removal of the fourth search volume, with the density estimation utilising only three search volumes. Secondly, cells in the 'thick' estimation domains can estimate from a single drillhole in the third search volume, whilst in the 'thin' domain's cells can estimate from a single drillhole in all search volumes. These conditions reflect the lower number of density samples and the distribution of these samples within these domains.

The search parameters for grades estimation are shown in Table 14.24.

Table 14.24 Ardich Search Volume Criteria for Density Estimation in All Domains

LITHOLOGY	DOMAIN TYPE	Search Ellipse Rotation			Search Volume		Search Distance (m)			No. of Samples		Max. Samples Per Hole
		Rotation 1 Axis 3	Rotation 2 Axis 1	Rotation 3 Axis 3	Search Pass	Volume Factor	X	Y	Z	Min.	Max.	
Jasperoid, Dolomite, Cataclasite, Ophiolite, & Listwanite	THICK	DA	DA	DA	1	-	40	40	10	4	8	3
					2	2	80	80	20	4	8	3
					3	5	200	200	50	3	8	3
Hornfels & Silica Cap	THICK	DA	DA	DA	1	-	40	40	10	4	8	3
					2	2	80	80	20	4	8	3
					3	5	200	200	50	3	8	3
Diorite	THICK	045	0	0	1	-	40	40	10	2	6	3
					2	2	80	80	20	2	6	3
					3	5	200	200	50	2	6	3
Jasperoid & Listwanite	THIN	DA	DA	DA	1	-	40	40	10	2	6	3
					2	2	80	80	20	2	6	3
					3	5	200	200	50	2	6	3
Diorite, Hornfels, & Silica Cap	THIN	045	0	0	1	-	40	40	10	4	8	3
					2	2	80	80	20	4	8	3
					3	5	200	200	50	3	8	3

14.3.13 Ardich Resource Classification

The grade estimates classifications were stored in the model field 'RESCAT'.

The values used to indicate classification in the RESCAT field are as follows:

- 1 = Measured
- 2 = Indicated
- 3 = Inferred
- 0 = Unclassified

RESCAT was considered in detail on the basis of the unique properties of each lithological unit within each domain.

Emphasis was placed on those lithologies that host the most-significant mineralisation in each domain (generally jasperoid and listwanite).

Unique polygons were digitised in plan view around areas considered to have higher confidence interpretations and estimates within each domain.

Because of the high degree of complication in the domain interpretations, and the variable nature of the mineralisation, the classification method was not based on any single attribute or parameter (such as drillhole spacing, search volume, distance to nearest sample, or number of samples). Rather, a more interactive and holistic approach to the classification was adopted.

The key factors in assessing classification were observations related to the geological and grade continuity of each domain. This, coupled with an assessment of the performance of the estimation, formed the primary basis for the classification. Additional information such as the drill spacing and the search volume were also considered during the classification process, however they were not used categorically or rigidly, and were not the primary drivers for defining the RESCAT shapes.

For example, a domain with well-understood geology and highly consistent mineralisation, but with relatively wide drill spacing, may receive a similar or a better RESCAT than a domain where the drill spacing is closer, but confidence in the geological interpretation is lower and the mineralisation is less consistent.

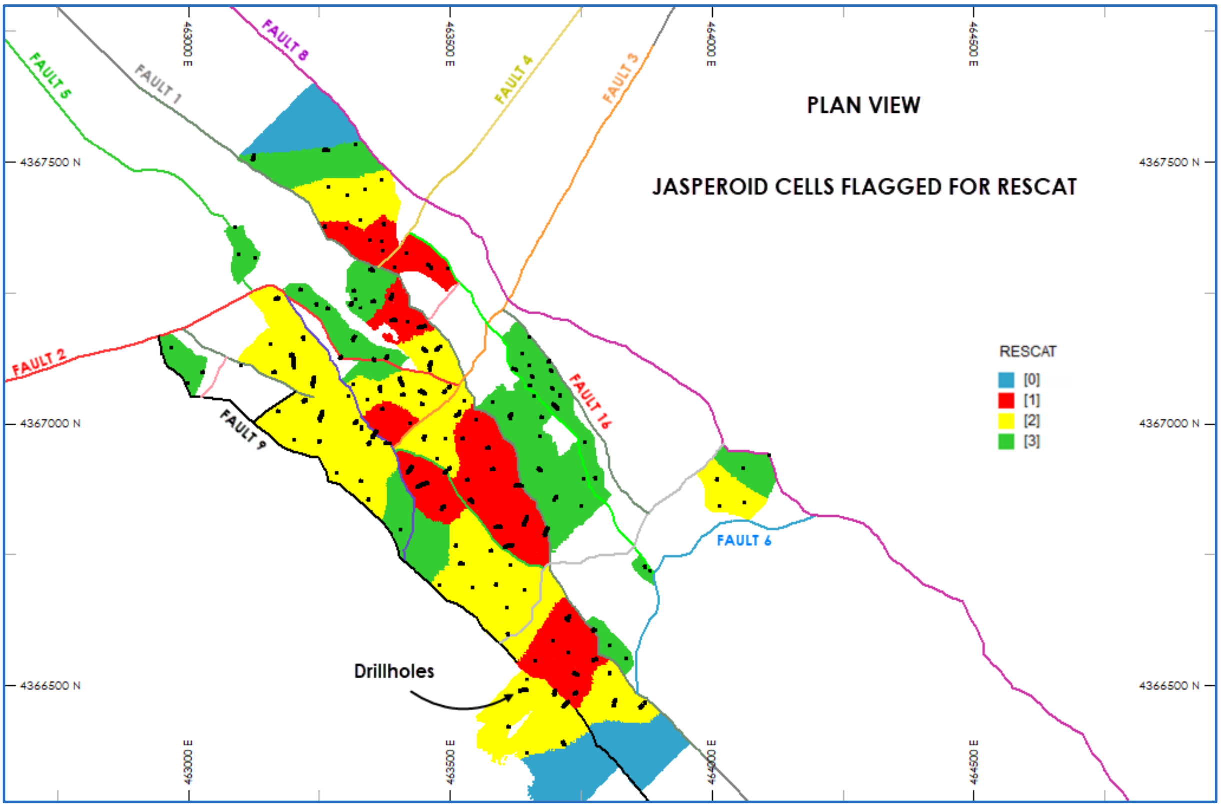
The most prominent role that the distribution of the drilling played in the classification was related to the overall number of drillholes informing the interpretation in a domain, rather than the drillhole spacing specifically.

A detailed log of the decision making process for each RESCAT polygon was developed and retained for future reference.

Figure 14.15, Figure 14.16, and Figure 14.17 show the model coloured by RESCAT for the jasperoid, listwanite, and upper dolomite lithologies.

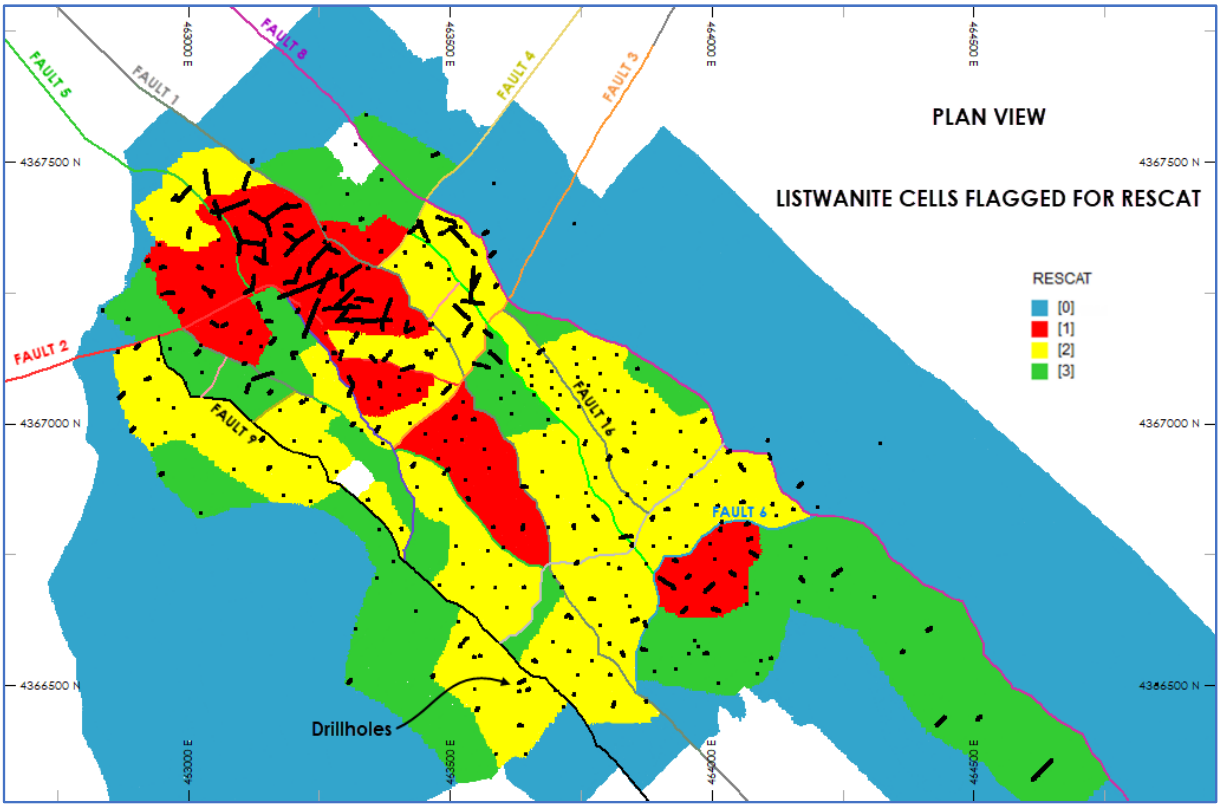
For unmineralised domains, such as the ophiolite, lower dolomite, and lower cataclasite, default RESCAT values were applied.

Figure 14.15 Plan View of Jasperoid Resource Classification



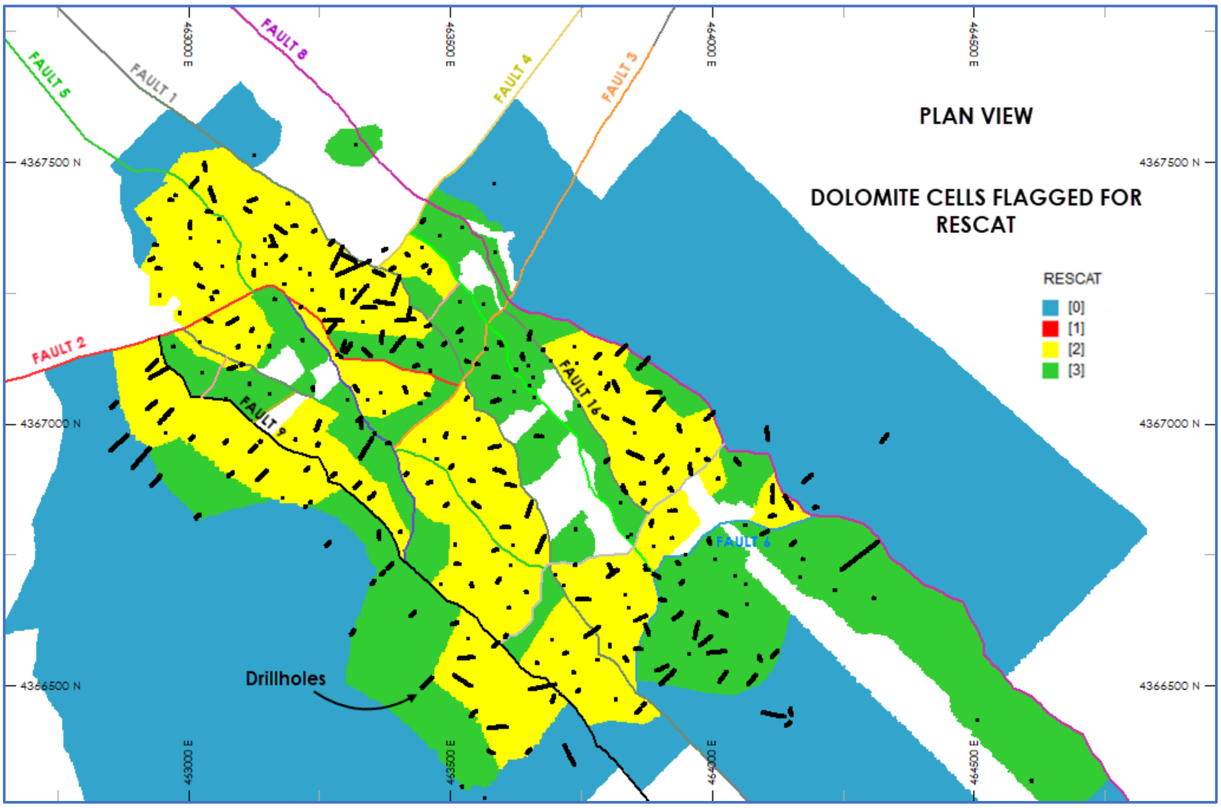
OreWin, 2021

Figure 14.16 Plan View of Listwanite Resource Classification



OreWin, 2021

Figure 14.17 Plan View of Upper Dolomite Resource Classification



OreWin, 2021

14.3.14 Ardich Model Validation

Model validation was approached in several ways:

- The estimated Au grades in the model were compared to the drillhole grades by visual inspection in plan views, sectional views, and in 3D. In general, the model and composite grades visually compared well and the estimates were considered to have honoured the interpreted mineralisation styles.
- The cell model was checked for global bias by comparing the Au and S statistics of the model estimates compared to the input sample file and a nearest neighbour (NN) estimation. The NN estimation produces a theoretically unbiased (de-clustered) estimate of the mean, offering an alternative metric to sample grade for comparing the estimation. The cells used to calculate these mean values were restricted to those cells that were classified as Measured, Indicated, or Inferred, thus removing any poorly estimated or default grade cells on the periphery of the domains. The means comparison shows that for the jasperoid and listwanite there is good correlation between the OK and NN estimations. Correlation is lower for the dolomite and cataclasite, which is not considered remarkable given the more-variable nature of those domains.

- Local trends in the grade estimates (also known as drift analysis) were assessed by plotting the mean values from the NN estimate versus the kriged results for Indicated model cells in east–west, north–south and vertical directions (swath plots). There is good correlation between the OK and NN estimations in all orientations. However, the correlation diverges at the model edges, especially in the north-eastern corner where data density becomes reduced and a single drillhole or sample can have a disproportionate effect on the NN estimates.

14.3.15 Ardich Change of Support

Mining has not occurred at the Ardich project area and therefore no production data is available. A 5 m mining bench is anticipated, with 5 m blastholes likely to be used. Grade estimation at Ardich is based on 1 m assay composites and interpreted geological (structural and lithological) boundaries to estimate resource model tonnes and grade using ordinary kriging.

14.3.16 Ardich Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,750/oz for gold with the parameters summarised in Table 14.25.

Table 14.25 Summary of Key Parameters Used in Conceptual Pit Shell at Ardich

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	40.0	73.0
POX Gold Recovery	%	82.9	82.9
Mining Cost per tonne mined	$/t	1.82	1.82
Process Costs Heap Leach	$/t	10.68	10.68
Process Costs POX	$/t	36.25	36.25
Site Support and G&A	$Mpa	15	15
Internal Au Cut-off – Heap Leach	g/t	0.25	0.46
Internal Au Cut-off – POX	$/t NSR	0.77	0.77
Internal Au Cut-off – Cu Conc.	$/t NSR	9.05 + 36.25 x Pyrite Mass Pull	
Royalty	%	2.0	2.0

14.3.17 Ardich Mineral Resource Tabulation

Ardich Mineral Resources have been tabulated by resource classification and oxidation state in Table 14.30. Mineral Resources are presented on a 100% basis.

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

The overall tonnage and grade estimate have increased for oxide and sulfide material from the previously-reported estimate in 2019. This change is predominantly due to the change in the conceptual model, supported by the additional drilling obtained since 2020.

Resource pit shells were generated by OreWin using a metal price assumption of $1,750/oz gold. Gold mineralisation modelled at Ardich is primarily oxidised with a smaller portion of sulfur mineralisation having estimated total sulfur grades >2%. Low-sulfur (LS) oxide is defined as material with <1% total sulfur. High-sulfur (HS) oxide is material with total sulfur >1% and <2%. Sulfide material has ⩾2% total sulfur. The Mineral Resources are shown in Table 14.30.

Internal cut-off grades for oxide material range from 0.30–0.55 g/t Au. Sulfide is material with >2% total sulfur above a 1.1 g/t Au cut-off.

14.4 Bayramdere Deposit

The Bayramdere deposit is located approximately 6.3 km east of the Çöpler mine and 5 km south-east of İliç. Bayramdere is within the Kartaltepe Mining Licence 7083. This licence is an operational licence and is 50% SSR-held.

Soil samples have been collected across the prospect on a 100 m x 100 m grid. Soil copper and gold anomalies are identified as coincident with each other, but the copper anomaly covers a larger area.

The Bayramdere mineralisation has an overall strike length of approximately 300 m. Mineralisation is localised within three stacked, shallow-dipping lodes that are very close to the surface, varying in depth 30–40 m below topography. Mineralisation appears to be open to the east and south.

The mineralisation has formed at the contacts of limestone and ophiolite lithologies with mineralisation replacing limestone along the contacts. The limestone to ophiolite contacts are low-angle thrusts, with limestone typically being trapped as wedges of material within a dominantly ophiolite stratigraphy. Mineralisation occurs within iron-rich gossan horizons.

Although a small deposit, Bayramdere is relatively high-grade and can support a high-stripping ratio to access mineralisation.

Small-scale open pit iron ore mining has occurred historically at Bayramdere. Iron mineralisation can be associated with gold mineralisation.

14.4.1 Bayramdere Domains

The geological interpretation was represented in the geological model through the creation of mineralised domains based on the continuity of the geology and mineralisation identified specific to each deposit and mineralised zone within the deposit. Separate domains were created for gold, silver, copper, and sulfur. In the creation of mineralised domains, a minimum mining width of 2.5 m was used based on anticipated open pit mining methods.

14.4.2 Bayramdere Geological Model

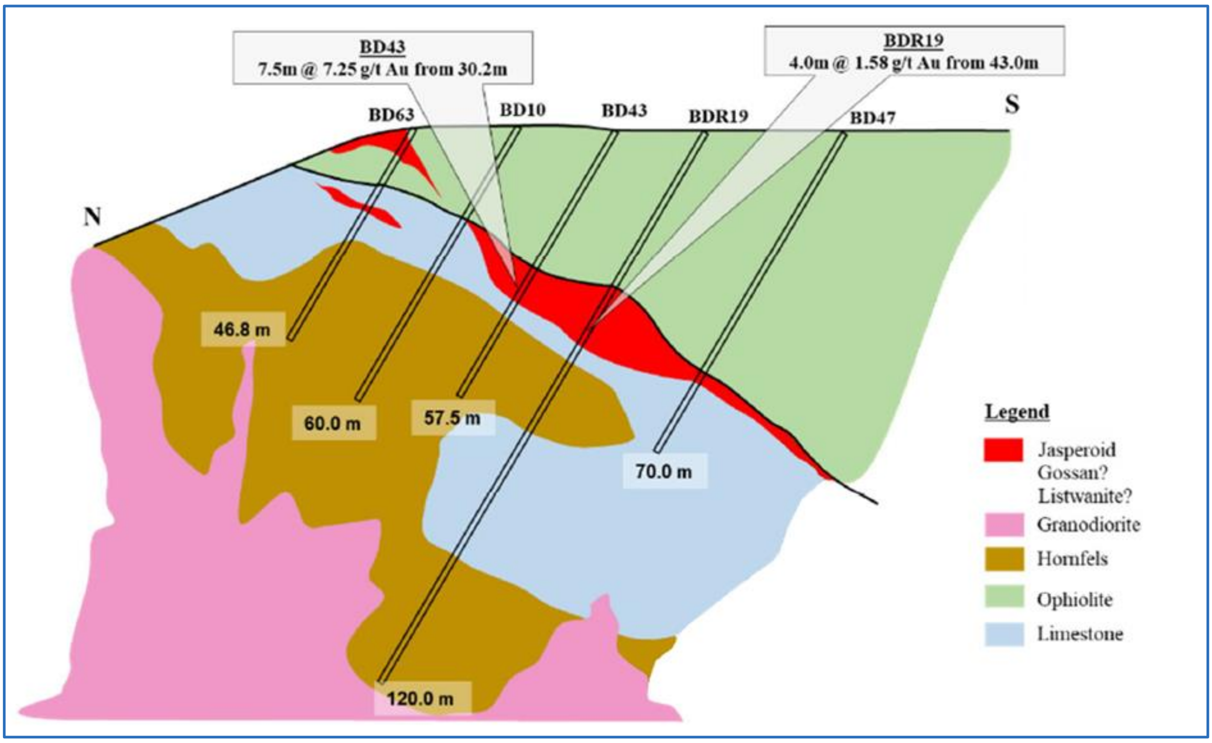
The Bayramdere deposit is a structurally controlled gold±minor copper±minor silver deposit displaying both epithermal and replacement mineralisation styles. At this stage of exploration, the deposit is dominantly represented by near-surface oxide mineralisation to a depth of up to 180 m below surface. Mineralisation is primarily associated with jasperoid and iron-rich gossan. Secondary pyrite is a commonly visible component within the jasperoids.

At depth, mineralisation transitions below the base of complete oxidation to disseminated pyrite, vein sulfides, and massive sulfide horizons generally occurring within shear zones, along shallow thrusts and diorite sill and dyke margins. The extent of sulfide mineralisation has not been tested.

As with the other Çöpler district deposits, Bayramdere is considered to be the result of a mineralised intrusion generating suitable conditions for mineralisation to be localised into a favourable geological setting of ophiolite, limestone, and hornfels lithologies (see Figure 14.18). A complex system of faults and thrusts have allowed mineralised fluids and diorite dykes and sills associated with the epithermal system to permeate into the stratigraphy.

Like the Çakmaktepe deposit, Bayramdere is associated with flat thrust structures. Key to each structurally associated style of mineralisation is the juxtaposition of ophiolites against limestone + hornfels to create suitable geochemical conditions for gold and other metals deposition. Ophiolite is not associated with mineralisation at Çöpler, this association at present is considered to be unique to Bayramdere and Çakmaktepe.

Figure 14.18 Bayramdere Geology Schematic Section



Anagold, 2017

14.4.3 Bayramdere Data Summary

The Bayramdere deposit has been drilled on 25 m lines with 20–25 m spaced holes on each line.

A total of 120 resource definition drillholes have been drilled at Bayramdere for a total length of 10,734.2 m, inclusive of metallurgical holes. The assay database includes 8,283 sample intervals for a total assayed length of 10,483.4 m.

When categorised according to type of drilling (excluding geotechnical and metallurgical drillholes), 30% are RC samples, 65% DD core samples, and 6% are a combination of RC and DD core.

Drilling has been completed on drill grids aligned at right angles to mineralisation trends or lithology dip and strike. Several areas contain scissor holes that test mineralisation at 180° from each other.

14.4.4 Bayramdere Drillhole Compositing

Sample compositing has not been applied. The predominant sample length is 1.0 m (52%), followed by 2 m as the next most prevalent length (17%).

14.4.5 Bayramdere Top Cutting

High-grade top cuts were applied after selecting appropriate limits based on cumulative frequency plots and value grade curves of the upper portion of the sample population.

14.4.6 Bayramdere Cell Model

The Bayramdere cell model parameters are shown in Table 14.26.

Sub-celling was permitted to 2 m x 2 m x 1 m to better honour the domain boundaries.

Table 14.26 Bayramdere Cell Model Prototype Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	Number of Cells
East	466,000	466,600	600	10	60
North	4,363,800	4,364,100	300	10	30
RL	1,250	1,420	170	5	34

14.4.7 Bayramdere Estimation Method

Estimation was limited to the interpreted domains, with each domain informed only by samples contained within that domain. Outside the mineralised domains a 'mineralised waste' estimate was completed.

Mineralisation domains were also developed for silver, copper, and sulfur.

Lithological domains were used for estimates outside of the mineralisation domains.

Ordinary kriging was used to estimate Au, Ag, and Cu into parent cells. Variography was completed to inform estimation.

14.4.8 Bayramdere Density Model

Density has been assigned as a default for each of the mineralisation and lithological domains (see Table 14.27 and Table 14.28 respectively). The assigned densities reflect the arithmetic average of the domain-relevant data taken from DD core samples.

Table 14.27 Bayramdere Density Values for Mineralisation Domains

Domain	Density (t/m^3)
mz100	2.69
mz101	2.49
mz200	2.55
mz201	2.79
mz300	2.29
mz301	2.29
mz400	2.49
mz500	2.42
mz600	2.49
mz700	2.49

Table 14.28 Bayramdere Density Values for Lithology Domains

Domain	Weathering State	Density (t/m^3)
Gossan	Weathered	2.50
Diorite		2.44
Limestone		2.54
Ophiolite		2.36
Gossan	Fresh	2.50
Diorite		2.44
Limestone		2.54
Ophiolite		2.36
Overburden	All	1.40

14.4.9 Bayramdere Resource Classification

Grade estimates were classified using the following Anagold guidelines:

- Indicated Mineral Resource should be quantified within relative ±15% with 90% confidence on an annual basis, and
- Measured Mineral Resources should be known within ±15% with 90% confidence on a quarterly basis.

Drillhole spacing for support of classification of Inferred Mineral Resources was required to be 50 m x 25 m spacing. For Indicated Mineral Resource classification, the drillhole spacing requirement was reduced to 25 m x 25 m spacing. Appropriate drillhole pattern spacing selection was based on the understanding of the nature of the mineralisation being structurally controlled, mineral continuity, and assessment of data quality.

The drillhole spacing at Bayramdere is considered sufficient to support grade continuity, geological continuity, depth, and lateral extents of mineralisation.

No Bayramdere estimates were classified in the Measured category.

Mineral Resources were tabulated using multiple cut-off grades due to variable recoveries and based on gold price only. Cut-off grades vary from 0.35–0.50 g/t Au and are calculated based on the equation:

$$Xc = Po / (r * (V-R))$$

where Xc = Cut-off Grade (g/t), Po = processing cost of ore (USD/tonne of ore), r = recovery, V = gold sell price ($/g), R = refining costs ($/g).

Mineral Resources are reported inclusive of Mineral Reserves.

14.4.10 Bayramdere Validation

Bayramdere grade estimates were validated against alternate interpolation methods. Estimated grades were compared to an ID2 model to check for global bias. Swath plots were used to check for a local bias. The estimated Au grades in the model were compared to the composite grades by visual inspection in plan views and cross-sections. Composite samples were queried by domain to confirm appropriate sample flagging.

14.4.11 Bayramdere Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,400/oz for gold and $19/oz for silver, with the parameters summarised in Table 14.29. These parameters have not been updated since 2017, primarily because no further work has been completed at Bayramdere since that time.

Table 14.29 Summary of Key Parameters Used in Conceptual Pit Shell at Bayramdere

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	75.0	75.0
Mining Cost per tonne mined	$/t	1.75	1.75
Process Costs Heap Leach	$/t	9.99	9.99
Site Support per tonne processed	$/t	3.19	3.19
Internal Au Cut-off – Heap Leach	g/t	0.35	0.50
Royalty	%	2.0	2.0

14.5 Mineral Resources Statement

Mineral Resources and Mineral Reserves in the CDMP21TR meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). Mineral Resources statement shown in Table 14.30 were estimated by Sharron Sylvester BSc (Geology), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology. Mineral Resources are presented on a project basis and have an effective date of 31 December 2021.

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

Mineral Resources are reported exclusive of Mineral Reserves and have been summarised by project, resource classification, and oxidation state in Table 14.30.

Table 14.31 shows the cut-off values, metallurgical recoveries, and SSR ownership percentage associated with the Mineral Resources.

Table 14.30 Summary of CDMP21TR Mineral Resources Estimates Exclusive of Mineral Reserves (as at 31 December 2021) Based on $1,750/oz Gold Price

Mineral Resource Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine – Oxide							
Measured	81	1.39	4.67	0.16	4	12	281
Indicated	27,173	0.84	2.30	0.16	737	2,012	97,057
Measured + Indicated	27,254	0.84	2.31	0.16	740	2,024	97,339
Inferred	35,021	0.90	6.87	0.13	1,016	7,741	97,941
Çöpler Mine – Sulfide							
Measured	151	0.83	3.72	0.18	4	18	590
Indicated	47,084	1.06	3.66	0.19	1,608	5,535	198,365
Measured + Indicated	47,235	1.06	3.66	0.19	1,612	5,553	198,955
Inferred	49,798	1.24	13.60	0.17	1,982	21,773	181,890
Çakmaktepe – Oxide							
Measured	–	–	–	–	–	–	–
Indicated	3,341	1.55	8.33	–	167	894	–
Measured + Indicated	3,341	1.55	8.33	–	167	894	–
Inferred	1,205	0.85	4.04	–	33	157	–
Ardich – Oxide							
Measured	2,840	1.67	3.99	0.02	153	364	1,031
Indicated	9,794	1.01	2.74	0.00	317	861	410
Measured + Indicated	12,634	1.16	3.02	0.01	469	1,226	1,442
Inferred	13,896	1.27	3.47	0.02	570	1,550	5,181
Ardich – Sulfide (Incl. sulfide and sulfide-with-Cu)							
Measured	234	5.76	8.25	0.04	43	62	215
Indicated	1,410	2.07	3.80	0.01	94	172	403
Measured + Indicated	1,645	2.59	4.44	0.02	137	235	619
Inferred	3,226	2.64	4.53	0.01	274	470	576
Bayramdere – Oxide							
Measured	–	–	–	–	–	–	–
Indicated	145	2.34	20.82	–	11	97	–
Measured + Indicated	145	2.34	20.82	–	11	97	–
Inferred	8	2.17	19.95	–	1	5	–
CDMP21 Mineral Resources – Oxide Subtotal							
Measured	2,920	1.67	4.01	0.02	156	376	1,313
Indicated	40,454	0.95	2.97	0.11	1,231	3,865	97,467
Measured + Indicated	43,374	0.99	3.04	0.10	1,387	4,241	98,780
Inferred	50,130	1.00	5.86	0.09	1,619	9,453	103,122
CDMP21 Mineral Resources – Sulfide Subtotal							
Measured	386	3.82	6.47	0.09	47	80	805
Indicated	48,494	1.09	3.66	0.19	1,702	5,707	199,265
Measured + Indicated	48,880	1.11	3.68	0.19	1,749	5,787	200,071
Inferred	53,024	1.32	13.05	0.16	2,256	22,243	182,465
CDMP21 MINERAL RESOURCES – OVERALL TOTAL (Exclusive of Mineral Reserves)							
Measured	3,306	1.92	4.30	0.03	204	457	2,118
Indicated	88,948	1.03	3.35	0.15	2,933	9,572	296,733
Measured + Indicated	92,254	1.06	3.38	0.15	3,136	10,029	298,851
Inferred	103,154	1.17	9.56	0.13	3,875	31,695	285,587

1. Mineral Resources are reported based on 31 December 2021 topography surface.
2. Mineral Resources are reported exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis. Çöpler Mineral Resources are located on ground held 80% by SSR, Çakmaktepe and Bayramdere Mineral Resources are located on ground held 50% by SSR, and approximately 96% of Ardich Mineral Resources are located on ground held 80% by SSR, with the remainder located on ground 50% held by SSR.
4. Oxide definitions: At Çöpler: oxide is defined as material <2% total sulfur and sulfide material is ⩾2% total sulfur. At Ardich and Çakmaktepe, oxide is comprised of low-sulfur (LS) oxide (<1% total sulfur) and high-sulfur oxide (⩾1% and <2% total sulfur). At Bayramdere: oxide is defined as material <2% total sulfur.
5. Sulfide definitions: At Ardich, sulfide is comprised of standard sulfide material (⩾2% total sulfur) and sulfide-with-Cu material (sulfide with Cu>0.10%).
6. At Çöpler and Ardich: sulfide cut-off uses an NSR value in $/t based on gold price $1,750/oz, silver price $22.00/oz Ag and copper price $3.95/lb with allowances for payability, deductions, transport, and royalties.
7. All Mineral Resources in the CDMP21TR were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells ($1,400/oz for gold and $19/oz for silver for Bayramdere, and $1,750/oz for gold, $22/oz for silver for all other projects).
8. The point of reference for Mineral Resources is the point of feed into the processing facility.
9. Tonnage is metric tonnes, ounces represent troy ounces, and g/t represents grams per metric tonne.
10. Totals may vary due to rounding.

Table 14.31 Summary of Cut-off Values, Metallurgical Recoveries, and SSR Ownership of CDMP21TR Mineral Resources Estimates Exclusive of Mineral Reserve (as at 31 December 2021) Based on Gold Price $1,750/oz, Silver Price $22.00/oz Ag and Copper Price $3.95/lb

Mineral Resource Classification	Tonnage (kt)	Grades			Cut-off Value/s	Metallurgical Recovery (%)	SSR Ownership (%)
		Au (g/t)	Ag (g/t)	Cu (%)			
Çöpler Mine – Oxide							
Measured	81	1.39	4.67	0.16	0.19–0.24 g/t Au	62.3–78.4	80
Indicated	27,173	0.84	2.30	0.16			
Measured + Indicated	27,254	0.84	2.31	0.16			
Inferred	35,021	0.90	6.87	0.13			
Çöpler Mine – Sulfide							
Measured	151	0.83	3.72	0.18	$34.88/t NSR or >0.10% Cu and $7.68/t NSR	Au 55–91 Ag 10–45 Cu 84–98	80
Indicated	47,084	1.06	3.66	0.19			
Measured + Indicated	47,235	1.06	3.66	0.19			
Inferred	49,798	1.24	13.60	0.17			
Çakmaktepe – Oxide							
Measured	–	–	–	–	0.36–0.76 g/t Au	38.0–80.0	50
Indicated	3,341	1.55	8.33	–			
Measured + Indicated	3,341	1.55	8.33	–			
Inferred	1,205	0.85	4.04	–			
Ardich – Oxide							
Measured	2,840	1.67	3.99	0.02	0.23–0.41 g/t Au	40.0–73.0	75
Indicated	9,794	1.01	2.74	0.00			76
Measured + Indicated	12,634	1.16	3.02	0.01			75
Inferred	13,896	1.27	3.47	0.02			65
Ardich – Sulfide (Incl. sulfide and sulfide-with-Cu)							
Measured	234	5.76	8.25	0.04	$36.25/t NSR or >0.10% Cu and $9.05/t NSR	Au 55–91 Ag 10–45 Cu 84–98	78
Indicated	1,410	2.07	3.80	0.01			71
Measured + Indicated	1,645	2.59	4.44	0.02			75
Inferred	3,226	2.64	4.53	0.01			71
Bayramdere – Oxide							
Measured	–	–	–	–	0.35–0.50 g/t Au	75	50
Indicated	145	2.34	20.82	–			
Measured + Indicated	145	2.34	20.82	–			
Inferred	8	2.17	19.95	–			
CDMP21 Mineral Resources – Oxide Subtotal							
Measured	2,920	1.67	4.01	0.02	As Above	As Above	75
Indicated	40,454	0.95	2.97	0.11			75
Measured + Indicated	43,374	0.99	3.04	0.10			75
Inferred	50,130	1.00	5.86	0.09			74
CDMP21 Mineral Resources – Sulfide Subtotal							
Measured	386	3.82	6.47	0.09	As Above	As Above	78
Indicated	48,494	1.09	3.66	0.19			80
Measured + Indicated	48,880	1.11	3.68	0.19			80
Inferred	53,024	1.32	13.05	0.16			79
CDMP21 MINERAL RESOURCES – OVERALL TOTAL (Exclusive of Mineral Reserves)							
Measured	3,306	1.92	4.30	0.03	As Above	As Above	76
Indicated	88,948	1.03	3.35	0.15			77
Measured + Indicated	92,254	1.06	3.38	0.15			77
Inferred	103,154	1.17	9.56	0.13			77

1. Mineral Resources are reported based on 31 December 2021 topography surface.
2. Mineral Resources are reported exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis. SSR Ownership is an average based on location of Mineral Resources (gold) relative to licenses: Çöpler and part of Ardich are on Anagold 80:20 ground on which SSR holds 80% rights, and Çakmaktepe, Bayramdere and the remainder of Ardich are on Kartaltepe 50:50 ground on which SSR holds 50% rights. Totals and Ardich ownership percentages are weighted averages.
4. Oxide definitions: At Çöpler: oxide is defined as material <2% total sulfur and sulfide material is ⩾2% total sulfur. At Ardich and Çakmaktepe, oxide is comprised of low-sulfur (LS) oxide (<1% total sulfur) and high-sulfur oxide (⩾1% and <2% total sulfur). At Bayramdere: oxide is defined as material <2% total sulfur.
5. Sulfide definitions: At Ardich, sulfide is comprised of standard sulfide material (⩾2% total sulfur) and sulfide-with-Cu material (sulfide with Cu>0.10%).
6. At Çöpler and Ardich: sulfide cut-off uses an NSR value in $/t based on gold price $1,750/oz, silver price $22.00/oz, and copper price $3.95/lb with allowances for payability, deductions, transport, and royalties.
7. All Mineral Resources in the CDMP21TR were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells ($1,400/oz for gold and $19/oz for silver for Bayramdere, and $1,750/oz for gold, $22/oz for silver for all other projects).
8. The point of reference for Mineral Resources is the point of feed into the processing facility.
9. Tonnage is metric tonnes and g/t represents grams per metric tonne.
10. Totals may vary due to rounding.

14.6 Comparison of 2021 Mineral Resource to Previous Mineral Resource

The 2021 Mineral Resource inventory is detailed in Table 14.30.

For comparison, a summary of the overall 2020 Mineral Resource inventory is shown in Table 14.32. This comparison compares Mineral Resources exclusive of Mineral Reserves.

Table 14.32 EOY 2020 Mineral Resources Summary – Exclusive of Mineral Reserves (as at 31 December 2020)

Mineral Resource Classification	EOY 2020 Mineral Resources Total (Exclusive of Mineral Reserves)						
	Tonnage (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Measured	5,707	1.70	0.20	0.00	312	36	482
Indicated	70,619	1.34	3.10	0.04	3,049	7,039	59,884
Measured + Indicated	**76,326**	**1.37**	**2.88**	**0.04**	**3,362**	**7,076**	**60,366**
Inferred	73,741	1.30	9.16	0.06	3,089	21,708	94,896

The factors contributing to the differences between the 2021 Mineral Resources and the previous Mineral Resources reported as at 31 December 2020 are as follows:

Ardich

- A significant proportion of the Measured plus Indicated (M+I) tonnage increase in 2021 is attributed to the updated Ardich model, which incorporates 194 additional drillholes (233 holes in 2020 vs. 427 holes in 2021), with the net result of significantly improving confidence in the interpretation, thereby increasing M+I inventory.
- There was no Mineral Reserve declared for Ardich in 2020. A maiden Mineral Reserve has been declared for Ardich in 2021, and this depletes the report of Mineral Resources exclusive of Mineral Reserves.

Other

- A drop in cut-off grade and the inclusion of copper extraction has resulted in a larger conceptual pit shell for Çöpler that contains additional volume above the cut-off.
- Review of metallurgical recoveries.
- Depletion through mining since 31 December 2020.

There has been an 30% increase in tonnage above the cut-off across all combined Mineral Resource categories, with a corresponding 9% increase in contained gold.

14.7 Subpart 1300 of US Regulation S–K Mining Property Disclosure Rules

The Mineral Resources reported in the CDMP21TR are suitable for reporting as Mineral Resources using Subpart 1300 of US Regulation S-K Mining Property Disclosure Rules (S K 1300).

The CDMP21TR includes an Initial Assessment Case that uses Measured, Indicated, and Inferred Mineral Resources to examine the impact of adding two new processing options to extract copper. The Initial Assessment Case is considered to be the same as an Initial Assessment under S–K 1300.

The Initial Assessment has been prepared to demonstrate economic potential of the Mineral Resources at the Çöpler deposit. The Initial Assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have modifying factors applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that this economic assessment will be realised.

15 MINERAL RESERVES ESTIMATES

15.1 Summary

Mineral Resources and Mineral Reserves in the CDMP21TR meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). Mineral Reserves were estimated by Bernard Peters BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director - Mining. Mineral Reserves are presented on a project basis and have an effective date of 31 December 2021.

Open pit mining at the Çöpler project is carried out by a mining contractor and managed by Anagold. The mining method is a conventional open pit method with drill and blast to facilitate extraction utilising excavators and trucks. Anagold currently operates a sulfide process plant and an oxide heap leach facility. Costs are based on actual operational costs and the Anagold budget assumptions.

The Mineral Reserves were developed based on mine planning work completed in 2021 and estimated based on an end-of-September 2021 topography surface. Çöpler oxide ore cut-off grades vary from 0.47–0.59 g/t Au. The Çöpler sulfide ore cut-off grade is 1.05 g/t Au. Çakmaktepe oxide cut-off grades vary from 0.52–0.69 g/t Au. There is no Çakmaktepe sulfide Mineral Reserve. Average oxide gold recoveries are 61% and average sulfide gold recoveries are 91%.

The cut-off grades for the Mineral Reserves estimates are based on a gold price of $1,350/oz. There are no credits for silver or copper in the cut-off grade calculations. Economic analysis has been carried out using long-term metal prices of $1,600/oz gold, $20.25/oz silver, and $3.05/lb copper, and average metal prices of $1,658/oz gold, $21.55/oz silver, and $2.95/lb copper.

15.2 Mineral Reserves Statement

Mineral Resources and Mineral Reserves in the CDMP21TR meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). The Mineral Reserves statement shown in Table 15.1 were estimated by Bernard Peters BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining. Mineral Reserves are presented on a project basis and have an effective date of 31 December 2021.

Table 15.2 shows the cut-off values, metallurgical recoveries, and SSR ownership percentage associated with the Mineral Reserves.

The CDMP21TR Reserve Case is at a feasibility level of study. The Mineral Resource estimates have been reported in the CDMP21TR. The Mineral Resource models include dilution. Measured Mineral Resources were converted to Proven Mineral Reserves, and Indicated Mineral Resources were converted to Probable Mineral Reserves. Inferred Mineral Resources were treated as waste and were not converted to Mineral Reserve. The Çöpler Mineral Reserve has been demonstrated to be viable by the CDMP21TR.

Table 15.1 Summary of CDMP21TR Mineral Reserves Estimates (as at 31 December 2021) Based on $1,350/oz Gold Price

Mineral Reserve Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	2,204	1.22	11.17	0.13	87	792	6,304
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	2,204	1.22	11.17	0.13	87	792	6,304
Çöpler Mine – Sulfide							
Proven Mineral Reserve	408	2.02	6.69	–	26	88	–
Probable Mineral Reserve	35,828	2.13	4.96	–	2,455	5,713	–
Probable – Stockpile	12,468	2.25	–	–	900	–	–
Total Mineral Reserve	48,703	2.16	3.70	–	3,382	5,801	–
Çakmaktepe Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	274	1.26	10.91	–	11	96	–
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	285	1.32	10.49	–	12	96	–
Ardich – Oxide Reserve							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	381	464	208
Probable Mineral Reserve	13,305	1.74	1.98	0.01	742	849	2,933
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	20,050	1.74	2.04	0.01	1,124	1,313	3,141
Ardich – Sulfide							
Proven Mineral Reserve	1,871	5.55	10.83	–	334	651	–
Probable Mineral Reserve	2,253	3.13	4.35	–	227	315	–
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	4,124	4.23	7.29	–	560	966	–
CDMP21 Mineral Reserves – Oxide Subtotal							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	381	464	208
Probable Mineral Reserve	15,783	1.66	3.42	0.03	840	1,736	9,237
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	22,539	1.69	3.04	0.02	1,222	2,200	9,445
CDMP21 Mineral Reserves – Sulfide Subtotal							
Proven Mineral Reserve	2,278	4.92	10.09	–	360	739	–
Probable Mineral Reserve	38,081	2.19	4.92	–	2,682	6,028	–
Probable – Stockpile	12,468	2.25	–	–	900	–	–
Total Mineral Reserve	52,827	2.32	3.98	–	3,942	6,768	–
CDMP21 MINERAL RESERVES – OVERALL TOTAL							
Proven Mineral Reserve	9,024	2.55	4.15	0.00	741	1,203	208
Probable Mineral Reserve	53,863	2.03	4.48	0.01	3,522	7,765	9,237
Probable – Stockpile	12,479	2.25	–	–	901	–	–
Total Mineral Reserve	75,366	2.13	3.70	0.01	5,164	8,968	9,445

1. Mineral Reserves are reported based on 31 December 2021 topography surface.
2. The Mineral Reserves were scheduled based on End-of-August 2021topography surface. Small differences between the Mineral Reserve statement and the production schedule may occur.
3. Mineral Reserves are shown on a 100% basis. Çöpler and part of Ardich are on Anagold 80:20 ground on which SSR holds 80% rights, and Çakmaktepe and the remainder of Ardich are on Kartaltepe 50:50 ground on which SSR holds 50% rights.
4. Mineral Reserve cut-offs are based on $1,350/oz gold price; average oxide recoveries are 61% and average sulfide recoveries are 91%.
5. Cut-off values are shown in Table 15.2. All cut-off values include allowance for royalty payable. There are no credits for silver or copper in the cut-off calculations.
6. There is no Çakmaktepe sulfide Mineral Reserve or Bayramdere Mineral Reserve.
7. Economic analysis has been carried out using a long-term gold price of $1,600/oz.
8. The point of reference for Mineral Reserves is the point of feed into the processing facility.
9. Tonnage is metric tonnes, ounces represent troy ounces, and g/t represents grams per metric tonne.
10. Totals may vary due to rounding.

Table 15.2 Summary of Cut-off Values, Metallurgical Recoveries, and SSR Ownership of CDMP21TR Mineral Reserves Estimate (as at 31 December 2021) Based on $1,350/oz Gold Price

Mineral Reserve Classification	Tonnage (kt)	Grades			Cut-off Value/s (g/t Au)	Metallurgical Recovery (%)	SSR Ownership (%)
		Au (g/t)	Ag (g/t)	Cu (%)			
Çöpler Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	2,204	1.22	11.17	0.13	0.47–0.59	62.3–78.4	80
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	2,204	1.22	11.17	0.13	0.47–0.59	62.3–78.4	80
Çöpler Mine – Sulfide							
Proven Mineral Reserve	408	2.02	6.69	–	1.05	85	80
Probable Mineral Reserve	35,828	2.13	4.96	–			
Probable – Stockpile	12,468	2.25	–	–			
Total Mineral Reserve	48,703	2.16	3.70	–			
Çakmaktepe Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	274	1.26	10.91	–	0.52–0.71	14–80	50
Probable – Stockpile	11	2.69	–	–			
Total Mineral Reserve	285	1.32	10.49	–			
Ardich – Oxide							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	0.44–0.80	40–73	77
Probable Mineral Reserve	13,305	1.74	1.98	0.01			
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	20,050	1.74	2.04	0.01	0.44–0.80	40–73	77
Ardich – Sulfide							
Proven Mineral Reserve	1,871	5.55	10.83	–	1.11	83	78
Probable Mineral Reserve	2,253	3.13	4.35	–			72
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	4,124	4.23	7.29	–	1.11	83	75
CDMP21 Mineral Reserves – Oxide Subtotal							
Proven Mineral Reserve	6,745	1.76	2.14	0.00	0.44–0.80	14–80	77
Probable Mineral Reserve	15,783	1.66	3.42	0.03			77
Probable – Stockpile	11	2.69	–	–	0.52–0.71	14–80	50
Total Mineral Reserve	22,539	1.69	3.04	0.02	0.44–0.80	14–80	77
CDMP21 Mineral Reserves – Sulfide Subtotal							
Proven Mineral Reserve	2,278	4.92	10.09	–	1.05–1.11	83–85	78
Probable Mineral Reserve	38,081	2.19	4.92	–			79
Probable – Stockpile	12,468	2.25	–	–			80
Total Mineral Reserve	52,827	2.32	3.98	–			79
CDMP21 MINERAL RESERVES – OVERALL TOTAL							
Proven Mineral Reserve	9,024	2.55	4.15	0.00	0.44–1.11	14–85	77
Probable Mineral Reserve	53,863	2.03	4.48	0.01			79
Probable – Stockpile	12,479	2.25	–	–			80
Total Mineral Reserve	75,366	2.13	3.70	0.01			78

1. Mineral Reserves are reported based on 31 December 2021 topography surface.
2. The Mineral Reserves were scheduled based on End-of-August 2021topography surface. Small differences between the Mineral Reserve statement and the production schedule may occur.
3. Mineral Reserves are shown on a 100% basis. SSR Ownership is an average based on location of Mineral Reserves (gold) relative to licenses: Çöpler and part of Ardich are on Anagold 80:20 ground on which SSR holds 80% rights, and Çakmaktepe and the remainder of Ardich are on Kartaltepe 50:50 ground on which SSR holds 50% rights. Totals and Ardich ownership percentages are weighted averages.
4. Mineral Reserve cut-offs are based on $1,350/oz gold price; average oxide recoveries are 61% and average sulfide recoveries are 91%.
5. All cut-off values include allowance for royalty payable. There are no credits for silver or copper in the cut-off calculations.
6. There is no Çakmaktepe sulfide Mineral Reserve or Bayramdere Mineral Reserve.
7. Economic analysis has been carried out using a long-term gold price of $1,600/oz.
8. The point of reference for Mineral Reserves is the point of feed into the processing facility.
9. Tonnage is metric tonnes and g/t represents grams per metric tonne.
10. Totals may vary due to rounding.

Significant factors that could materially affect the Mineral Reserve are:

- Environmental, Permitting, Social, and Community – the Çöpler project is subject to the laws and regulations of Turkey, the mine has several local communities that are nearby. In order to operate the mine, Anagold must maintain appropriate relations with all the authorities and stakeholders. Social, community and government relations are managed by Anagold and include programmes and engagement with the local communities and both local and national governments. Anagold has remained in compliance with all aspects of the Environmental Impact Assessments (EIA) and operating permits throughout the history of the project.
- Seismic impacts – the Çöpler project is in an area with a history of significant seismic activity that could negatively impact mining operations.
- Metal price impacts – gold is the primary revenue element and silver and copper are produced as by-products. The ore is mined at an elevated cut-off grade and low-grade ore is stockpiled for processing after mining is completed. The use of the elevated cut-off grade serves to mitigate the risks from periods of lower gold prices.
- Mining impacts – the mining equipment is suitable for a selective mining unit (SMU) of approximately 3 m x 3 m x 5 m. This allows for selectivity in mining and enhances the opportunities for blending the feed to the sulfide plant. The total mining rates in the CDMP21TR mine plan are at 22.5 Mtpa (Çöpler mining only), In the past, total mining rates of 36.5 Mtpa (combination of Çöpler and Çakmaktepe mining) have been achieved, increasing the total mining rate may allow gold to be brought forward in the production schedule but will require additional stockpile storage areas.
- Geotechnical impacts – slope recommendations have significant impacts on the Mineral Reserve and the continued study will allow the Mineral reserves to be maximised.
- Processing impacts – the processing analysis in the Reserve Case includes incorporation of a flotation circuit into the existing sulfide plant to upgrade sulfide sulfur (SS) to fully utilise grinding and pressure oxidation (POX) autoclave capacity. Continued debottlenecking of the sulfide plant and optimisation of the flotation circuit may improve costs and recoveries, changing cut-off grades and impacting the Mineral Reserve.
- The addition of the flotation circuit to the sulfide plant required new grade control protocols and associated stockpile strategies to be implemented to manage the required sulfide plant feed blend. It is likely that there will need to be ongoing modification of the stockpiling cut-offs and procedures for both short-term and longer term blending as the mine progresses. Measures such as increasing the number of active mining areas, increasing the mining rate, and increasing the size or number of run-of-mine (ROM) stockpiles may be required.

15.3 Comparison of 2021 Mineral Reserves to 2020 Mineral Reserves

The 2021 Mineral Reserves inventory is detailed in Table 15.1.

For comparison, a summary of the overall 2020 Mineral Resource inventory is shown in Table 15.3.

Table 15.3 EOY 2020 Mineral Reserves Summary as at 31 December 2020

Mineral Resource Classification	EOY 2020 Mineral Resources Total (Exclusive of Mineral Reserves)						
	Tonnage (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Proven	2,196	2.31	7.70	0.01	163	544	249
Probable	54,610	2.10	5.03	0.02	3,681	8,836	18,116
Probable Stockpile	–	–	–	–	84	–	–
Total	**56,807**	**2.10**	**5.14**	**0.01**	**3,928**	**9,379**	**18,365**

The factors contributing to the differences between the 2021 Mineral Reserves and the previous Mineral Reserves reported as at 31 December 2020 are as follows:

Ardich

- There was no Mineral Reserve declared for Ardich in 2020. A maiden Mineral Reserve has been declared for Ardich in 2021, and this has added 24Mt at 2.17 g/t Au to the Mineral Reserve, increasing the total by 1.68 Moz of gold.

Other

- New designs for two new phases beneath the Çöpler pit
- Depletion through mining since 31 December 2020

There has been an 33% increase in tonnage above the cut-off across all combined Mineral Reserve categories, with a corresponding 31% increase in contained gold.

15.4 Subpart 1300 of US Regulation S–K Mining Property Disclosure Rules

The Mineral Reserves reported in the CDMP21TR are suitable for reporting as Mineral Reserves in accordance with Subpart 1300 of US Regulation S-K Mining Property Disclosure Rules (S-K 1300).

16 MINING METHODS

The objective of the CDMP21TR is to provide a consistent and structured growth plan for the business. Mine plans were updated to improve metal production, through a revised development sequence, available information was then consolidated into a growth strategy, for communication to all levels of the business, using recognised progress reporting systems.

16.1 Geotechnical

16.1.1 Pit Slope Stability – Çöpler

This section contains a summary of the feasibility study level mining geotechnical investigation and design conducted for the Çöpler mine. Much of this work has been prepared prior to 2020, however the work and the recommendations are still applied to the mine designs and workings.

The Çöpler mine maintains an on-site geotechnical monitoring programme that consists of 58 prisms, 33 extensometers, a long-range synthetic aperture radar, and daily data and field monitoring. Additional work is currently in progress to implement pit slope depressurisation. It is expected that pit slope depressurisation will be used extensively throughout the Main pit as the sulfide pit phases are progressed.

In April 2015, Golder Associates (Golder) completed a pit slope optimisation study intended to further optimise the pit slope angles as defined in their earlier study completed in April 2014. This programme included the drilling of five oriented geotechnical core holes to identify any prevalent jointing throughout the Çöpler deposit.

Golder completed the 2015 pit slope optimisation study using recommendations from the 2014 Golder pit slope review with the intention of identifying opportunities to increase definition of potential problem areas within the Çöpler pit. This would allow for mine planning and designs to take advantage of steeper slope angles in some areas. No material changes in-pit slope recommendations were made with the updated report. Anagold chose to continue using the more conservative slope angle recommendations made by Golder in 2014.

The results of the study have provided Anagold with a much better understanding of potential highwall conditions. Not all slope angle recommendations made by Golder were able to be fully followed due to a lack of data and modelling of alteration zones within the Çöpler deposit. Where slope angles were not able to be further refined, Golder recommended that Anagold follow the recommendations set forth in the 2014 geotechnical review.

16.1.2 Review of 2021 Geotechnical Studies

Golder completed three additional Geotechnical Studies / Reports in 2021 which include:

- Golder, Data Review and Geotechnical Model, Çöpler Geotechnical Design Review (October 2021), (PowerPoint Presentation).
- Golder, Çöpler Pit Slope Design Review (November 2021).

- Golder, 2021 Ardich Project Slope Stability Study, Geotechnical support for the Pre-Feasibility Study (December 2021).

A geotechnical review of the 2021 Golder reports was completed in December 2021. Key areas of concern and recommendations, where applicable, are provided in Section 16.1.2. The Golder reports focus on the following keys areas:

- Update on structural data
- Update on rock and soil strengths
- Update on rock mass quality
- Comments on the geological surfaces
- Comments on geotechnical studies commissioned by Anagold
- Geometric review of design
- Review of piezometer data

16.1.2.1 Structural Data

The outcomes of the update on structural data, with focus of areas where faults provided potential control on overall slopes (circled areas in Figure 16.3), are considered appropriate.

16.1.2.2 Rock and Soil Strengths

Whilst the November 2021 report suggests limited 'new' laboratory testing of rock strength materials, there was significant testing by Golder in 2020 to justify a review of the 2019 intact rock strengths. Table 16.1 provides an overview and indicates significantly lower intact strengths to those reported by Golder in 2019 (Pit Slope Stability Evaluation, Çöpler Open Pit Mine, November 2019) and with the bolded values in Table 16.1 highlighting the significant changes. Based on Hoek & Brown envelopes and with comparison over a normal stress range of 40–700 kPa the 2021 strengths are somewhat lower than previous Golder studies and with reductions of nominally 35% for Diorite and 12% for Metasediments.

Table 16.1 Golder Intact Strength Estimates

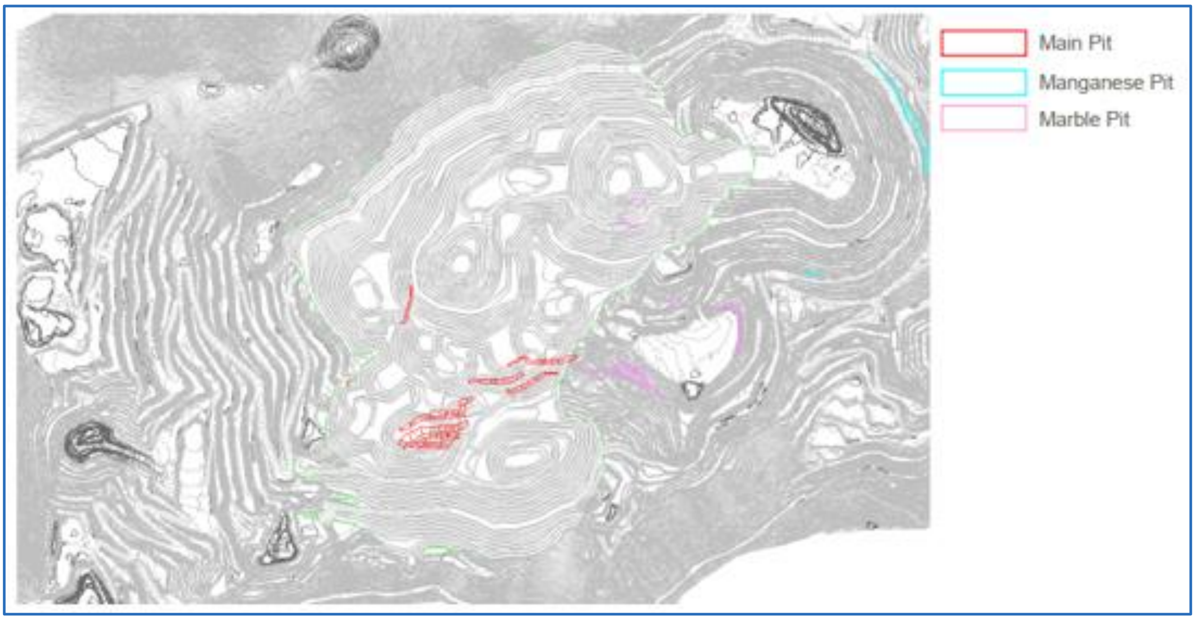
Rock Type	Golder 2019		Golder 2021[1]	
	UCS (MPa)	mi	UCS (MPa)	mi
Metasediments	49	22	41	**14**
Diorite	42	28	**22**	24
Carbonates	41	10		

[1] 35% percentile values suggested by Golder

16.1.2.3 Rock Mass Quality

It is understood that Anagold has undertaken mapping of rock mass quality, GSI, within the pit between 2015 and 2017, Figure 16.1.

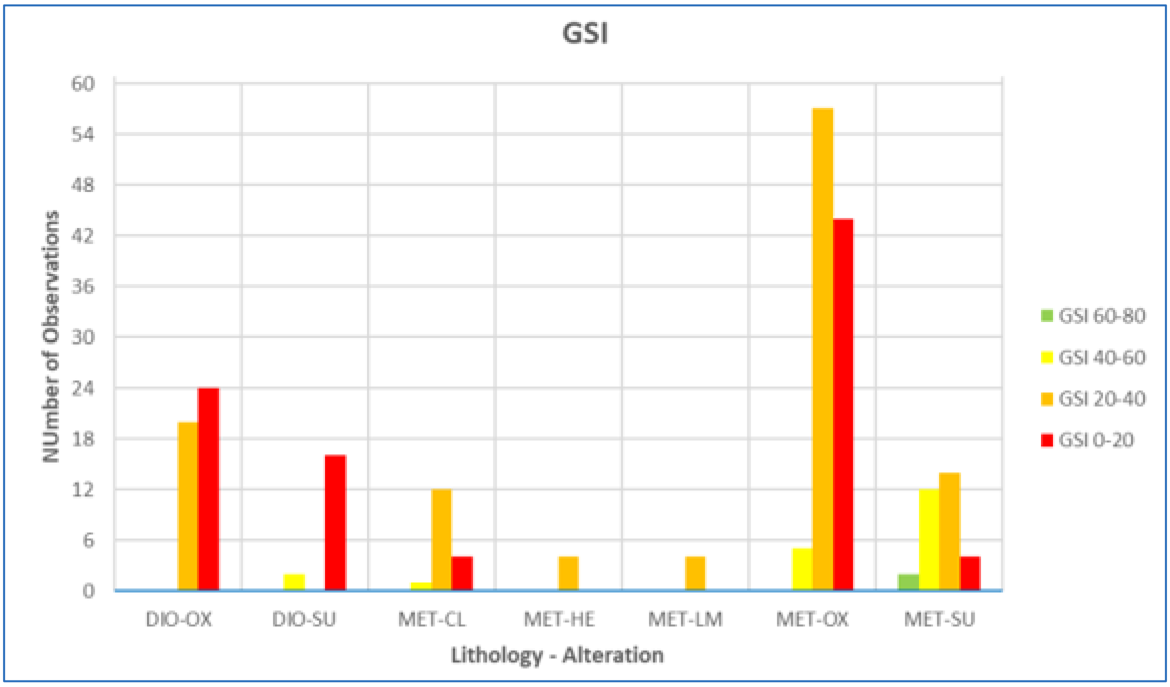
Figure 16.1 Rock Mass Quality Mapping Locations (2015–2017)



The mapping indicates GSI values typically below 40. Figure 16.2 provides the data from mapping of the Main pit, predominantly of Diorite and Metasediments and where the majority of mapping took place (225 measurements). Whilst additional mapping of the Marble and Manganese pits also took place, mapping was limited to 42 data points. There is no legend provided for alteration types so it is assumed the alteration noted as OX relates to altered materials and SU relates to presence of sulphides and hence fresh rock. The data in Figure 16.2 infers median GSI values of nominally:

- 19 for altered diorite
- Less than 20 for fresh diorite
- 23 for altered metasediments
- 38 for fresh metasediments

Figure 16.2 Rock Mass Quality Mapping of Main Pit



The Anagold mapping indicates significantly lower rock mass quality than what has been adopted in the Golder studies and with the following values noted for rock like materials by Golder:

- 41 for diorite
- 52 for metasediments
- 61 for marble

Golder assign the lower GSIs mapped by Anagold, largely as a result of two factors:

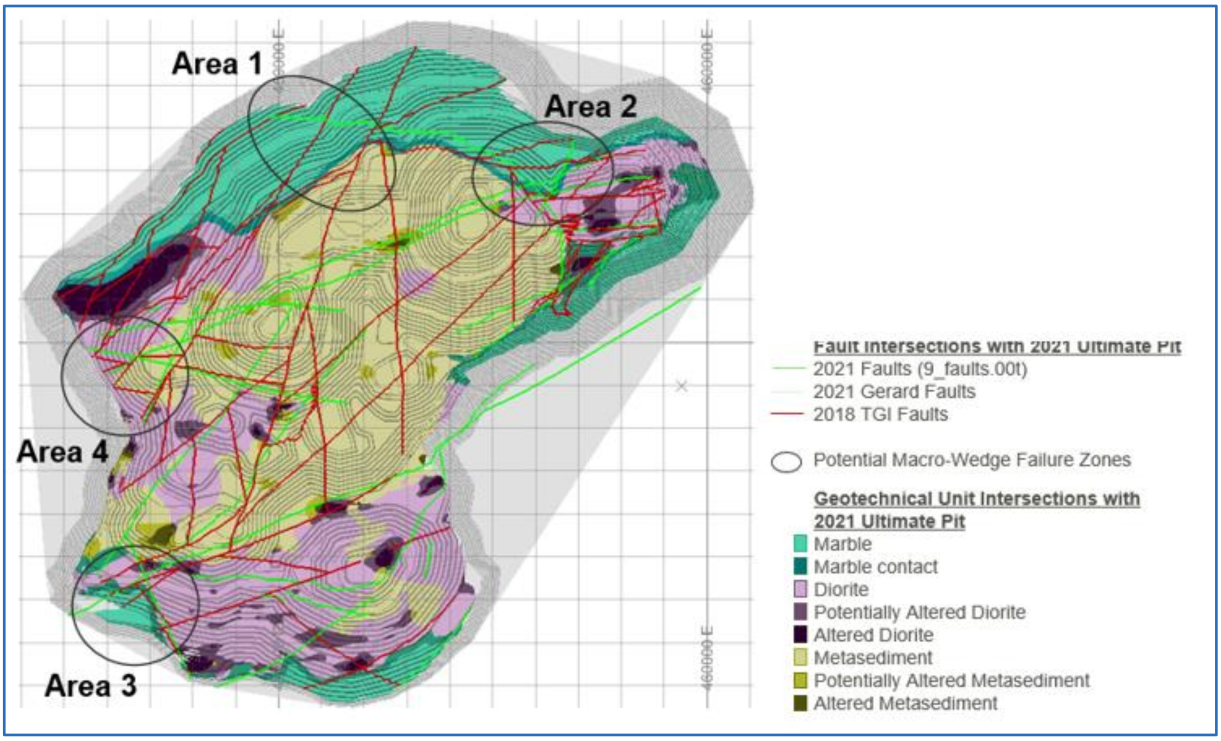
- Observed intact strengths and RQD in the field being lower than typically seen in the core which had been utilised by Golder in assessing rock mass quality of the rock like materials.
- Greater percentage of soil-like material in the exposures and hence significantly poorer GSIs.

The last item is considered as significant and particularly in view that Golder has assumed the percentages of rock-like material comprise 60% of the diorite and 75% of the metasediments. This aspect is further discussed in the following Section.

16.1.2.4 Geological Surfaces

Golder discusses the Anagold geological models that comprise major lithological units and an RQD block model, the latter utilised by Anagold to provide alteration state. Anagold has utilised comments in the Barr 2012 study (Pit Wall Stability Analysis, Çöpler Mine, August 2012) whereby an RQD of less than 15% implied as signifying altered material, Anagold nominally accepting an RQD of 15 to 25% as being potentially altered and an RQD greater than 25% as unaltered. Figure 16.3 provides the Anagold geotechnical units and which suggest alteration in the final pit occurs in limited areas.

Figure 16.3 Geotechnical Units



Golder comments that the RQD ranges selected by Anagold are too narrow and suggest altered material could have RQD of up to 40% and unaltered material with RQD greater than 60%. These comments by Golder appear appropriate if one considers the location of the rock mass quality mapping discussed above, compare Figure 16.1 and Figure 16.2, which is largely in unaltered materials according to the Anagold model but with the mapping results suggesting largely soil-like materials.

It's recommended a rock mass quality model be created to allow the slope design recommendations to be appropriately implemented. Such a model needs to take into consideration the Anagold rock mass quality mapping, which suggests there is a higher proportion of soil like materials in the slopes than the Golder estimates.

16.1.2.5 Geotechnical Reporting Commissioned by Anagold

Golder notes several studies, including both external and internal reports. Two reports relate to internal design checks, which indicate a requirement for localised flattening of designs in areas of altered materials. For the two examples presented, the issues relate to potential impacts on slopes over two benches high and with very high Factors of Safety (FOS) for global stability. Golder also notes "local instabilities may occur where pockets of altered rock mass are exposed on benches. If the altered rock mass area is significant (i.e., exposed over more than 2 benches) or if the local instabilities cannot be managed by operations, then the mine may need to locally adapt the slope configuration to the altered rock mass slope design or change the whole domain to the shallower altered rock mass design angles".

Golder notes Anagold had engaged a third party, Professor Tamer Topal, to address the south wall failure in the Marble pit. The scope appears to be limited to a single stability analysis (i.e., one cross-section) based on piezometer data from specific drilled boreholes and utilising all available monitoring (inclusive of inclinometers). However, the scope does not comprise slope recommendations for the failure. Whilst the study has merit, with three large failures and with the Marble pit south wall having failed twice previously it would be prudent that the failures be appropriately back-analysed to confirm that the Geotechnical Model and design assumptions remain appropriate as a key check on inputs for the designs. Without consideration of all failures the results of the Topal study, limited to the current failure, may not provide robust feedback on geotechnical parameters for slope designs going forward.

16.1.2.6 Geometric Review of Design

Golder notes the following regarding the 2021 LOM design "the geometric design for the Çöpler pits conforms with Golder pit slope design recommendations, however, some critical areas have been identified and further design evaluations are recommended". Following is a summary of the key aspects noted by Golder:

- "Details of the operational pit slope performance is currently unknown to Golder. The pit slope performance would provide valuable information".
- Requirement for slope stability analyses to address "maximum vertical height for uninterrupted inter-ramp slopes and geotechnical berm widths".
- "Anagold has recently developed 3D solids for altered and potentially altered rock masses based on the RQD evaluation from the entire database. Golder currently is not completely aware of the details of how these models were developed and how they are representative of the field conditions".
- "Adequacy of the planned set back distance from designed pit crest to the existing waste rock dump should be evaluated with the slope stability analyses".

For Items 2 and 4 above, Golder does not clarify if these stability analyses would utilise revised geotechnical parameters. It is considered these would be best addressed once the rock mass quality model is created and which would include and consider results of back-analyses and slope performance.

Item 3 above requires appropriate interaction between Anagold and Golder such that an appropriate rock mass quality model is developed either “back-boned” to existing Anagold models or appropriately utilising geotechnical borehole logging data and rock mass quality mapping to develop a model. The primary aim, regardless of approach is that the Golder slope design recommendations relate to an Anagold model so that designs can be appropriately implemented.

16.1.2.7 Piezometer Data

The available seepage locations and piezometer data are provided by Golder but there is key information that is not addressed by Golder and these comprise:

- The seepage is focused at the contact between Diorite and Metasediments, compare Figure 16.3 and Figure 16.4.
- No clear indication of significance of the piezometer data.

The nominal piezometer locations are presented in Figure 16.4 (red dots) and the compiled data in form of a hydrograph in Figure 16.5. The followings trends observed are as follows:

- Significant compartmentalisation in piezometer Pa3.
- Lower groundwater level near limestone contact, piezometer Pa1.
- Groundwater conditions elsewhere indicating phreatic surface at the mined slope and with a Hu of nominally 70% (i.e., 70% of hydrostatic) and indicating depressurisation of the slopes.

Figure 16.4 Locations of Seepage and Ponding

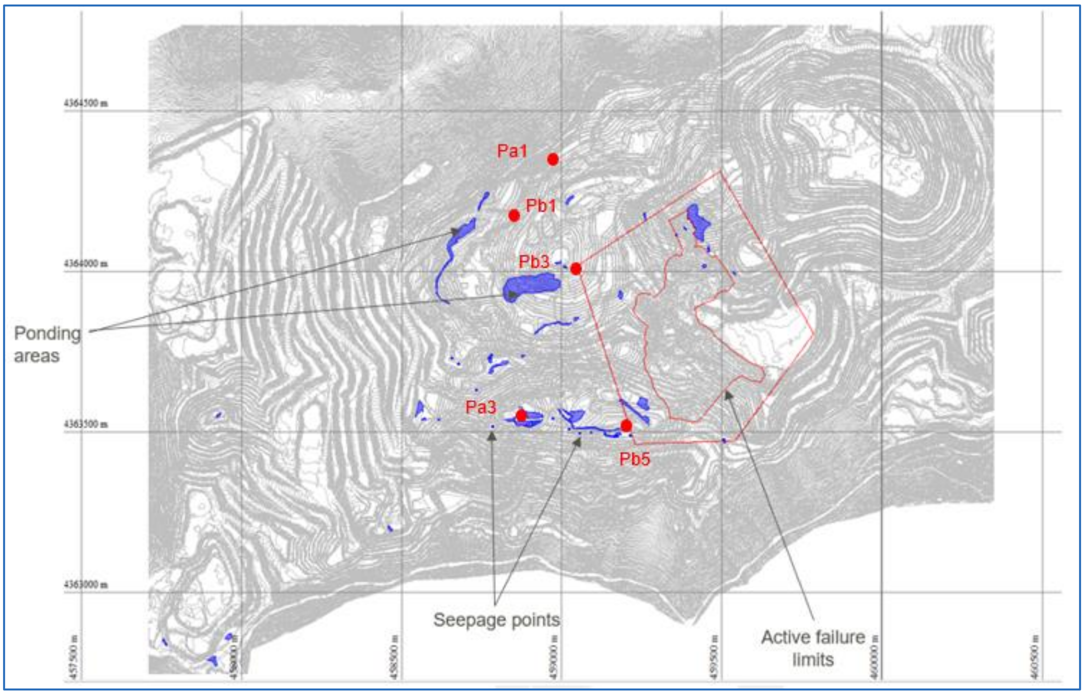
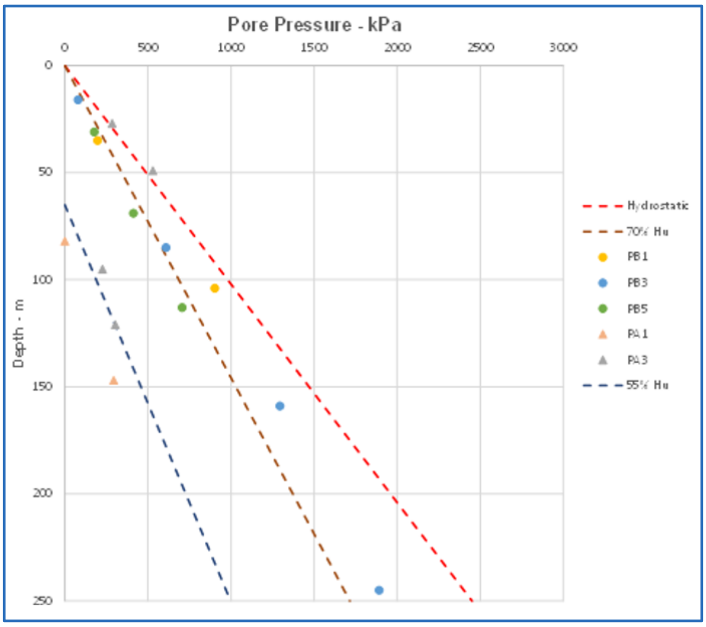


Figure 16.5 Çöpler VWP Data



16.1.2.8 Review Summary of 2021 Çöpler – Golder Geotechnical Reports

The review of the 2021 Çöpler – Golder Geotechnical Reports has highlighted potential issues of concern. The following are key recommendations from the Geotechnical review.

- Appropriate interaction between Anagold and Golder is required such that an appropriate rock mass quality model is developed either "back-boned" to existing Anagold models or appropriately utilising geotechnical borehole logging data and rock mass quality mapping to develop a model. The primary aim, regardless of approach is that the Golder slope design recommendations relate to an Anagold model so that designs can be appropriately implemented.
- A Feedback loop of appropriate revision of strengths based on back-analysis of failure and review of slope performance.
- Stability analyses once the above components are completed and with appropriate revision of slope design parameters.

16.1.3 RQD Model

RQD is used as a simple and inexpensive indication of rock mass quality. RQD does not account for joint orientation, continuity, or gouge material. Joints sets parallel to the core axis will not intersect the core and therefore is it recommended to use RQD in combination with other geotechnical inputs. RQD is a measure of percent core recovery with artificial fractures ignored.

At the Çöpler project, it has been determined that RQD is a generally reliable indicator of alteration. Therefore, areas with RQD modelled as being less than 15% are considered altered.

Standard testing of RQD was collected on 661 diamond core holes, 30 of which were drilled within the pit for metallurgical purposes. The 661 holes represent approximately 34% of all drilling in the Çöpler deposit. The Main pit contains RQD measurements for holes evenly spaced with data gaps occurring in the Manganese, Marble, and West pits.

RQD was interpolated in the resource model using the inverse distance method, weighted to the power of two (ID2) with 2 m drillhole composites. A total of six domains were used to estimate RQD values and included a distinction between oxide and sulfide material. To account for the variance in sample spacing, a two-pass approach was used to capture available samples. Model cell estimates were limited to the search distances used with no attempt to assign RQDs to unestimated cells.

16.1.4 Pit Slope Design Parameters

The pit slope design parameters remain unchanged and those applied for each deposit are shown in Table 16.2. Note that for Çakmaktepe, design parameters are in relation to the Central pit and based on the 2018 Golder study which are defined based on azimuth (i.e., direction the slope faces).

Table 16.2 Çöpler Mine Pit Slope Parameters

Çöpler Rock Type	**Interramp Slope Angle Çöpler Pits**	
	Altered RQD<15%	**Un-altered (Fresh) RQD>15%**
Diorite	23	38
Metasediment	32	43
Marble	50.5	50.5
Gossan Massive Sulfides	40	40

Çakmaktepe Slope Direction	Interramp Slope Angle Çakmaktepe Central Pits
0° to 180° (south-west wall)	34
180° to 360° (all other walls)	40

Golder site review, Çöpler and Golder 2018 for Çakmaktepe

16.1.5 Mine Operations Monitoring and Management

Pit slopes in the Çöpler pit are monitored daily to ensure safety and stability. Daily inspections of the active mining areas are conducted by shift engineers to identify hazards such as unstable rock on benches above, excessive water in and around the highwalls, and any visible cracking and movement of the highwalls. In addition, Anagold employs a geotechnical management team consisting of surveyors, geologists, and geotechnical engineers. This team conducts regular highwall inspections, measurement of movement through extensometers and prism surveys, and data collection and interpretation of the long-range synthetic aperture radar measurements.

Mining at Çöpler utilises perimeter pre-split blasting techniques in areas where competent rock is encountered (typically, limestone/marble, unaltered metasediment, and unaltered diorite). The pre-split holes are drilled according to the bench face angle recommendations as shown in Table 16.2. Blasting is conducted in a manner to minimise back-break through usage of delays and providing adequate relief. A typical pre-split highwall at Çöpler is shown in Figure 16.6.

Where pre-splitting is not practical, highwalls are sloped by excavator to the recommended bench face angle.

Figure 16.6 Typical 15 m Pre-Split at Çöpler Mine

Anagold, 2020

16.1.6 Geotechnical Domains

Based on the 2014 Golder geotechnical site review, the following geotechnical domain categories are considered appropriate for design recommendations to be founded upon:

- Marble / limestone – characterised by competent rocks and marbleised near the Çöpler intrusion.
- Fresh diorite – characterised as a fresh to slightly weathered or altered moderately strong rock.
- Hydrothermally altered diorite – alteration sufficient to significantly reduce strength relative to fresh diorite, but without the shearing and intense clay alteration of contact and fault zones.
- Weathered diorite and metasediment – highly weathered, extremely weak rock and soil that occurs in the oxidised zone (depth typically to 30 m).
- Fresh metasediment – fresh to slightly weathered, weak to moderately strong rock consisting of a turbidite sequence that may also be structurally complex near faults.
- Hydrothermally altered metasediment – alteration sufficient to significantly reduce strength relative to fresh metasediment, but without the shearing and intense clay alteration of Contact and Fault zones.
- Fault gouge including intrusive contact and intense sulfide alteration – slicken sided plastic clay with rock fragments that occurs in fault zones including the intrusive contacts.

The character and extent of the hydrothermal alteration beyond the fault zones is poorly defined. Where data are lacking within the alteration zones the most conservative pit slope angle is assumed, representing up-side potential should the alteration zone be further defined in the geological model.

The above listed geotechnical domains are mostly well known and modelled in a geologic model. The alteration zones, however, vary significantly and have not been modelled to an extent to where variations by alteration type are well defined. It has been recommended by Golder that the best way to identify alteration zones is by modelling RQD in the geologic model. For this purpose, RQD values of 15% and less are considered altered and RQD values greater than 15% are considered un-altered, or fresh.

16.1.7 Pit Dewatering

Earlier studies have predicted the formation of pit lakes at various stages of mining. Golder's hydrogeological study was used to predict pit lake formation. The groundwater flow model predicted that a pit lake would form over time after mining. These results, in conjunction with the acid rock drainage (ARD) work being conducted by SRK Turkey, are being used to predict pit lake water quality.

Sources of groundwater recharge include direct infiltration of precipitation and/or infiltration during storm water run-off events throughout the entire site. Fractured or karstic openings in the bedrock and alluvial sediments along drainages are considered to be the predominant pathways for infiltration. The main hydrogeological units and features considered in the groundwater model were:

- Limestone (modelled hydraulic conductivity = 0.6 m/day)
- Diorite (modelled hydraulic conductivity = 0.0002 m/day)
- Metasediments (modelled hydraulic conductivity = 0.0002 m/day)
- Alluvium (modelled hydraulic conductivity = 10 m/day)
- Various fault systems (Sabirli, Çöpler, and Other) (modelled hydraulic conductivity = 6.1 m/day)

The calibrated groundwater model was used to predict pit inflows and pit lake development based on a pit design with a maximum depth to 875 m. This analysis estimated pit inflow at less than approximately 1,100 m^3/day. Estimations of pit lake formation suggest that over a 100-year scenario, based on a pit design with a maximum depth to 875 m, pit lake water elevations are projected to reach the 906 m elevation (±20 m). Modelling results indicate that water from beneath the Lower Çöpler West waste rock dump (WRD) will take more than 1,000-years to flow to the Karasu River. Groundwater located beneath the Lower Çöpler East WRD is estimated to discharge to the Karasu River within approximately 300-years.

Revisions to the pit design since the groundwater model was constructed and calibrated (in 2012) show that the minimum pit elevation (895 mRL) will be higher than the minimum pit elevation simulated in the model (875 mRL). Additionally, the area on the north side of the pit and the southern and south-eastern portions of the pit will be mined to a lower elevation than simulated in the model. Limestone in these areas may increase discharge to the pit during dewatering and may impact the formation of a pit lake following closure. Updating and possibly recalibrating the model based on the revised ultimate pit configuration and available data since 2012 would be required to better quantify the magnitude of the increase or impact.

16.2 Mine Plan

Open pit mining at the Çöpler project is carried out by a mining contractor and managed by Anagold. The mining method is a conventional open pit method with drill and blast and utilising excavators and trucks operating on bench heights of 5 m. The mining contractor provides operators, line supervisors, equipment, and ancillary facilities required for the mining operation. SSR provides management, technical, mine planning, engineering, and grade control functions for the operation.

SSR currently operates a sulfide process plant and an oxide heap leach facility. Costs are based on the actual operational costs and project budget assumptions. Production schedules and costs are based on current site performance and contracts.

The parameters, costs and throughput assumptions used to prepare cut-off grades and the production schedule are listed in the following sections.

16.2.1 Ore Definition

A revised set of processing parameters was used to calculate the internal Au cut-off grades for ore definition. The cut-off grades for the CDMP21TR were calculated using the parameters described in the following sections.

16.2.1.1 Oxide Heap Leach Parameters

Table 16.3 details the gold recovery parameters by material type and location.

Table 16.3 Heap Leach – Gold Recoveries

Location	Unit	Material Types						
		LMS	M/SED	GOSS	JASP	DIO	MNDIO	OPH
Çöpler Manganese	%	78.4	66.8	71.2	–	71.2	71.2	–
Çöpler Main	%	68.6	66.8	71.2	–	71.2	71.2	–
Çöpler Marble	%	75.7	66.8	65.1	–	62.3	62.3	–
Çakmaktepe Central	%	70.0	80.0	–	73.0	61.0	–	70.0

Table 16.4 details the silver recovery parameters by material type and location.

Table 16.4 Heap Leach – Silver Recoveries

Location	Unit	Material Types						
		LMS	M/SED	GOSS	JASP	DIO	MNDIO	OPH
Çöpler Manganese	%	27.3	32.5	27.5	–	37.8	37.8	–
Çöpler Main	%	24.6	32.5	27.5	–	37.8	37.8	–
Çöpler Marble	%	34.0	32.5	27.5	–	32.0	32.0	–
Çakmaktepe Central	%	17.0	28.0	–	17.0	24.0	–	19.0

Table 16.5 details the copper recovery parameters by material type and location.

Table 16.5 Heap Leach – Copper Recoveries

Location	Unit	Material Types						
		LMS	M/SED	GOSS	JASP	DIO	MNDIO	OPH
Çöpler Manganese	%	3.5	13.8	3.3	–	15.8	15.8	–
Çöpler Main	%	3.5	13.8	3.3	–	15.8	15.8	–
Çöpler Marble	%	3.5	13.8	3.3	–	15.8	15.8	–
Çakmaktepe	%	–	–	–	–	–	–	–

Table 16.6 details the operating costs by location.

Table 16.6 Oxide Operating Costs

Parameter	Unit	Çöpler	Çakmaktepe
Rehandle Cost	$/t	0.32	0.64
Processing – Fixed	$/t	3.05	3.05
Processing – Variable	$/t	8.94	8.94
G&A (Process and Site)	$/t	3.17	3.17
Ore Haulage	$/t	–	1.53
Mining Cost	$/t mined	1.89	1.59

16.2.1.2 Sulfide Plant Parameters

The following sections outline the processing parameters for the sulfide plant. Average life-of-mine (LOM) sulfide gold recoveries are 91%.

Throughput

Total Plant Throughput = Direct POX Feed + Float Plant Feed

POX Plant Throughput = Direct POX Feed + Float Plant Concentrate

Table 16.7 details the maximum plant throughputs for each part of the plant. The front-end limit of 400 tph means when the flotation plant is running at full capacity (i.e., 150 tph), the direct feed to the pressure oxidation (POX) circuit will be limited to 250 tph.

Table 16.7 Plant Throughput Limits

Parameter	Unit	Maximum Throughput
Float Plant	t/hr	150
POX Plant	t/hr	280
Total	t/hr	400

Float Plant Throughput = 216345 x Feed SS%^{2} – 30592 x Feed SS% + 980.24

Float Concentrate Mass Pull = 277.09 x Feed SS%^{2} – 15.17 x Feed SS% + 0.33

The POX circuit throughput is also limited by the sulfide sulfur (SS) in the feed to the autoclave, which must be less than 13.75 tph. If the SS content is too high, then the POX circuit throughput will need to be reduced until the rate is less than 13.75 tph SS.

Recovery – POX Gold

POX Gold Recovery = a x (1 - EXP(- b x (Au(g/t) - c))) + d.

Table 16.8 details the POX gold recovery factors by material type.

Table 16.8 POX – Gold Recovery Parameters

Material Type	a	b	c	d
Limestone/Marble	98.3	1.4	–1.5	–1.00
Metasediment	97.7	1.4	–1.4	–1.00
Gossan	98.3	1.4	–1.5	–1.00
Jasperoid	98.3	1.4	–1.5	–1.00
Diorite	98.3	1.4	–1.5	–1.00
Mn Diorite	96.7	1.2	–1.4	–1.00
Ophiolite	98.3	1.4	–1.5	–1.00

Recovery – Float Plant

Float Concentrate Gold Recovery = 55%

Float Tails Gold Recovery = 43%

Float Concentrate SS Recovery = 75%

Table 16.9 details the operating costs by location

Table 16.9 Sulfide Operating Costs

Parameter	Unit	Amount
Rehandle Cost	$/t	0.90
Processing – Fixed	$/t	8.32
Processing – Variable	$/t	19.10
Processing – Variable (SS)	$/t SS	2.68
G&A (Process and Site)	$/t	6.60

16.2.1.3 Metal Prices and Realisation Assumptions

Cut-off grades were determined using a gold price of $1,350/oz. There are no credits for silver or copper in the cut-off grade calculations. Table 16.10 details revenue and realisation assumptions for the Au cut-off grades.

Table 16.10 Au Cut-off Grade Revenue and Realisation Assumptions

Parameter	Unit	Au Cut-off Assumption
Payment and Deductions		
Gold	$/oz	1,350
Payable	%	100
Treatment and Refining		
Selling	$/oz	8.54
Royalties		
Çöpler	%	2
Çakmaktepe	%	4

16.2.2 Ore Cut-off Grades

Internal cut-off grades have been calculated for each of the material types based on the economic inputs and assumptions are shown in Table 16.11. Internal cut-off grades have been used to calculate process quantities within the Reserve Case pit stages.

The addition of the flotation circuit to the sulfide plant required new grade control protocols and associated stockpile strategies to be implemented to manage the required sulfide plant feed blend. It is likely that there will need to be ongoing modification of the stockpiling cut-offs and procedures for both short-term and longer term blending as the mine progresses. Measures such as increasing the number of active mining areas, increasing the mining rate, and increasing the size or number of ROM stockpiles may be required.

Table 16.11 Internal Au Cut-off Grades

Mining Area	Ore Type	Rock Type	Zone	COG (Au g/t)
Çöpler	Oxide	Limestone / Marble	Manganese	0.47
			Main	0.53
			Marble	0.48
		Metasediment	Manganese	0.55
			Main	
			Marble	
		Gossan	Manganese	0.51
			Main	0.51
			Marble	0.56
		Diorite	Manganese	0.51
			Main	0.51
			Marble	0.59
		Mn Diorite	Manganese	0.51
			Main	0.51
			Marble	0.59
	Sulfide	All	All	1.05
Çakmaktepe	Oxide	Limestone / Breccia	Central	0.60
		Jasperoid		0.57
		Diorite		0.69
		Metasediment		0.52
		Ophiolite		0.60

16.2.3 Pit Design

New pit designs were created in 2021 based on updated metal prices and costs.

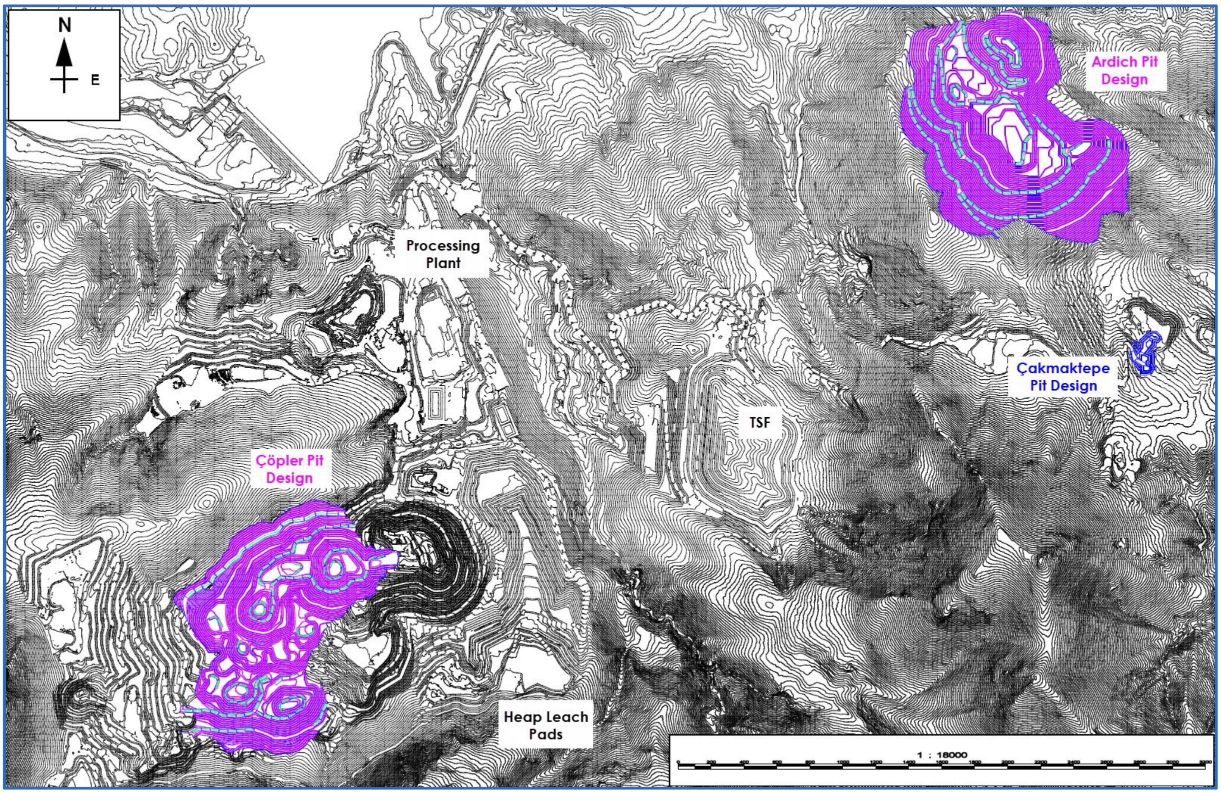
The key aims of the optimised pit designs are:

- Minimise mining costs and maximise economic return by exposing the highest value ore with minimum waste mining.
- Address operational requirements for loading, hauling, slope stability, and rockfall, as follows:
  - Loading – the phases were designed with a minimum operational width of 15–30 m between phases (depending on bench configuration) to allow efficient mining for the equipment scale.

– Hauling – generally, two exit haul roads per phase were included: the west bound exit to the crusher, low-grade stockpile, and west dump; and the east bound exit to the potentially acid forming (PAF) and non-acid forming (NAF) dumps. Haul roads are generally 15 m wide at a 10% gradient. Single-lane haulage traffic is allowed in the lower benches of the mine and is set at 10 m wide.

Pit designs for the Çöpler pit were updated in 2021. Ardich pit designs were prepared in 2021 and updated in 2022. The Çöpler, Ardich, and Çakmaktepe pit design for 2034, when in-pit mining is completed for the Reserve Case, is shown in Figure 16.7. Following completion of in-pit mining, the sulfide plant will be fed from stockpiles until 2043.

Figure 16.7 Ultimate Pit Designs – CDMP



Anagold, 2022

16.2.4 Waste Dump and Stockpile Design

The mine plan allows for the use of five WRDs to store mined waste rock and sulfide ore that is extracted during mining operations. These five WRDs are Lower Çöpler East, Lower Çöpler West, Upper Çöpler, West, and Marble Backfill WRDs. Current operations do not use the Lower Çöpler West and Marble Backfill WRDs. The Lower Çöpler East and Upper Çöpler WRDs will primarily be utilised as sulfide ore stockpile areas, with the Upper Çöpler WRD being mined out to allow for future pushback extension of the Marble pit towards the north and allow for leach pad extensions to the west. Figure 16.8 shows the site layout.

The Lower Çöpler East WRD has a capacity of 14.9 Mm^3 (26.8 Mt) of mine waste and 5.5 Mm^3 (9.9 Mt) of sulfide ore. The total surface area impacted by the Lower Çöpler East WRD is 51.5 ha. The Lower Çöpler West WRD has a capacity of 94.6 Mm^3 (170.3 Mt) of mine waste and 12.4 Mm^3 (22.3 Mt) of sulfide ore. The total surface area impacted by the Lower Çöpler West WRD is 206.5 ha. The Upper Çöpler WRD has a capacity of 7.6 Mm^3 (13.6 Mt) of sulfide ore. The total surface area impacted by the Upper Çöpler WRD is 26.1 ha. The West WRD complex has a capacity of 34.4 Mm^3 (61.9 Mt) of mine waste. The total surface area impacted by the West WRD is 108.9 ha.

An estimated 69.8 Mt of waste rock will be consumed in the construction of the tailings storage facility, haul road, and tailings pipeline corridor. Total constructed waste rock storage capacity is 155.0 Mm^3 (279.1 Mt). The total surface area impacted by all WRDs and stockpiles is 366.9 ha. When possible and economically preferable, waste rock will be backfilled within mined out areas of the pits as they become available.

16.2.4.1 Waste Rock Dump (WRD) Geotechnical Design

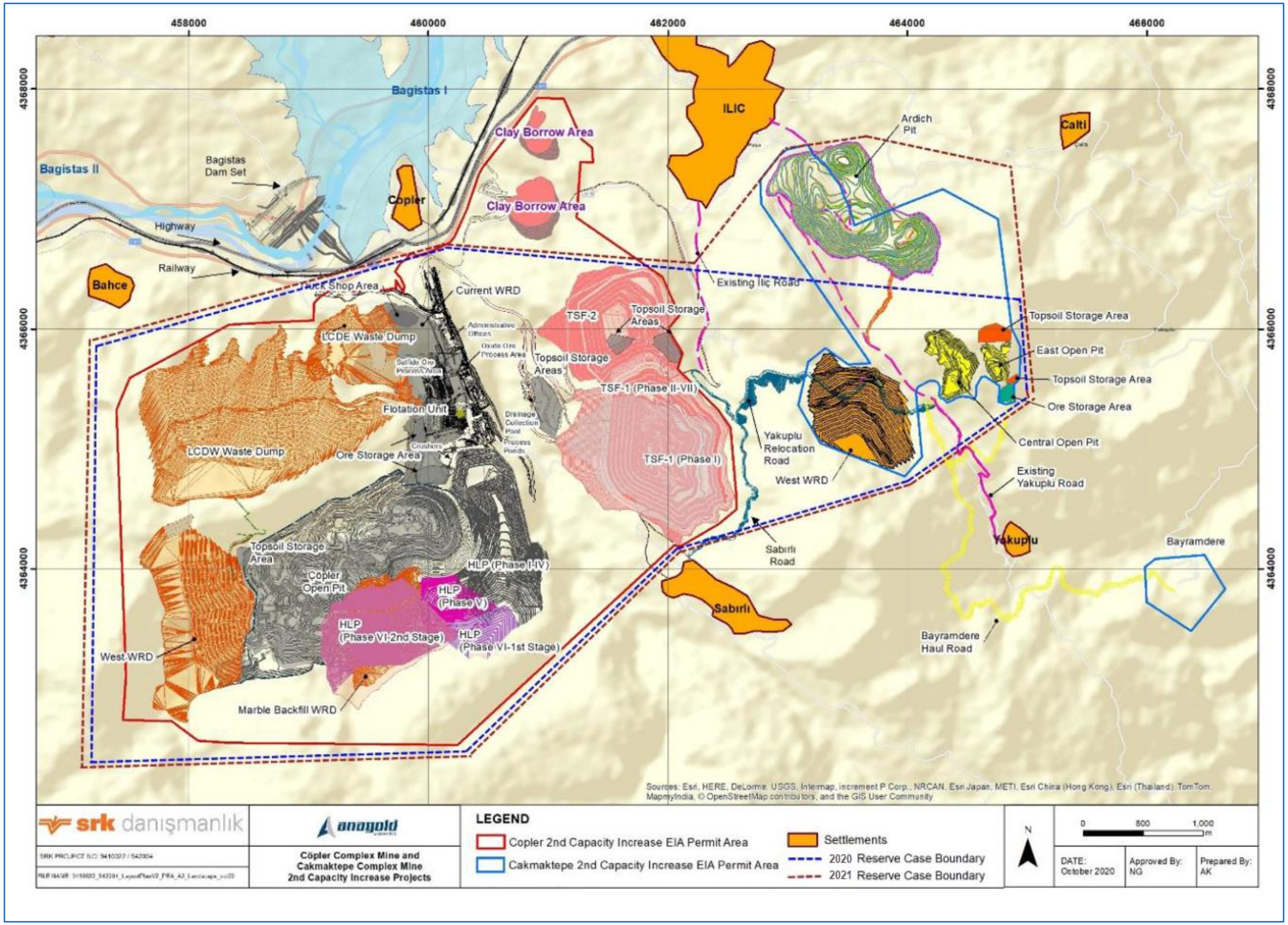
The WRDs will generally consist of 15 m tall lifts deposited at the waste material's angle of-repose of approximately 1.33H:1V. The typical bench width will be 17 m and 15 m wide haul roads will be used to construct the WRDs. The WRDs will have overall slopes ranging from 2.5H:1V to 2.6H:1V.

In February 2014, Golder completed an evaluation of the geotechnical stability of the four WRD designs (Golder, 2014a), later updated in May 2015 (Golder, 2015b) to account for the updated material properties developed by Golder during the pit slope optimisation study and the updated waste dump designs and layouts developed by Anagold. Six of the most critical cross-sections were evaluated to determine the minimum Factor of Safety (FOS) for the proposed waste dumps. The sections were aligned to pass through the highest part of the waste piles, the steepest waste pile slopes, and the steepest foundation grades.

In addition to static stability analyses, pseudo-static stability analyses were performed to account for seismic loading conditions for the WRDs. The pseudo-static analyses were conducted based on the procedure proposed by Hynes-Griffin and Franklin (1984) in which a horizontal acceleration equal to 50% of the peak ground acceleration at bedrock is applied to the model. The design criteria peak ground acceleration is 0.30 g for the magnitude 7.0 operating basis earthquake (OBE). Therefore, a horizontal pseudo-static acceleration of 0.15 g was applied to the WRD sections in the seismic stability analyses.

The results of the stability analysis are summarised in Table 16.12.

Figure 16.8 CDMP21TR Reserve Case Site Plan



Anagold, 2022

Table 16.12 Waste Rock Dump (WRD) Design Factor of Safety (FOS)

WRD	Section	Loading Condition	Failure Surface Location	Minimum Computed FOS
Lower Çöpler East	A	Static	Shallow	1.4
		Pseudo-static		1.1
		Static	Deep	1.9
		Pseudo-static		1.3
	B	Static	Shallow	1.7
		Pseudo-static		1.3
		Static	Deep	1.9
		Pseudo-static		1.3
Lower Çöpler West	C	Static	Shallow	1.7
		Pseudo-static		1.3
		Static	Deep	1.9
		Pseudo-static		1.3
	D	Static	Shallow	1.6
		Pseudo-static		1.2
		Static	Deep	1.8
		Pseudo-static		1.3
West Çöpler	E	Static	Shallow	1.6
		Pseudo-static		1.1
		Static	Deep	1.9
		Pseudo-static		1.3
	F	Static	Shallow	1.6
		Pseudo-static		1.2
		Static	Deep	2.0
		Pseudo-static		1.4

The Lower Çöpler East WRD facility will be constructed over a portion of the existing North-east WRD. Foundation conditions underlying the existing North-east WRD and the proposed Lower Çöpler East facility consist of Munzur Limestone. Minimum computed factors of safety for the Lower Çöpler East facility are 1.4 and 1.1 for static and seismic loading conditions, respectively.

The Lower Çöpler West WRD facility will be founded on Munzur limestone. Limit equilibrium stability analyses indicate minimum computed FOS of 1.6 and 1.2 for static and seismic loading conditions, respectively (Golder, 2015b).

The West WRD is to be constructed adjacent to the Çöpler open pit and will be founded on Munzur Formation limestone and metasediment with sporadic diorite intrusions. Minimum computed FOS are 1.9 and 1.3 for static and seismic loading conditions, respectively.

16.2.4.2 Waste Rock Geochemical Review

Anagold mines and monitors the waste rock types to determine PAF and NAF material according to the Çöpler waste rock management plan to ensure proper disposal of PAF material as it is encountered during the ore control process. SRK established the criteria for identifying PAF and NAF material as shown in Table 16.13.

Table 16.13 Waste Rock Geochemical Classification

Lithology	Sulfide Sulfur (SS%) Cut-off Grade	Waste Rock Groups	Descriptions
Diorite	0.8	PAF/High-sulfide diorite	Diorite with SS ⩾0.8%
		NAF/Low-sulfide diorite	Diorite SS <0.8%
Metasediment	0.8	PAF/High-sulfide MTS	Metasediment with SS ⩾0.8%
		NAF/Low-sulfide MTS	Metasediment with SS <0.8%
Limestone / Marble	2	High-sulfide LMS	Limestone with SS ⩾2%.
		Low-sulfide LMS	Limestone with SS <2%.
Gossan	–	Gossan – NAF	All Gossan unit
MnOx	–	MnOx – NAF	All MnOx unit
Massive Pyrite	–	Massive Pyrite – PAF	All Massive Pyrite unit

In September 2015, SRK completed a geochemical impact assessment for the Çöpler WRD facilities. The key findings from the SRK report suggests that all WRD facilities at Çöpler, except one, have a neutralising potential (NP) to acid potential (AP) ratio of greater than 20:1; indicating that the Çöpler material has excellent neutralisation capacity for ARD. The one exception to this was the West WRD which was estimated to have a NP:AP ratio 1:3. It was recommended that Anagold optimise the WRD construction sequencing in order to take advantage of the neutralisation potential of the other WRD facilities by blending higher quantities of NAF material into the West WRD. Anagold anticipates that this will be a readily achievable solution that will not add any additional costs to the Project.

A series of waste rock samples representing the LOM distribution were tested by SRK in order to measure the immediate reactivity, future acid potential, and long-term acid potential of the waste rock.

Regarding immediate reactivity, a paste pH test was conducted that resulted in all samples generating near-neutral and slightly alkaline paste pH.

Regarding future acid potential, a large majority of all samples taken reside above the NP:AP 1:1 boundary. The remainder of the samples that fall below the 1:1 boundary are extremely close to the 1:1 boundary and should only pose a minimal risk to ARD generation. In terms of long-term acid potential, only two samples registered below the 1:1 NP:AP ratio.

16.2.5 Ore Stockpiles, Rehandle and Blending

Oxide and sulfide ore are processed through separate crushing circuits.

Oxide ore that is unable to be directly dumped into the crushing circuit is placed on the appropriate stockpile for processing at a later time. Oxide ore is typically segregated according to clay content and grade. The processing engineer determines the desired blend on a daily basis in order to maintain a consistent feed grade and rock type blend going to the heap leach pad.

All sulfide ore is currently placed in one of three primary stockpiles: High-grade, medium-grade, and low-grade. Sulfide ore is directed to the primary stockpiles or to the crusher pad. There is no allowance for material to be directly dumped into the sulfide crushing circuit. All material is rehandled by a loader from the crushing pad into the crushing circuit.

The following Au grade bin assumptions were used for the Mineral Reserves:

- High-grade Au >4.0 g/t Au
- Medium-grade Au 2.5–4.0 g/t Au
- Low-grade Au 1.05–2.5 g/t Au

Currently site typically experience a lack of sulfide sulfur (SS) feed to the POX, requiring additional cost to run the POX plant. The flotation plant was designed to upgrade (increase) the SS feed into the POX circuit. For the POX autoclave to operate autogenously, SS feed must be above 10.20 tph and less than 13.75 tph to achieve target oxidation with current oxygen availability. If the SS feed rate is too high, then the feed to the plant will need to be reduced until the POX SS feed rate is less than 13.75 tph limit. Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph sulfide sulfur to be treated. This oxygen utilisation efficiency along with increased oxygen availability is upside to the CDMP21TR Reserve Case.

Plant feed will therefore need to be blended to achieve the target SS feed range of 10.20–13.75 tph into POX.

To blend on SS feed, new grade control protocols have been developed and implemented on-site. Site grade control is currently being done on Au and SS grades to aid in achieving the ideal range for SS feed into the plant and assist with the development of a new stockpile strategy.

The following SS grade bin assumptions were used for the Mineral Reserves inside each Au grade bin:

- High-grade SS >4.8% SS
- Medium-grade SS 3.2% to 4.8% SS
- Low-grade SS <3.2% SS

The effectiveness of these new grade bins in controlling the SS blend will need to be monitored on an ongoing basis as the plant matures and adjustments to the grade bin parameters (and size of stockpiles) may be required. This work will need to continue as the mine progresses and new mining areas are included.

The smallest parcel size for plant feed considered for the Mineral Reserves was one month.

The operation will need to be in control of the plant feed blend at a more granular level than was modelled for the Mineral Reserves. If maintaining a plant feed blend at a more granular level was found to be problematic, there are several measures that site could implement to manage any short-term and longer term concerns:

- Mine working areas

  Given the relatively small size of the mining fleet, the number of active mining working areas could be increased, increasing mining selectivity, and therefore improving the blending capacity from the mine.

- Stockpile size

  The size of stockpiles could be adjusted to reduce feed impacts from short-term fluctuations coming from the mine.

- Mining rate

  Given the current site contract mining arrangement, site could ramp up the mining rates to reach sufficient material (of the required type) to maintain the required blend.

- Variation of grade bins

  Grade bin designations could be adjusted to have better control of the grade bands that are causing problems in the plant feed blend.

16.2.6 Grade Control

All ore control operations are managed by Anagold technical staff. Anagold maintains an on-site laboratory with the capacity to assay an average of 600 blasthole samples per day.

Prior to sampling, blastholes are identified as ‘potential ore’ (oxide or sulfide) or ‘potential waste’ (oxide or sulfide) based on grade control data from the bench above and the mining model prediction. A 10 m outside buffer is then applied to the potential ore areas to ensure appropriate sampling density. All potential ore blastholes are sampled for AuFA (fire assay for Au). Approximately 50% of potential ore blastholes are sampled for AuCN (cyanide soluble Au), total carbon, and total sulfur. Additionally, all potential sulfide ore blastholes are sampled for SS. Approximately 25% of potential waste blastholes are sampled for AuFA, AuCN, total carbon, and total sulfur.

Sampling of the blasthole drill cuttings is performed according to the formal procedure by using a sample scoop to extract a complete cross-section of the cutting pile. The sampled cuttings are deposited into a canvas bag, which is labelled with the drillhole identifier (ID) and with a laboratory information management system (LIMS) bar code tag inserted into the bag with the cuttings. Sample bags are sealed and sent to the on-site laboratory for analysis. The sample scoop is cleaned prior to collecting each sample to avoid contamination between samples.

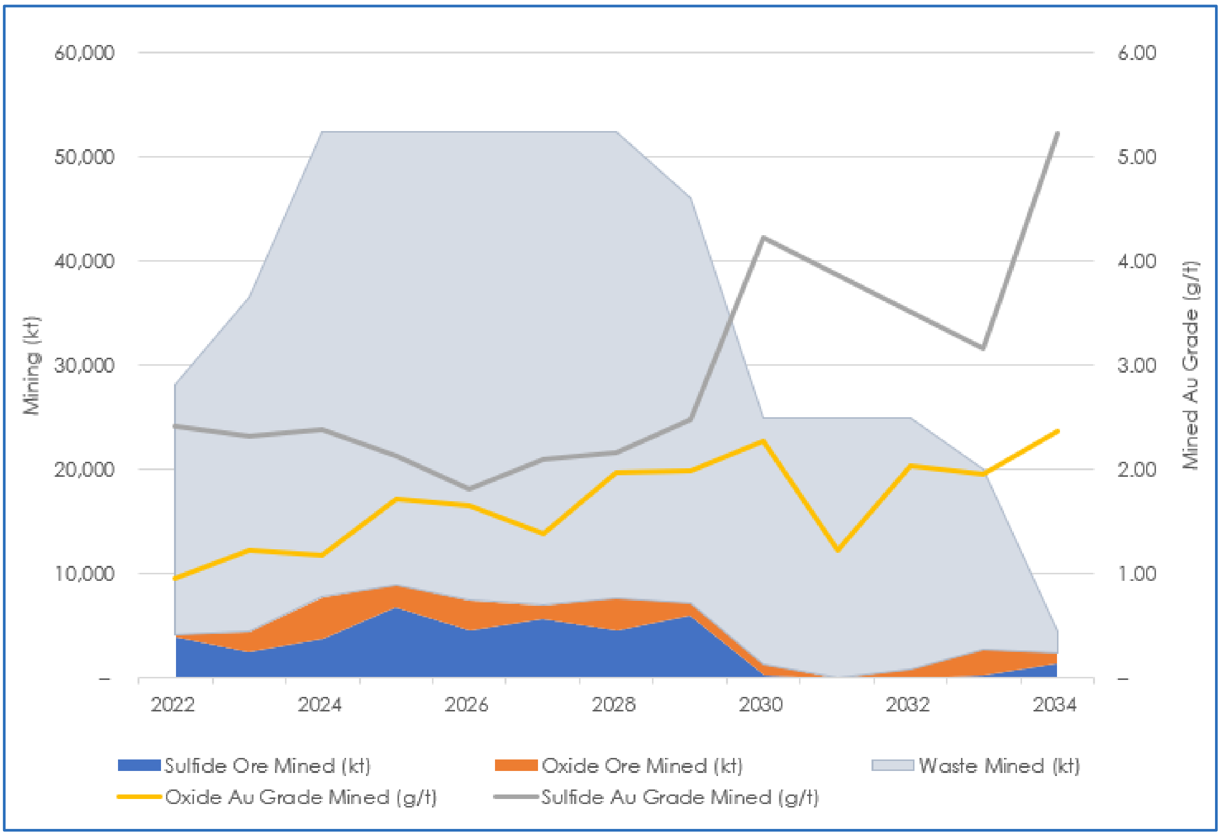
Assay results are uploaded to the ore control database with reference to each specific drillhole ID. The assay results are then estimated into a cell model with parent cell sizes of 3 m x 3 m x 5 m using ordinary kriging (OK) to estimate ore grade and type. The ore control geologist will then digitise mining shapes with a minimum width of 3 m (to match the SMU) and minimum tonnage of 500 t. These mining shapes are then sent to the survey group for layout in the mine using colour coded flagging under the supervision of the ore control geologist.

To effectively blend the sulfide feed on SS content, new grade control protocols were developed and implemented on-site in 2021. They are undergoing further review to optimise and improve production.

16.3 Mine Production Schedule

The Reserve Case has examined production from three open pit mining locations at the Çöpler mine, the Çöpler deposit, the Ardich deposit and the Çakmaktepe deposit. The Çakmaktepe pit, which contains only oxide ore, is almost exhausted. Anagold has prepared the open pit production schedules. The case adopted for the Reserve Case is based on Mineral Reserves only and does not include Inferred Mineral Resources. Figure 16.9 shows total mine production and the tonnages and grades for each ore type.

Figure 16.9 CDMP21TR Reserve Case Mining Production



OreWin, 2022

16.3.1 Scheduling Assumptions

The following scheduling methodology was used to balance mine, mill, and stockpile quantities:

- Heap leach:
  - Oxide ore is not limited by processing capacity.
  - Oxide ore that is unable to be directly dumped into the oxide crushing circuit is placed in the appropriate stockpile for future processing.
  - Oxide ore is segregated dependent on clay content and average grade.
- Sulfide plant:
  - All sulfide ore is segregated into one of three primary gold stockpiles: high-grade, medium-grade, and low-grade, which are each further split by SS grade.
  - Existing stockpiles are mined at the average grade of each stockpile.
  - All material is rehandled by a loader from the crushing pad into the crushing circuit (no direct tipping).

– The flotation circuit was commissioned in December 2021 with circuit ramp up and a transition to stable operations expected in early-2022.
– Plant throughput capacity is calculated from the available mill hours and varies by material type.

- The production schedules are based on Proven and Probable Mineral Reserves only. No Inferred Mineral Resources were used.
- The open pit schedules were based on mining inventories by bench reported within the pit stages.
- Low-grade stockpiling was used to balance the mining rate where necessary.

16.3.2 Production Schedule

The input assumptions for Reserve Case were adjusted based on current mine and production performances including throughput rates and recoveries.

All throughput rates are reported inclusive of all availability and utilisation factors on a calendar year. Total mine production is limited to an annual average of 22.5 Mtpa. The throughput assumptions are supported by current mining rates including productivity allowances for winter and summer conditions. Mining rates are limited based on vertical advance and bench configuration in order to ensure that the schedule is achievable. Production is not limited by the mining rate and increases in rate would be possible to bring forward oxide ore or increase stockpiling to bring higher grade feed to the sulfide plant.

Mining in the Reserve Case is completed in 2034, after which the sulfide plant is fed from stockpiles.

The objective of the production schedule is to maximise the early cash flow by delaying costs and bringing revenue forward with ore feed to meet concentrator throughput capacity. Considerations for the LOM scheduling include:

- Ensuring continuous ore supply to the concentrator by delivering the highest value ore first and meeting physical mining and milling hours capacity constraints.
- Achieving excavator productivities and sinking rates to deliver ore at maximum utilisation of milling hours available at the concentrator.
- Maximising annual utilisation hours for the mine loading equipment.
- Maintaining a balance of ore throughput rates (material types) and mill cut-off grades that allows milling hours to be maximised.

The mine schedule incorporates strategic stockpiling considerations by optimising the number of excavators on the benches of early phases, increasing the opportunity to raise mill cut-off grades. This leads to stockpiling medium-grade and low-grade material and sending higher grade ore to the mill sooner. The open pit total movement is shown in Table 16.14.

Table 16.14 CDMP21TR Reserve Case Mining Schedule

Mining Year	Total Tonnes (kt)	Oxide Ore				Sulfide Ore				Waste Tonnes (kt)
		Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Tonnes (kt)	Au (g/t)	Ag (g/t)	SS (%)	
2022	28,065	353	0.95	5.74	0.11	3,884	2.41	4.27	–	23,828
2023	36,516	1,920	1.22	9.43	0.06	2,582	2.32	8.73	–	32,014
2024	52,435	3,980	1.18	2.04	0.00	3,814	2.39	2.33	–	44,641
2025	52,375	2,196	1.72	2.04	0.00	6,817	2.13	5.32	–	43,362
2026	52,331	2,880	1.66	2.05	0.01	4,607	1.81	3.93	–	44,843
2027	52,340	1,266	1.39	2.04	0.01	5,745	2.09	4.47	–	45,329
2028	52,389	3,136	1.97	2.72	0.01	4,614	2.16	4.70	–	44,639
2029	46,088	1,230	1.99	5.57	0.04	5,988	2.48	6.37	–	38,870
2030	25,000	992	2.27	2.04	–	392	4.23	4.23	–	23,616
2031	25,000	30	1.23	2.04	–	–	–	–	–	24,970
2032	25,000	918	2.03	2.04	–	–	–	–	–	24,082
2033	20,000	2,504	1.95	2.04	–	338	3.16	4.23	–	17,158
2034	4,496	1,083	2.37	2.04	–	1,384	5.22	4.23	–	2,029
Total	**472,035**	**22,486**	**1.69**	**3.02**	**0.01**	**40,167**	**2.34**	**4.90**	**–**	**409,382**

Table shows mining schedule does not show processing or existing stockpile rehandle

16.3.3 Processing Schedule

The processing schedule was balanced to meet the maximum build rates for the oxide heap leach pads, or available mill hours for the sulfide plant.

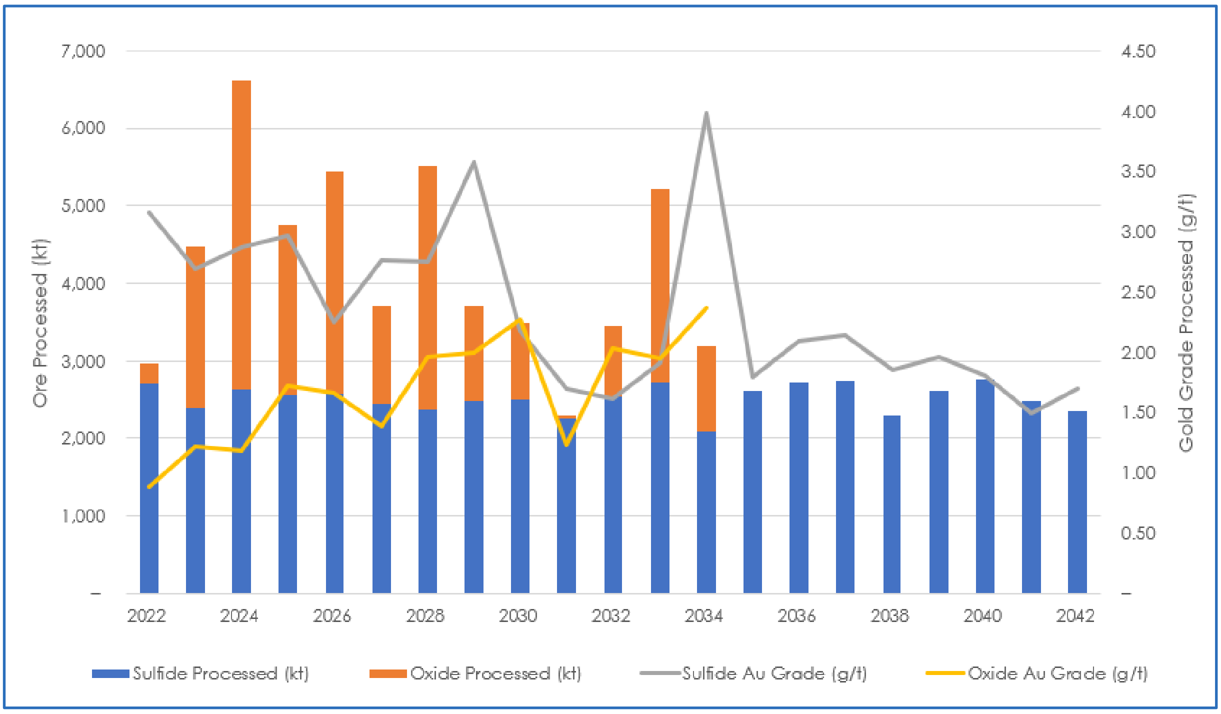
Sulfide ore production throughputs are limited dependent on ore tonnage, SS tonnage, and carbonate content, (expressed as C). The sulfide plant crusher / grinding circuit is limited to 400 tph, while the limitations on SS tonnage exist due to the consumption of oxygen by SS in the POX circuit and carbonate content to maintain an operable acid balance through the acidulation and POX circuits. The process facilities are limited by the amount of oxygen that can be provided to the POX process. Based on current performance, high-SS is unlikely to be a problem, and any higher material would be blended down using low-SS material. The carbonate:SS ratio will potentially be an issue with declining SS grades. The main issue currently appears to be a lack of SS in the feed, forming the justification for the flotation circuit. The flotation circuit upgrades the SS content into the autoclave feed and rejects carbonate.

In order to target the highest value material, the sulfide production schedule is therefore required to target the highest value material, while also balancing the plant throughput rates and required range of sulfide sulfur into the autoclave.

The Reserve Case production is predominantly from sulfide ore. The oxide heap leach and sulfide plant processing schedules feed type, Au grade, and gold production are shown in Figure 16.10. Gold production and recovery is shown in Figure 16.11. The annual production schedule is in Table 16.15.

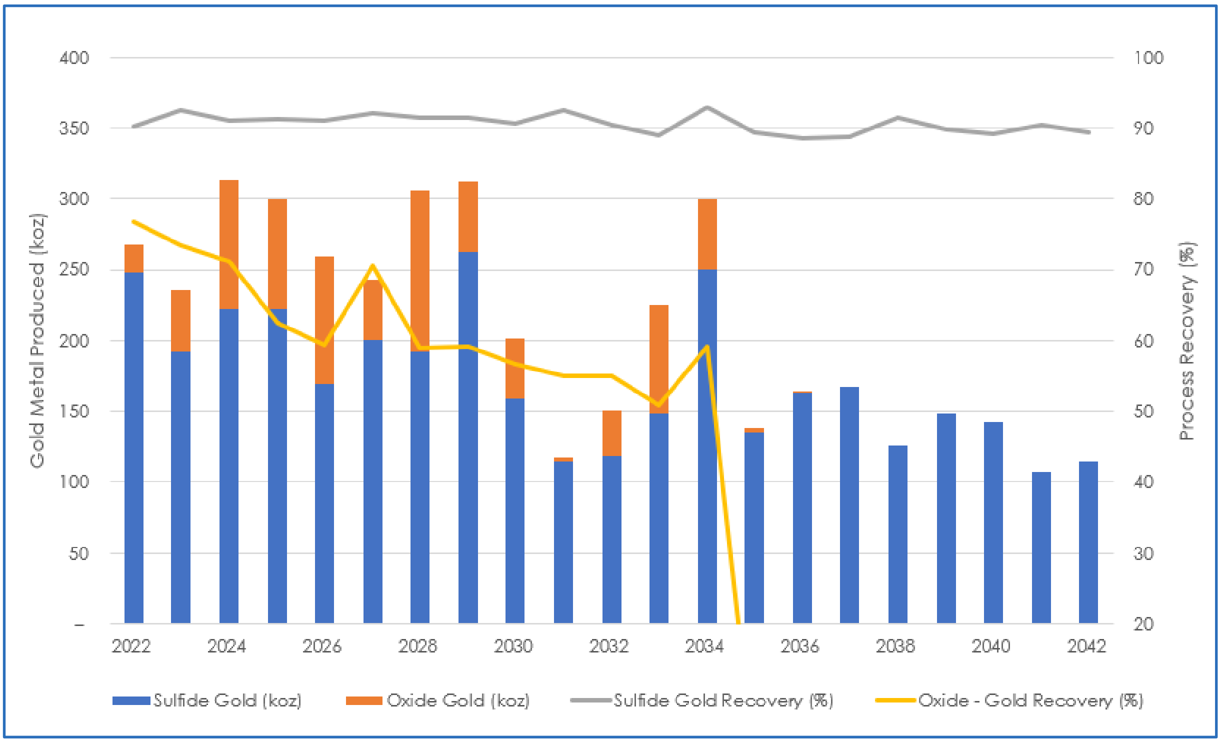
The Reserve Case production includes 22.5 Mt at 1.69 g/t Au oxide ore processed by heap leaching and 52.9 Mt at 2.33 g/t Au processed in the sulfide plant. Total ore production is 75.4 Mt at 2.14 g/t Au. Total gold production is 4.4 Moz. Mining at the Çöpler pit is completed in 2029 and at Ardich in 2034. Oxide heap leach stacking is completed in in 2034, while sulfide processing will continue from stockpiles until 2042. Sulfide processing will continue from stockpiles until 2042 for a 21-year mine life. The production schedule is for the period 1 January 2022 through 2042.

Figure 16.10 CDMP21TR Reserve Case Processing Schedule



OreWin, 2022

Figure 16.11 CDMP21TR Reserve Case Gold Production and Recovery



OreWin, 2022

Table 16.15 CDMP21TR Reserve Case Production Schedule

Description	Units	Total	Year																				
			2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Heap Leach Stacked	**kt**	**22,557**	**263**	**2,080**	**3,980**	**2,196**	**2,880**	**1,266**	**3,136**	**1,230**	**992**	**30**	**918**	**2,504**	**1,083**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**
Au Feed Grade	g/t	**1.69**	0.88	1.22	1.18	1.72	1.66	1.39	1.97	1.99	2.27	1.23	2.03	1.95	2.37	–	–	–	–	–	–	–	–
Ag Feed Grade	g/t	**3.04**	4.90	9.41	2.04	2.04	2.05	2.04	2.72	5.57	2.04	2.04	2.04	2.04	2.04	–	–	–	–	–	–	–	–
Cu Feed Grade	%	**0.01**	0.10	0.07	0.00	0.00	0.01	0.01	0.01	0.04	–	–	–	–	–	–	–	–	–	–	–	–	–
Sulfide Plant Feed	**kt**	**52,892**	**2,708**	**2,395**	**2,635**	**2,551**	**2,569**	**2,442**	**2,378**	**2,490**	**2,504**	**2,269**	**2,538**	**2,718**	**2,101**	**2,607**	**2,730**	**2,739**	**2,305**	**2,623**	**2,759**	**2,476**	**2,356**
Au Feed Grade	g/t	**2.33**	3.16	2.69	2.87	2.97	2.25	2.77	2.75	3.58	2.18	1.70	1.61	1.91	3.99	1.80	2.10	2.14	1.86	1.96	1.80	1.49	1.69
Ag Feed Grade	g/t	**3.89**	3.82	4.71	1.99	3.96	3.67	4.72	4.48	7.24	4.89	4.80	5.39	4.11	4.34	2.14	1.33	0.68	3.08	5.99	4.13	4.49	2.34
Total Feed	**kt**	**75,448**	**2,970**	**4,475**	**6,614**	**4,747**	**5,449**	**3,709**	**5,514**	**3,720**	**3,495**	**2,299**	**3,456**	**5,222**	**3,184**	**2,607**	**2,730**	**2,739**	**2,305**	**2,623**	**2,759**	**2,476**	**2,356**
Total Metal Recovered																							
Au Recovered	koz	**4,369**	268	238	318	302	264	245	310	314	203	118	152	228	301	139	164	167	126	148	143	107	115
Ag Recovered	koz	**663**	21	156	55	38	43	28	66	74	26	12	23	38	22	5	3	2	7	15	11	11	5
Cu Recovered	klb	**161**	24	87	12	3	17	1	5	12	1	0	–	–	–	–	–	–	–	–	–	–	–

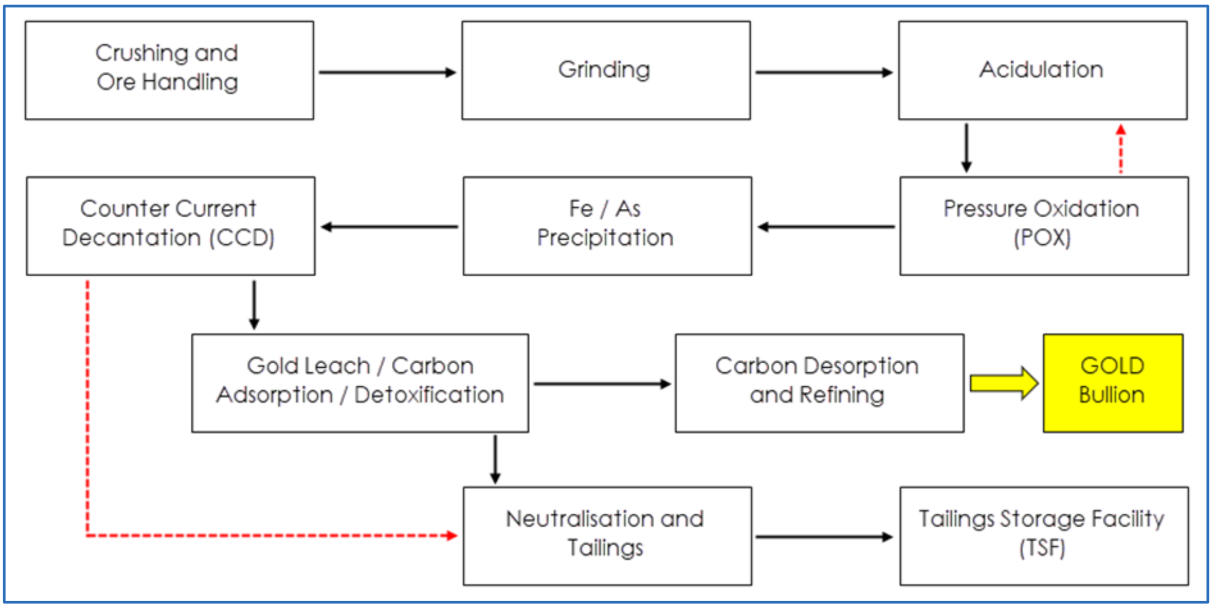
17 RECOVERY METHODS

17.1 Sulfide Ore Processing

The sulfide plant commenced commissioning in Q4'18. The basic flow sheet is shown in Figure 17.1 and comprises:

- Crushing and ore handling
- Grinding
- Acidulation
- Pressure oxidation
- Iron / arsenic precipitation
- Counter current decantation (CCD)
- Gold leach, carbon adsorption, and detoxification
- Carbon desorption and refining
- Neutralisation and tailings
- Tailings storage facility (TSF)

Figure 17.1 Çöpler Process Flow Sheet for Sulfide Plant

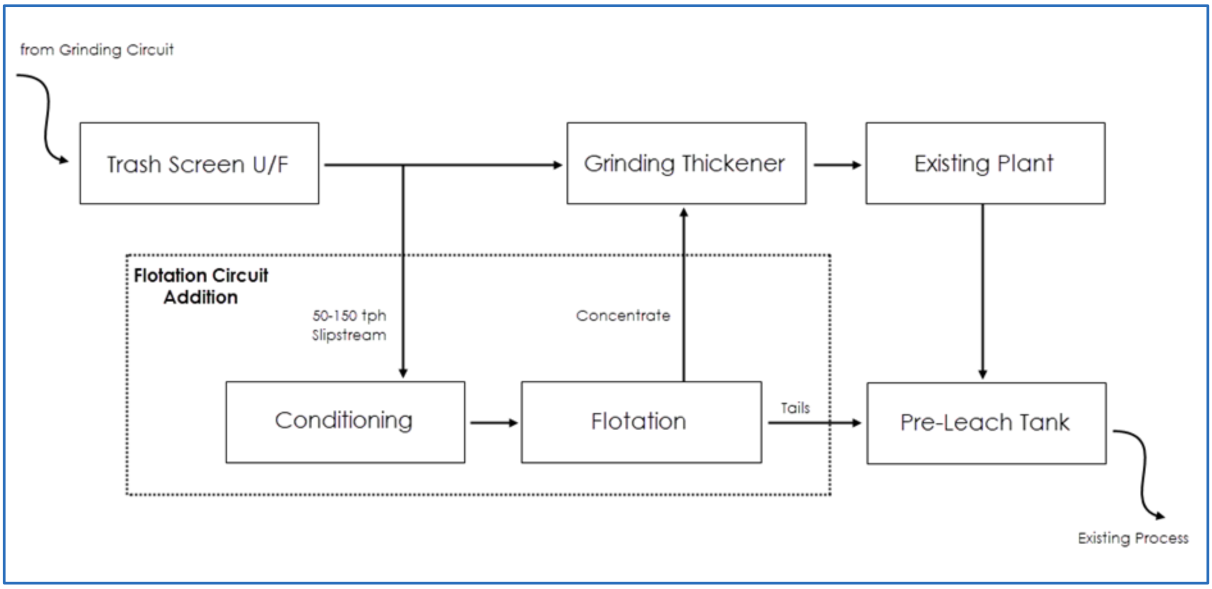


Anagold, 2020

The incorporation of a flotation circuit into the existing sulfide plant to upgrade sulfide sulfur (SS) to fully utilise grinding and pressure oxidation (POX) autoclave capacity has completed construction and commenced commissioning on ore in January 2022. This addition to the sulfide plant is incorporated between grinding and acidulation, as shown in Figure 17.2, by taking a bleed / slip stream from the grinding thickener feed, floating sulfides, and returning the sulfide concentrate to the grinding thickener to be combined with direct feed. Gold not recovered to flotation concentrate will report with flotation tails to the gold leaching and recovery circuit and combined with material process through the POX autoclave circuit to recover gold.

The flotation circuit will also reject carbonates to flotation tails, bypassing acidulation and POX, providing additional benefits in the acid balance through POX.

Figure 17.2 Flotation Block Flow Diagram



Anagold, 2020

The existing sulfide circuit, before the addition of flotation, has demonstrated additional latent capacity in throughput controlling sections of the circuit, crushing/grinding and autoclaves. The incorporation of flotation will allow the POX autoclaves to maximise throughput and sulfide sulfur oxidation capacity, utilising latent capacity in the process plant, in particular the grinding and pressure oxidation circuits. Fully utilising this latent capacity with the addition of a small flotation plant allows with minimal capital cost the increase in overall plant throughput.

The throughput from crushing and grinding was designed with a nominal volumetric capacity of 306 tph will increase up to a maximum of 400 tph. Additionally, the POX autoclave circuit has demonstrated it can process up to a long-term average maximum of 280 tph feed (two autoclave operation) and 13.75 tph sulfide sulfur, compared to design of 245 tph and 12.5 tph respectively. The limit of 13.75 tph sulfide sulfur is dictated by the capacity of the oxygen supply to effect oxidation of the sulfides, design 96%.

The flotation plant feed rate is variable between 50–150 tph based on sulfide sulfur feed grade and the oxidation capacity of the POX autoclaves to oxidise sulfides. Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph sulfide sulfur to be treated. Alternatively, increased autoclave throughput with reduced sulfide oxidation is possible, with a resultant reduction in overall gold recovery. This oxygen utilisation efficiency, along with increased oxygen availability, is upside to the CDMP21TR Reserve Case.

17.1.1 Sulfide Plant Performance

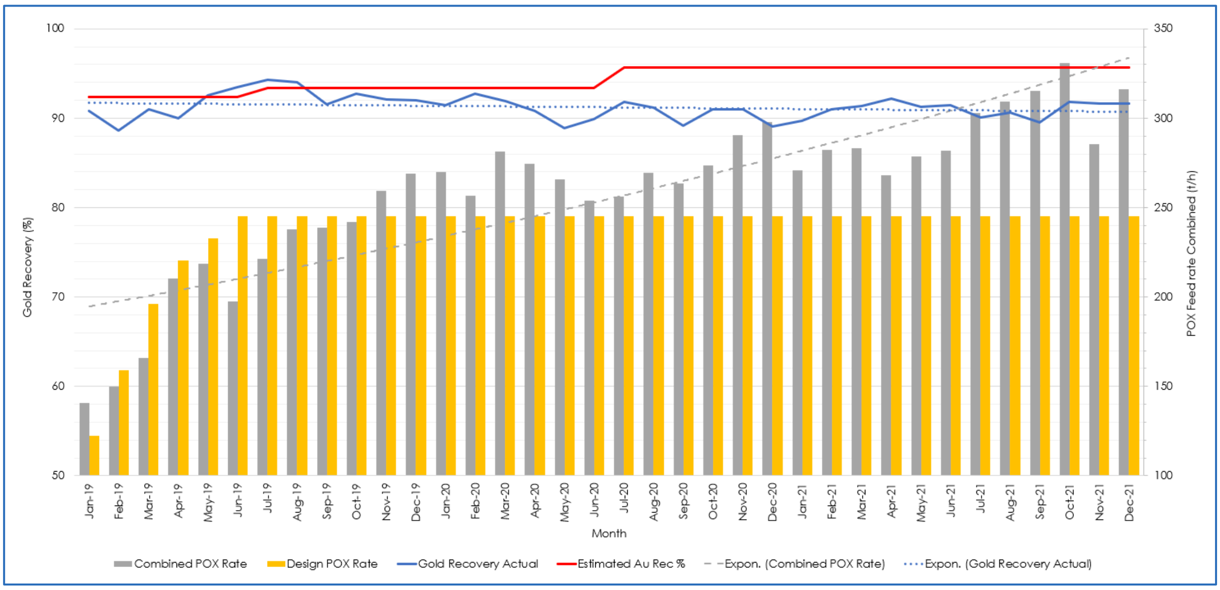
The sulfide plant commenced commissioning in Q4'18.

The operating performance is summarised in Figure 17.3 for throughput and recovery against the design for the period 2020 and 2021.

Since completing ramp-up of the sulfide plant in June 2020, POX throughput has progressively improved to exceed design up to a monthly average peak of 330 tph and at the maximum SS of 13.7 tph. The gold recovery has remained at around 91%, lower than design, with the tailings grade remaining stable between 0.25–0.30 g/t Au.

Further improvements have been implemented during 2020–2021. The includes the installation of oxygen to leach to supplement air to maintain sufficient oxygen levels for gold leaching has led to improved recoveries.

Figure 17.3 Gold Recovery and Throughput Comparison



Anagold, 2020

17.1.2 Sulfide Plant Description

A detailed sulfide flow sheet is shown in Figure 17.4. The following description of the sulfide plant includes the existing operating circuits and the flotation circuit.

17.1.2.1 Crushing and Ore Handling

Haul trucks from the mine tip ore onto designated stockpile fingers. The ore is withdrawn from stockpiles by front end loader (FEL) and deposited into the run-of-mine (ROM) dump hopper. A static grizzly is fitted to the top of the ROM bin to remove coarse oversize.

ROM ore is reclaimed from the bin by the sizer apron feeder, which discharges material into the mineral sizer. The sizer is a tooth roll unit which crushes the ore from a feed top size of 500 mm to a nominal top size of 250 mm. Discharge from the sizer drops down a chute onto the sizer discharge conveyor.

The sizer teeth are configured in a manner to direct oversize rocks to one end where they pass through a spring-loaded oversize rejection gate and fall to a reject bunker. The crushed product is carried by the sizer product conveyor to the semi-autogenous grind (SAG) mill feed conveyor. The SAG mill feed conveyor has a belt scale to monitor the ore flow to the SAG mill and this information is used to control the sizer apron feeder speed.

17.1.2.2 Grinding

The SAG milling stage consists of a high aspect SAG mill with water cannon pebble recycle. The SAG mill grinds the crushed ore to produce a discharge particle size distribution P_{80} of approximately 1,400 µm.

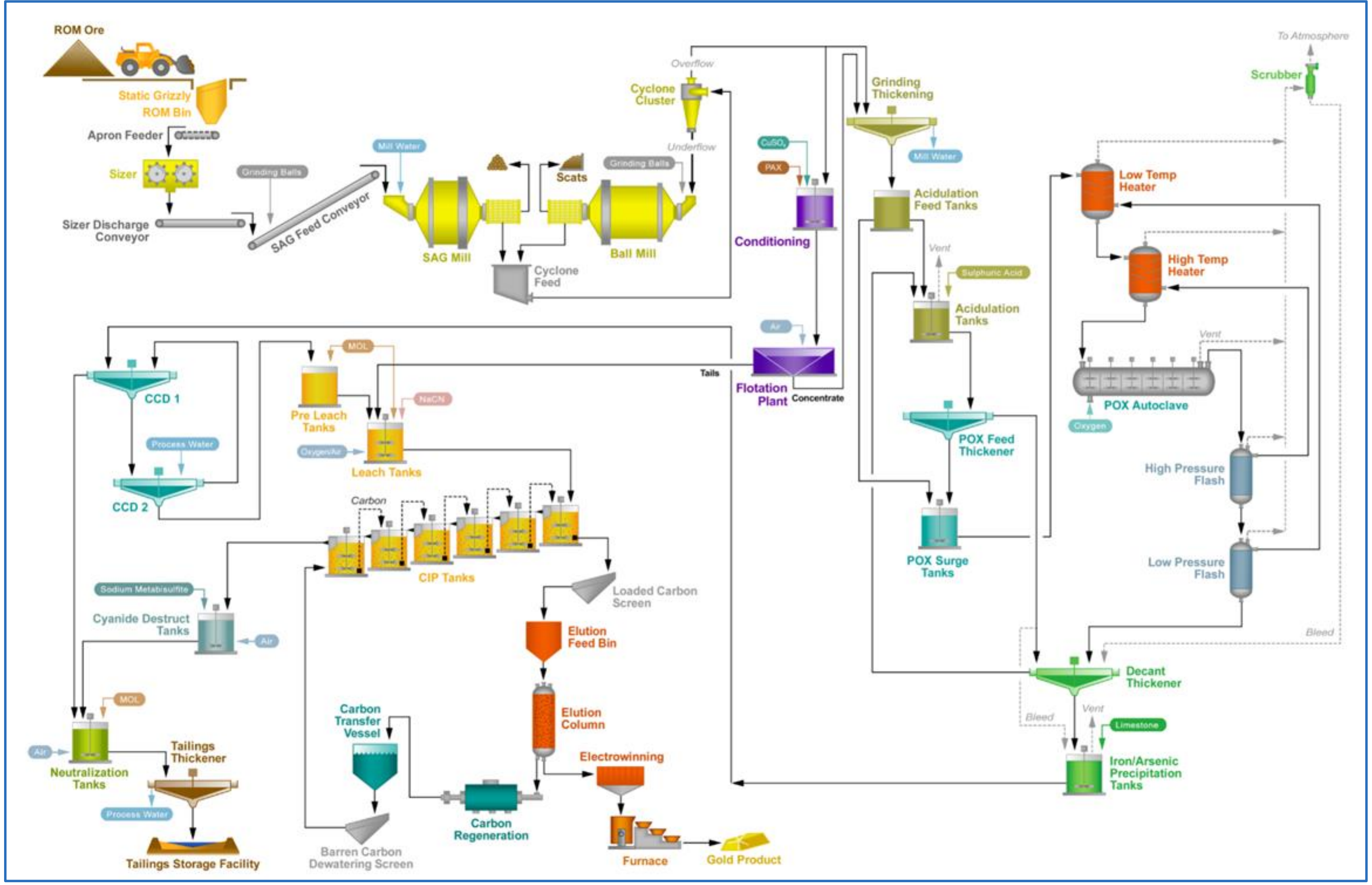
Large ore particles are retained in the SAG mill by the internal SAG discharge grate. Particles too large for ball milling are retained as oversize on the SAG mill trommel screen and this oversize is washed by trommel sprays. The trommel screen oversize is either projected back into the SAG mill using a high-pressure water cannon or rejected via a conveyor. Slurry that passes through the trommel screen discharges into the grinding cyclone feed pump box where it mixes with the ball mill discharge slurry and density control water.

Slurry collected in the grinding cyclone feed pump box from the SAG mill and ball mill is fed to the grinding cyclone cluster. The cyclones produce an overflow product with a P_{80} of 100 µm, which is screened to remove any trash (organic material, etc.) by the grinding trash screen. Coarse particles report to cyclone underflow, which is returned to the ball mill for further size reduction until it is fine enough to report to cyclone overflow and leave the circuit.

The slurry product from the grinding circuit, trash screen undersize, is currently thickened in a high rate thickener and excess water reports to the thickener overflow for immediate re-use within the grinding circuit. The thickened slurry discharging from the thickener underflow is pumped to the grinding thickener underflow storage tanks.

To provide for the flotation circuit, a portion of the trash screen undersize, dependent on POX autoclave sulfide sulfur requirements, will be diverted to the flotation circuit where the remaining slurry continues to the thickener.

Figure 17.4 Process Flow Sheet for Sulfide Plant



Anagold, 2020

17.1.2.3 Flotation

A portion of the grinding trash screen undersize will be diverted to the flotation circuit and pumped to the conditioning tanks. This proportion, between 50 tph and 150 tph, will depend on SS feed grade and POX autoclave SS requirements. The flotation circuit can operate as a single or dual train, each train will have a maximum throughput of 75 tph.

The flotation plant consists of two equally sized conditioning tanks, in series, for copper sulfate, if required, and potassium amyl xanthate (PAX) conditioning with a nominal residence time of seven minutes each tank. From conditioning, the slurry is pumped to two equally sized flotation trains consisting of six 50 m^3 tank cells with a residence time of nominally 60 minutes at maximum throughput (75 t/h each). Frother dosing and supplemental collector dosing will occur down the trains in every second cell. The plant is designed to handle high mass pull to maximise sulfide recovery, with preference to high recovery over high selectivity.

The flotation concentrate is pumped to the grinding thickener feed mixing with slurry directly from the grinding circuit upgrading the sulfide sulfur material fed to the acidulation and POX circuit. The flotation tail is pumped to the gold leach tanks for recovery of gold present in the non-sulfidic portions of the ore.

17.1.2.4 Acidulation

The grinding thickener underflow storage tanks provide process surge and effectively decouple the upstream crushing, grinding and flotation, when operating, circuits from the downstream hydrometallurgical circuit. If the acidulation feed tanks reach their high-level limit then ore feed to the upstream circuits will be stopped. If the tanks are approaching their low-level limit then the upstream circuit feed rate can be increased to compensate.

The tanks are agitated for solids suspension and mixing and have a total residence time of 12-hours. Agitation achieves short term blending of the incoming feed from the upstream circuits and this provides a relatively slow-changing feed composition to the downstream hydrometallurgical circuit. Antiscalant can be added to these tanks if necessary, to reduce scale build-up in the downstream acidulation circuit.

The acidulation circuit uses recycled solution, containing free acid, from the decant thickener to leach the carbonate minerals in the ore. Supplemental concentrated sulfuric acid can also be added, when required, to meet total acid addition demand. The total acid addition targets nearly-complete destruction of acid soluble carbonates in the acidulation tanks. Acidulation is conducted in two reaction tanks. The acidulation tanks are agitated to disperse the slurry, acid and decant thickener overflow recycle throughout the tank and ensure the carbonates in the ore react with the acid in solution.

Depending on the ore type being processed the slurry from the grinding thickener underflow storage tanks is split between acidulation and the POX feed tanks. The proportion of this split is determined by how much carbonate in the feed material requires destruction to achieve the target of 22.5 g/L free acid content in the POX autoclave discharge slurry. This free acid level favours the formation of an iron mineral reaction product which exhibits better settling behaviour in downstream thickeners (hematite favoured over jarosite), while also reducing the potential for excessive CO_2 gas evolution and gypsum scaling in the POX autoclaves.

Additional concentrated sulfuric acid is added if required to maintain the targeted acid soluble carbonate destruction in the acidulation tanks. When there are low-carbonate levels in the feed, and little or no acidulation is required, POX feed thickener overflow solution is recycled to the acidulation tanks (instead of decant thickener overflow solution) to limit the maximum concentration in the tanks to 30% solids.

Slurry overflows from acidulation tank 1 into acidulation tank 2 and then discharges into the POX feed thickener mix tank. Either of the acidulation tanks can be bypassed, if required. The diluted slurry from acidulation requires thickening prior to storage in the POX feed tanks. The POX feed thickener recovers excess solution and advances it to the decant thickener (as wash water) and/or to the iron / arsenic precipitation circuit (to maintain the water balance in the acidulation circuit) or recycles it to acidulation tank 1.

POX thickener underflow slurry is pumped to the POX feed thickener underflow surge tank. The storage in the surge tank allows blending in the correct proportions of the acidulated slurry with the un-acidulated grinding thickener underflow slurry in the POX feed tank to ensure the total level of acid soluble carbonates in the POX feed slurry is within target levels.

The decant thickener recovers acid (that is generated in the POX autoclaves) from the POX discharge slurry and recycles it to the acidulation circuit for carbonate destruction. The underflow slurry is pumped from the thickener to the iron / arsenic precipitation circuit by the decant thickener underflow pumps. Thickener overflow gravitates to the decant thickener overflow tank from where it is pumped to the acidulation tanks by the decant thickener overflow pumps. Solution is bypassed to the POX feed thickener overflow tank when processing low-carbonate ores.

17.1.2.5 Pressure Oxidation

The POX feed surge tanks 1 and 2 are a common feed system that services both POX autoclave trains (T1 and T2). The tanks are agitated to mix / blend the incoming slurry and provide approximately 18 hours of slurry storage to minimise disruptions to the POX circuit. For simplicity, where only POX T1 is discussed in this document it is assumed that both T1 and T2 have identical configurations and controls.

Slurry is pumped to the POX trains 1 and 2 low-temperature heaters by the POX heating feed pumps. The low-temperature (LT) heater receives incoming feed slurry and vent gas (predominantly steam) recovered from the LT flash vessel. The gas heats the slurry to approximately 95°C before being transferred to the high-temperature (HT) heater. The steam in the gas condenses and any excess is vented to the wetted elbow of the POX T1 Venturi scrubber.

The HT heater receives slurry from the LT heater and vent gas (predominantly steam) recovered from the HT flash vessel. The gas heats the slurry to approximately 150°C before being pumped to the POX autoclave. The steam in the gas condenses and any non-condensing gases accumulate in the vapor space at the top of the vessel, prior to being vented.

Slurry is pumped to the autoclave by two pumping trains.

If one full autoclave train is offline, the remaining autoclave train can operate at 150% of normal capacity, provided both of its feed pumping trains are operating.

A horizontal multi-compartment autoclave is used to oxidise the sulfides in the ore at high-temperature and pressure using gaseous oxygen. The oxidation of sulfide material in the autoclave generates heat and when the rate of heat generation exceeds that required to achieve the target temperature of 220°C quench water is added. Sufficient quench water is added to control the temperature to the target. The quench water is pumped through the same sparge pipe that introduces gaseous oxygen addition into the autoclave. There is one sparge pipe underneath each autoclave agitator.

A vent controls the pressure in the autoclave to prevent the water boiling. This pressure is called overpressure and results from the presence of gases such as oxygen, nitrogen, and CO_2.

Slurry discharges from the autoclave through a severe service let down valve to the HT flash vessel. The HT flash vessel operates at a lower pressure than the autoclave and the resulting pressure drop for the discharge slurry entering the HT flash results in steam being flashed from the slurry. The flashing of steam cools the slurry to the equilibrium temperature corresponding to the pressure in the flash vessel.

Steam vented from the HT flash is sent to the HT heater to heat the feed to the autoclave, excess steam is vented to the venturi scrubber for treatment prior to discharge.

Slurry discharges from the HT flash vessel through a severe service let down valve to the LT flash vessel. The LT flash vessel operates at a lower pressure than the HT flash vessel, the resulting pressure drop for the discharge slurry entering the LT flash results in steam being flashed from the slurry. The flashing of steam cools the slurry to approximately 100°C at a pressure just above atmospheric. Slurry is forced from the HT flash vessel to the LT flash vessel by the pressure difference between the two vessels.

Steam vented from the LT flash is sent to the LT heater to heat the feed to the HT heater, excess steam is vented from the LT heater to the Venturi scrubber for treatment prior to discharge.

Steam, entrained slurry, together with gas, including carbon dioxide and unreacted oxygen vented from various points in the autoclave circuit, is scrubbed in Venturi scrubber to remove entrained acidic slurry droplets.

Demineralised water is used in the POX circuit for steam production and for seal water.

Flashed slurry is pumped from the LT flash vessel by decant thickener feed. The decant thickener was described previously and the decant thickener underflow is feed to iron / arsenic precipitation.

17.1.2.6 Fe/As Precipitation

Iron / arsenic precipitation uses limestone slurry addition to the decant thickener underflow slurry to neutralise the free acid and raise the pH to approximately 2.8, which removes ferric iron and arsenic from solution.

The decant thickener underflow duty pump transfers the thickener underflow slurry to iron / arsenic precipitation tank 1. Limestone is added for pH control, and low-pressure air is sparged into the tanks to oxidise any ferrous iron that may be present to ferric iron. The ferric ions combine with the residual arsenic, also leached in the POX circuit, and precipitate together as the pH of the solution is raised. Limestone reacting with the free acid generates carbon dioxide gas and gypsum.

The two iron / arsenic precipitation tanks normally operate in series. The treated slurry overflows from the second iron / arsenic precipitation tank to the CCD 1 Mix Tank.

The low-pressure air and CO_2 generated during the limestone neutralisation reactions rise above the slurry surface on top of the tanks and carry some entrained solution / slurry.

These off gases from the iron / arsenic precipitation tanks (1 and 2) are vented via the iron / arsenic precipitation tank fans 1 and 2 and fed to the iron / arsenic scrubber.

The iron / arsenic scrubber is a Venturi type scrubber. The off gases are cooled and scrubbed of the entrained solution / slurry in the scrubber. The clean gases are emitted to the atmosphere.

17.1.2.7 Counter Current Decantation

Counter current decantation (CCD) washes the iron / arsenic stage discharge slurry with process water using two stages of thickeners operating in counter current mode. The remaining soluble metals in solution exiting the iron / arsenic precipitation circuit are washed from the slurry and report to CCD 1 overflow. The slurry discharging from CCD 2 underflow has the soluble metals washed from the slurry to sufficiently low-levels to feed into the cyanide leach circuit.

CCD thickener 1 overflow solution gravitates into the CCD thickener 1 overflow tank. The duty CCD thickener 1 overflow pump transfers the CCD thickener 1 overflow solution to the neutralisation circuit. The CCD thickener 1 underflow pump transfers the thickener underflow slurry to CCD 2 mix tank. Process water is added in the CCD 2 mix tank as wash solution to wash the solids. Diluted flocculant solution is added in the CCD 1 and 2 thickener feeds to aid in the settling of solids in the thickeners. Duty CCD thickener 2 underflow pump transfers the underflow slurry from the CCD thickener 2 to the pre-leach tank.

17.1.2.8 Cyanide Leach, Carbon Adsorption and Detoxification

The cyanide leach circuit consists of one pre-leach tank and two leach tanks. Slurry is received in the pre-leach tank from the duty CCD thickener 2 underflow pump and flotation tails. The pre-leach tank has a residence time of nominally 10 minutes and is used to raise the pH of the slurry to pH 10–11 prior to the slurry entering the leach tanks where cyanide is added for gold leaching.

The leach tanks have a total residence time of up to six-hours and slurry flows through the leach tanks by gravity and discharges the final leach tank to enter the carbon adsorption circuit. The leach tanks operate at 30% solids concentration and have low-pressure air and oxygen, from the Air Liquide oxygen plant, added to maintain sufficient oxygen in solution for gold leaching.

The carbon adsorption circuit consists of six agitated tanks with a total residence time of up to 12-hours. Each tank contains activated carbon to adsorb the leached gold contained in solution. Slurry flows by gravity from tank 1 to tank 6 and discharges into the detoxification circuit. Carbon flow is counter-current to slurry and therefore is transferred stage wise from tank 6 through to tank 1, using dedicated recessed impeller pumps. Each tank has interstage screens installed so that the carbon remains in each tank and does not follow the direction of the slurry flow.

Gold is loaded onto the carbon as it moves from tank 6 to tank 1 and reaches its maximum loading in adsorption tank 1. The loaded carbon is pumped from adsorption tank 1 to the loaded carbon screen where spray water on the screen washes the carbon prior to it entering the elution column for carbon desorption and recovery of gold through the refining circuit.

Slurry exiting adsorption tank 6 flows to the detoxification circuit where destruction of the residual cyanide contained in the slurry occurs. The detoxification circuit consists of one tank with a total residence time of one-hour. Air and sodium metabisulfite are added to the circuit to destroy the residual cyanide down to a concentration of less than 5.0 ppm CNWAD. Residual copper in the slurry catalyses the cyanide destruction process.

17.1.2.9 Carbon Desorption and Refining

The carbon desorption method selected is a split AARL elution. A common stainless steel column is used for acid wash, cold cyanide strip for copper, when required, and a hot gold elution cycle to recover gold. The elution column is a 6 t column and is designed to handle the stripping of three carbon batches per day. Loaded carbon enters the elution column via the loaded carbon screen.

The first step of stripping the carbon is an acid wash using nitric acid solution to remove loaded impurities such as calcium. After the acid wash, a pre-soak solution is added to the elution column prior to commencement of the eluent recycle for initial stripping of copper, when required, followed by a hot elution cycle to strip gold from the carbon.

Pregnant eluate is collected in the pregnant eluate tank and pumped through electrowinning cells with gold metal plated out onto stainless steel cathodes. Smelting of gold recovered from the stainless-steel cathodes is conducted in the gold refinery.

Desorbed carbon from the elution column is regenerated through a horizontal diesel fired rotary kiln to remove organic material loaded onto the carbon.

17.1.2.10 Neutralisation and Tailings

Slurry from cyanide destruction and the CCD 1 thickener overflow solution are neutralised with lime to precipitate residual metals in solution. Air is added for the oxidation and removal of ferrous iron and manganese.

Normally the two neutralisation tanks operate in series. Discharge from the neutralisation feed box gravity flows into neutralisation tank 1 prior to overflowing into neutralisation tank 2. Discharge from neutralisation tank 2 gravitates into the tailings thickener mix tank.

The first neutralisation tank is equipped with a sodium metabisulfite addition system and this allows it to be used for the detoxification step when the normal detoxification tank is bypassed for maintenance or descaling. Both neutralisation tanks can also be bypassed as required to allow for maintenance.

The discharge slurry from neutralisation flows by gravity into the tailings thickener mix tank before overflowing into the tailings thickener. Tailings thickener overflow water overflows directly into the process water storage tank. The underflow slurry from the tailings thickener is pumped to the agitated tailings tank. The discharge slurry from the tailings tank is pumped to a TSF on a continuous basis via the 4.3 km long tailings pipeline.

A schematic flow sheet of the process is shown in Figure 17.4 including the flotation circuit addition.

17.1.2.11 Tailing Storage Facility

The process tailings slurry is deposited into the TSF for final storage. Operators will alternate the location within the facility where the tailings are deposited to maximise the storage and dewatering within the facility.

In the TSF the solids compact and reject excess water which is recovered for recycling to the process plant. The controlled deposition of tailings at alternating locations around the perimeter of the TSF creates a pond that collects water, which decants from the tailings slurry as it settles and compacts. This decant water collected within the pond area is recycled to the process water system tank via the tailings water reclaim pumps.

The TSF is developed and constructed in stages ahead of requirements.

17.1.2.12 Reagents

There are ten major reagents used in the process plant, listed as follows:

- Oxygen
- Sulfuric acid
- Limestone
- Sodium hydroxide
- Flocculant
- Sodium metabisulfite
- Milk of lime
- Sodium cyanide
- Nitric acid
- Antiscalant

The flotation plant has the following main reagents:

- Frother
- Collector
- Copper Sulfate

All reagents are delivered in bulk tankers, containers, or bags with storage on-site. Any reagents that require dilution or mixing prior to use are prepared on-site on a batch wise basis, as required. Oxygen is produced on-site supplied from an Air Liquide owned and operated oxygen plant under a gas supply agreement. Additional oxygen can be delivered as liquid into on-site storage.

17.1.2.13 Utilities

The major utilities used in the process plant are as follows:

- Iron / arsenic low-pressure air
- CIP leach low-pressure air
- Plant air
- Instrument air
- Raw water
- Fire water
- Potable water
- Process water
- Diesel fuel

These utilities are reticulated throughout the process plant to their end user.

17.2 Oxide Heap Leach Processing

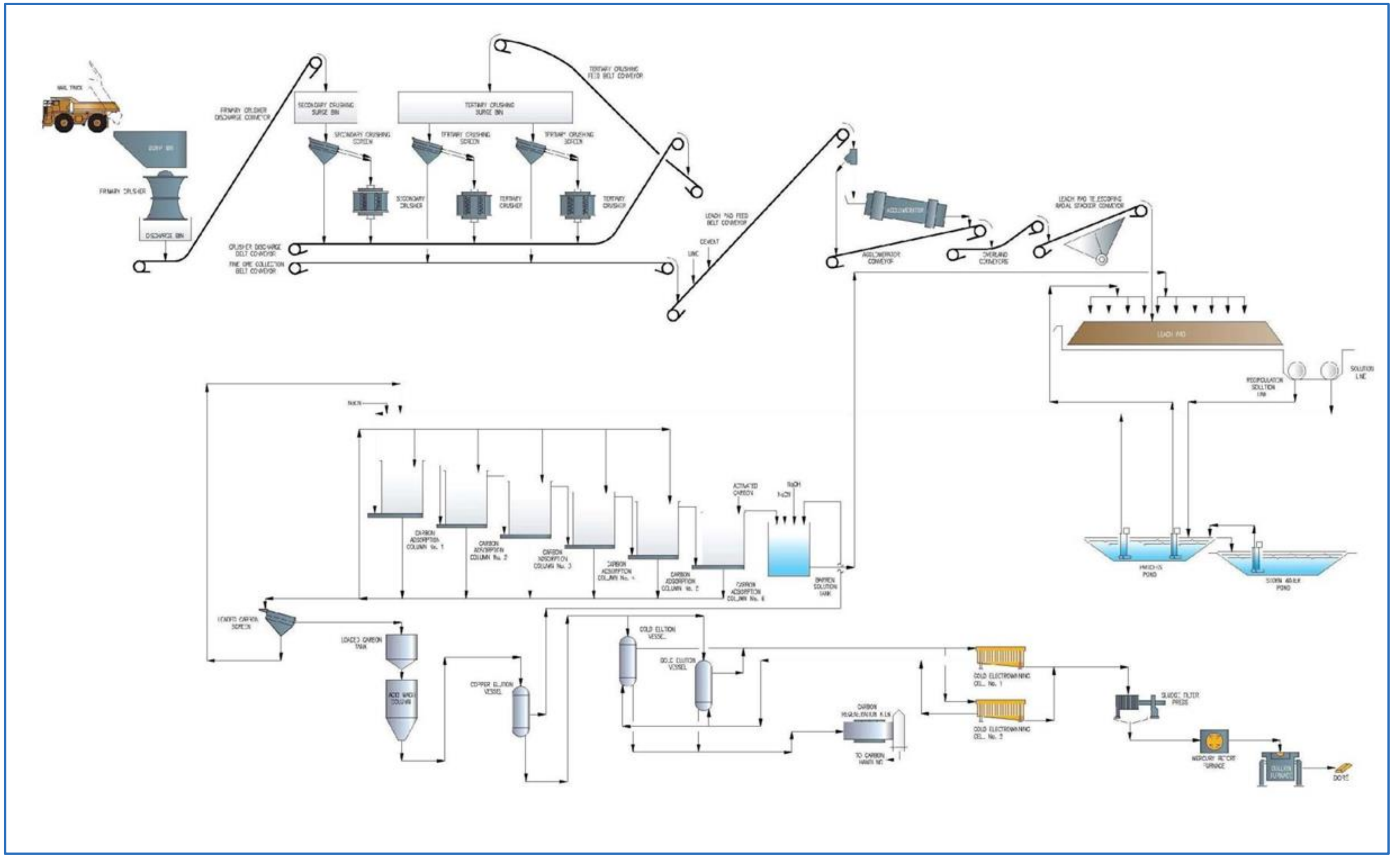
The oxide heap leaching and associated facilities were commissioned in the second half of 2010 and initial gold production was achieved in Q4'10. The process was originally designed to treat approximately 6.0 Mtpa of ore by three-stage crushing (primary, secondary, and tertiary) to 80% passing 12.5 mm, agglomeration and heap leaching on a lined heap leach pad with dilute alkaline sodium cyanide solution. Gold is recovered through a carbon-in-column (CIC) system, followed by stripping of metal values from carbon, electrowinning and melting to yield a doré (containing gold and silver) suitable for sale. Control of copper in leach solutions is undertaken in a sulfidisation, acidification, recycling, and thickening (SART) plant which also regenerates cyanide. The process flow sheet is summarised in Figure 17.5.

17.2.1 Oxide Heap Leach Performance

Since commissioning through the end-of-December 2021, an estimated 55.1 Mt of oxide ore was placed on the heap at an average grade of 1.35 g/t Au.

At the end-of-December 2021, a total of approximately 1,837 koz had been produced as bullion.

Figure 17.5 Heap Leach Process Flow Sheet



Anagold, 2016

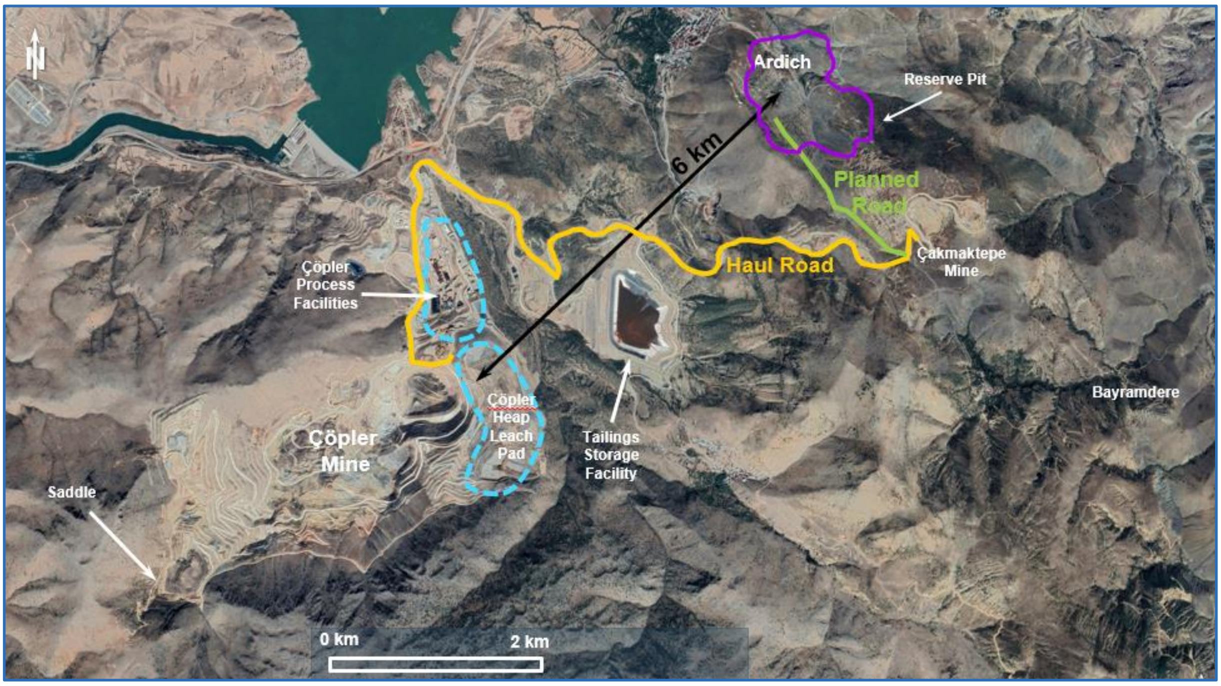
18 PROJECT INFRASTRUCTURE

18.1 Introduction

The facility infrastructure supports the mine, and process areas of oxide heap leach and sulfide plant. The existing infrastructure, and the tailings storage facility (TSF) and heap leach pad area when the planned expansion are complete will be sufficient for the current Mineral Reserves. The infrastructure for the addition of flotation to the sulfide plant will be supported by the existing facility infrastructure with some components modified to meet the addition of the flotation circuit. The flotation circuit is located within the sulfide plant foot-print adjacent to the grinding circuit building.

The location of the processing facilities, Çöpler mine, Ardich Reserve pit, TSF, and the haul road from Ardich to Çöpler is shown in the site plan in Figure 18.1.

Figure 18.1 Çöpler Project Plan



Anagold, 2022

The current leach pad consists of four phases designed to accommodate approximately 58 Mt of oxide ore heap with a nominal maximum heap height of 100 m above the pad liner. The additional two phases (5 and 6), with a capacity of 20 Mt will be constructed during 2022 to 2024 to accommodate the remainder of the Ardich Reserve. The phase 5 pad construction has been approved by the Ministry of Environment, Urbanisation, and Climate Change (MoEUCC).

The TSF is developed and constructed in stages. TSF 1 phase 3 has been constructed and approval for use was received in February 2021 by the MoEUCC. Ongoing work in ensuring sufficient long-term capacity for storage of tailings has been undertaken. Studies by Anagold have determined, that the effect of the addition of the flotation plant to the sulfide plant circuit would result in an increase in the solids content and improvement in the final settled density based on an increase in the rate of tailings consolidation.

Construction and development of TSF 1 will provide storage of tailings for up to 70.8 Mt, more than sufficient to accommodate the CDMP21TR tailings to be produced.

A PFS level study (TSF 2) has been carried out that identifies approximately 13.4 Mt additional tailings storage capacity in a site adjacent to TSF 1, should it be required in the future.

18.1.1 Existing Infrastructure

The existing site infrastructure supporting the existing operation includes the following:

- Site security gate and guard station
- Site administration building
- Site warehouse
- Site assay laboratory
- Container or modular type offices
- Cyanide receiving and mixing system
- Site kitchens and eating areas
- Site single living dormitory with adjacent multi-purpose room
- Site family housing
- Contractor (mining) dormitories, kitchens, and offices
- Site raw water wells, pumping system and storage tanks
- Site potable water treatment and distribution system
- Two sanitary wastewater collection and treatment systems
- Sulfide maintenance building
- Sulfide control rooms
- Combined oxide and sulfide gold refinery building
- Sulfide process buildings:
  - Grinding building
  - Pressure oxidation (POX) building
  - Carbon desorption building
- Tailings pump building
- Main control room and electrical building

- HV switchyard electrical building
- Crusher electrical building
- POX flocculant building
- Limestone building
- Potable water booster pump house
- Reagent building
- Tailings and process water pump house
- Plant and instrument air compressor building
- Counter current decantation (CCD) electrical building
- Reagent dry storage
- Leach air compressor building
- Aw water pump building
- Lime slaking (MOL) building
- Fe/As air compressor building
- Emergency diesel generators building
- TSF reclaim electrical building
- TSF drainage tank electrical building
- TSF OD-UD pond electrical building
- CIP CCD ablutions block
- Pump shelters with monorails

- Carbon elution building – electrical room
- Raw water bores P/P house and electrical building
- Gatehouse
- Fire water pump house
- Community relations centre
- Raw water wells

18.1.2 Flotation Building

The flotation circuit is an insulated engineered building. The building is equipped with an overhead crane for flotation cell and pump maintenance. Flotation reagent mixing and distribution are contained in a lean-to off the main flotation building.

18.2 Site Water Management

18.2.1 Hydrology Background

The only perennial surface water in the vicinity of the Çöpler Mine is the Karasu River flowing in the northern and western part of the area. All other valleys are either ephemeral streams or dry valleys. The average flow rate of the Karasu River measured at the Bağıştaş / Karasu Gauging Station in the upper Euphrates Basin, is approximately 145 m^3/sec, draining an area of 15,562 km^2. A hydroelectric dam (Bağıştaş -1 Dam) was built on the Karasu River downstream of the mine site. When the reservoir is at high-levels the impoundment will extend into the very lower reaches of both the Çöpler and Sabırlı Creeks and the maximum inundation elevation will be 916.5 m as it is released into the spillway. The Çöpler and Sabırlı streambeds in the project area do not flow perennially. They both discharge into the Karasu River. The drainage area of the Sabırlı Creek is approximately 35 km^2 and that of the Çöpler Creek is approximately 10 km^2.

The project submitted a Five-Year Water Management Report in December 2019, prepared by SRK Danışmanlık ve Mühendislik A.Ş., as part of the Environmental Impact Assessments (EIA) conditions. This report benchmarks the expected results with those achieved. Overall results achieved were generally as predicted. In 2020, as part of updating the EIA, further hydrogeology studies have been undertaken by SRK Danışmanlık ve Mühendislik A.Ş. The report has updated the surface water and hydrological models based on actual data over the operating period of the mine to improve the model.

18.2.2 Site-Wide Surface Water Hydrology

Existing mine site facilities are located primarily within the Çöpler and Sabırlı Creek watersheds immediately upstream of their confluence with the Karasu River. Site-wide surface water management for the included diversion facilities consist of a network of diversion channels and retention structures to minimise storm water run-on to the mine site facilities to prevent mine-impacted storm water run-off from exiting the site and discharging to the Karasu River.

The sub-basin areas, characterisation of the surface run-off conditions, and design rainfall data were used to construct the existing conditions hydrology model. The hydrology analysis utilised HEC-HMS software to develop estimates of the peak flow rates and volumes generated by the existing watersheds.

18.2.3 Surface Water Management Structures

Engineered surface water management structures are constructed to minimise effects of storm water run-off to critical mine facilities and to control the release of mine-impacted water to the environment. A combination of interim and permanent diversion channels and retention ponds are utilised to achieve these goals. Interim structures will be reclaimed at closure while permanent structures will remain in place post-closure. Other flood control structures were developed to control or direct run-off away from pit crests and are planned for run-off that does not discharge to surface water drainages or streams and therefore do not require lining.

Sediment ponds to control run-off and sediment release are lined based on the EIA commitments. Interim diversion channels are designed to convey the 25-year storm event with 1.5 m of freeboard and the 100-year storm with no freeboard. Permanent diversion channels are designed to convey the 100-year storm with 0.5 m of freeboard. Lined sediment ponds are downgradient of the waste dumps and are sized to contain the 100-year run-off volume with an emergency spillway to safely discharge the peak flow. The TSF is designed to contain the volume generated by the 24-hour probable maximum precipitation (PMP) within the operating freeboard.

18.2.4 Fresh Water Supply

Fresh water is supplied by existing wells to the site, supporting the operation. Figure 18.2 shows the location of the mine water extraction wells. An additional three wells were developed in 2018, wells WM-45, WM-46 and WM-47, to increase water supply for the project. Two raw water storage tanks support the demands of the heap leach and sulfide process equipment and the fire water requirements.

Figure 18.2 Mine Water Supply Well Locations

Anagold, 2020

18.2.5 Potable Water Treatment

The site is serviced by a potable water treatment system and distribution system. The system consists of multi-media filtration, carbon filtration, ultraviolet (UV) disinfection system (plus further softening and reverse osmosis for water used in the dining room), which directly feeds the site potable water distribution system.

18.2.6 Waste Management

Waste will be generated from multiple sources such as human waste, food spoilage, and process and maintenance wastes.

Hazardous wastes will be contained, packaged, and disposed of in accordance with local, regional, and national regulations. Non-hazardous wastes will either be buried on-site or transported offsite to the appropriate processing site in accordance with local, regional, and national regulations.

18.3 Power to Site

The existing 154 kV line provides power to the mine and process plant. The following structures are associated with site power distribution:

- HV switchyard 154 kV
- Main electrical building
- Oxygen plant substation
- CCD electrical building
- Crushing electrical building
- Grinding electrical building
- Carbon elution electrical room
- TSF area electrical buildings
- Bore field area electrical building

18.4 Emergency Backup Power

Motors and loads for certain critical equipment and systems were identified as requiring power in the event of a utility outage. A load shedding scheme is applied to feed critical electrical users automatically in the event of a utility outage.

Generators are diesel fuelled with a minimum of eight-hours of diesel storage based on generators operating under full load.

18.5 Communications

The Project uses networks for the DCS, precious metals search (PMS), the integrated process related and security CCTV system, security systems (access control / card reader), information technology (IT) and telephones and communication between the DCS and packaged control systems.

Single mode fibre and copper cabling is distributed within the sulfide plant area and selected buildings for the tailing pipeline and dam.

18.6 Site Roads

The Çöpler project has access provided via the main access road and sulfide plant roads.

Generally, site roads have an overall width of 6 m and provide everyday operational access for large trucks or facility access for site personnel vehicles. These roads are limited to a maximum grade of 9%. All roads are compacted hardstand surfaced with 100 mm wearing course and cross-sloped to provide positive drainage.

18.7 Plant Fire Protection System

A separate plant fire protection system is provided for the sulfide facility and will include the flotation building.

A combined sprinkler, hose reel and hydrant underground piping system is provided for the active fire protection of the facility.

A gas-based fire suppression system is used in the main control and electrical building.

18.8 Site Water Management

18.8.1 Hydrology Background

The only perennial surface water in the vicinity of the Çöpler Mine is the Karasu River flowing in the northern and western part of the area. All other valleys are either ephemeral streams or dry valleys. The average flow rate of the Karasu River measured at the Bağıştaş / Karasu Gauging Station in the upper Euphrates Basin, is approximately 145 m^3/sec, draining an area of 15,562 km^2.

A hydroelectric dam (Bağıştaş -1 Dam) was built on the Karasu River downstream of the mine site. When the reservoir is at high-levels the impoundment will extend into the very lower reaches of both the Çöpler and Sabırlı Creeks and the maximum inundation elevation will be 916.5 m as it is released into the spillway.

The Çöpler and Sabırlı streambeds in the project area do not flow perennially. They both discharge into the Karasu River. The drainage area of the Sabırlı Creek is approximately 35 km^2 and that of the Çöpler Creek is approximately 10 km^2.

The project submitted a Five-Year Water Management Report in December 2019, prepared by SRK Danışmanlık ve Mühendislik A.Ş., as part of the EIA conditions. This report benchmarks the expected results with those achieved. Overall results achieved were generally as predicted.

In 2020, as part of updating the EIA, further hydrogeology studies have been undertaken by SRK Danışmanlık ve Mühendislik A.Ş. The report has updated the surface water and hydrological models based on actual data over the operating period of the mine to improve the model.

18.9 Heap Leach Facility

The heap leach includes the leach pad and collection ponds that consist of process ponds and a storm pond. The current leach pad consists of four phases and designed to accommodate approximately 58 Mt of oxide ore with a nominal maximum heap height of 100 m above the pad liner. The additional two phases, 5 and 6, with a capacity of 20 Mt were approved in October 2021 and phase 5 has received construction approval from MoEUCC in November 2021. The heap is stacked in 8 m thick horizontal lifts at the natural angle-of-repose with intermediate benches to achieve an overall heap slope of 2H:1V.

18.9.1 Heap Leach Pad Development

The heap leach facility pad development is in six phases, and is in the same geographical area, adjacent to the Çöpler open pit as shown on Figure 16.8. The heap leach phases 1 to 4 are completed.

The remaining phases of pad development 5 and 6 are yet to be constructed and will have a combined capacity of approximately 20 Mt.

The phase 5 (15 Mt capacity) was approved for pad construction in November 2021.

The phase 6 (5 Mt capacity) sits above phase 4B and 5 and will be the last to be constructed and stacked. Approvals and construction will be scheduled well in advance of being required for ore stacking and leaching.

18.10 Tailings Storage Facility

The existing tailings storage facility (TSF) at the Çöpler mine was designed by Golder Associates Inc. (Golder) with support from Golder Associates Turkey, Ltd (Golder Turkey). The TSF initial design was developed to provide a capacity of 45.9 Mt through six phases with a crest elevation of 1,265 m. The TSF was permitted through submission of a Turkish Design Application Report to the MoEUCC and subsequently approved based on the design through phase 5.

Anagold is advancing the development of the Çöpler Mine. Recently developed a prefeasibility level design for an additional TSF, referred to as TSF 2 in the valley adjacent and to the north of the existing TSF 1. Both TSF 1 and TSF 2 were included in the EIA submitted by Anagold in 2014.

The current designs for TSF 1 and TSF 2 are within the 2014 EIA boundaries, except for a small portion of TSF 1, phase 7. Expansion beyond phase 3 of TSF 1 is currently limited by the construction and re-routing of a new road to Sabırlı Village as well as purchase of some small tracts of private land located within the phase 4 limits on the east side of the existing road to Sabırlı Village. Construction of the new Sabirli Village road commenced in Q3'21 and is on schedule. Acquisition of the private land parcels have substantially progressed through regulatory processes.

Based on the prefeasibility design, TSF 2 has capacity for 13.4 Mt. To maximise capacity of TSF 1, phase 7 was developed as part of the design to a crest elevation of 1,280 m at a conceptual level and to support further planning, including planned updates to the site Environmental Assessment. Select engineering evaluation of phase 7 has been completed to support future planning including updated stability analysis, water balance, and consolidation modelling. Anagold's preference is to continue with development of TSF 1 phase 4 and to consider other options, if required depending on tailings capacity requirements, due to the higher capital costs related to construction of TSF 2 at this time. Without construction of TSF 2, TSF 1 alone provides for tailings capacity of up to 70.8 Mt through phase 7.

Figure 18.3 through to Figure 18.7 show the revised TSF 1 design for phases 4–7, and the TSF 2 design.

18.10.1 TSF Development and Summary of Current Operations

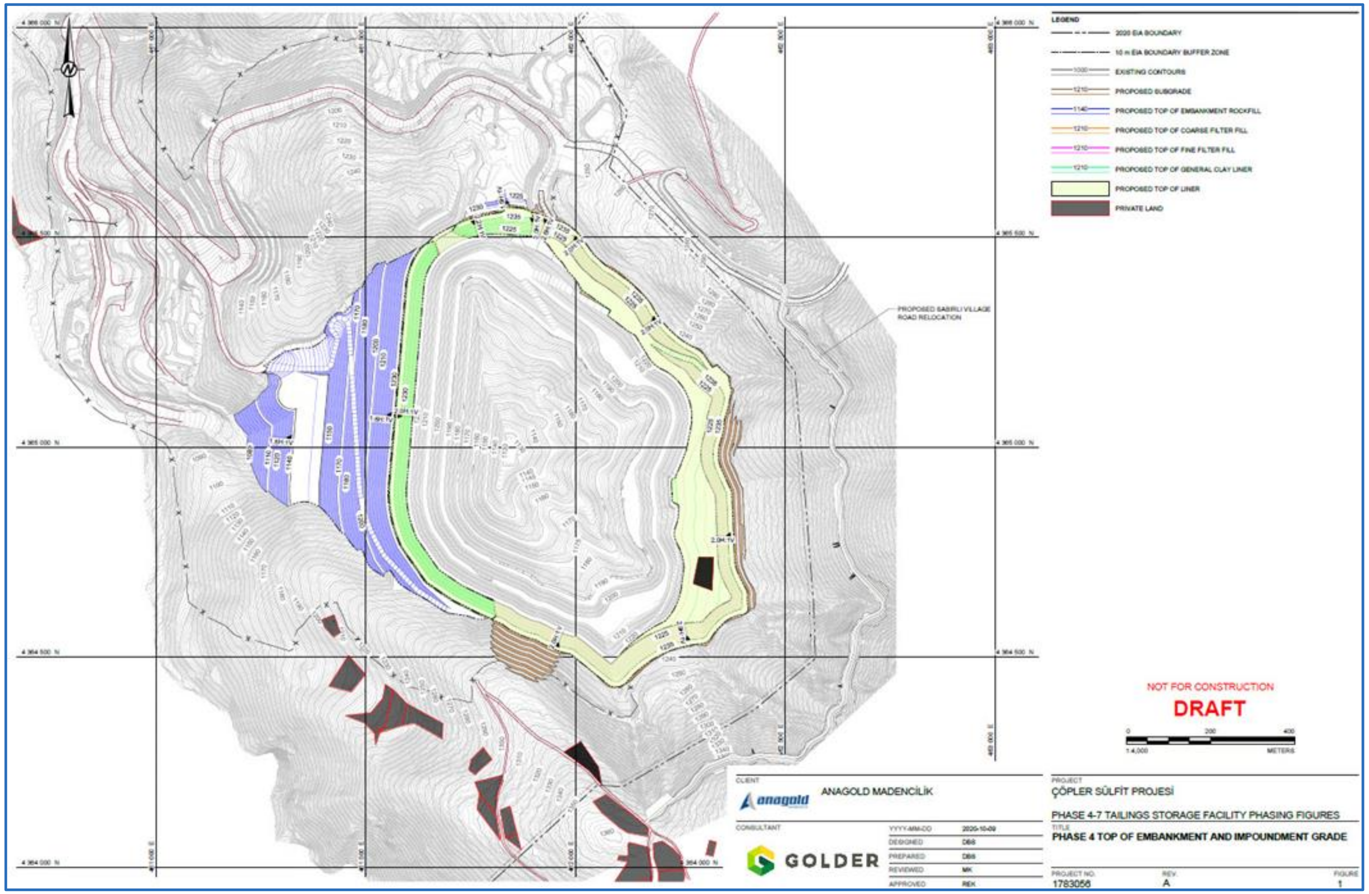
Construction of phase 1 of TSF 1 began in December 2016 and was completed in November 2018 with commissioning of the sulfide plant. Tailings were deposited initially from the emergency spigot and then typically from two to three spigots around the perimeter of the 1,190 m crest of the phase 1 embankment. The tailings initially have exhibited a solids content on the order of 24%. During the first two-years of operations 4–5 m of water has been present over the top of the tailings surface. Reclaim water was managed by pumps on a rail-mounted sidehill reclaim system. The second raise, or phase 2 of TSF 1, was completed in April 2020 and construction of phase 3 is ongoing. The management of reclaim water has improved in the past-year and currently the tailings surface is nominally 3 m below the top of the tailings water. A bathymetry survey was completed on 11 September 2020 and indicated a tailings average dry density of 0.68 t/m^3.

The reclaim water management system was converted to a conventional pontoon system accessible for maintenance from ramps constructed within the northern portion of the impoundment. Based on additional tailings testing completed in early 2020, the solids content of the tailings has improved to approximately 28% as a direct result of improved throughput stability at the sulfide plant, improvements to type of flocculants used and process control in the tailings thickener. As part of this tailings testing in early 2020, Golder evaluated the effect of the addition of the flotation plant to the sulfide circuit. The testwork indicated an increase in the solids content to 34% and improvement in the final settled density based on an increase in the rate of tailings consolidation.

18.10.2 Site Classification

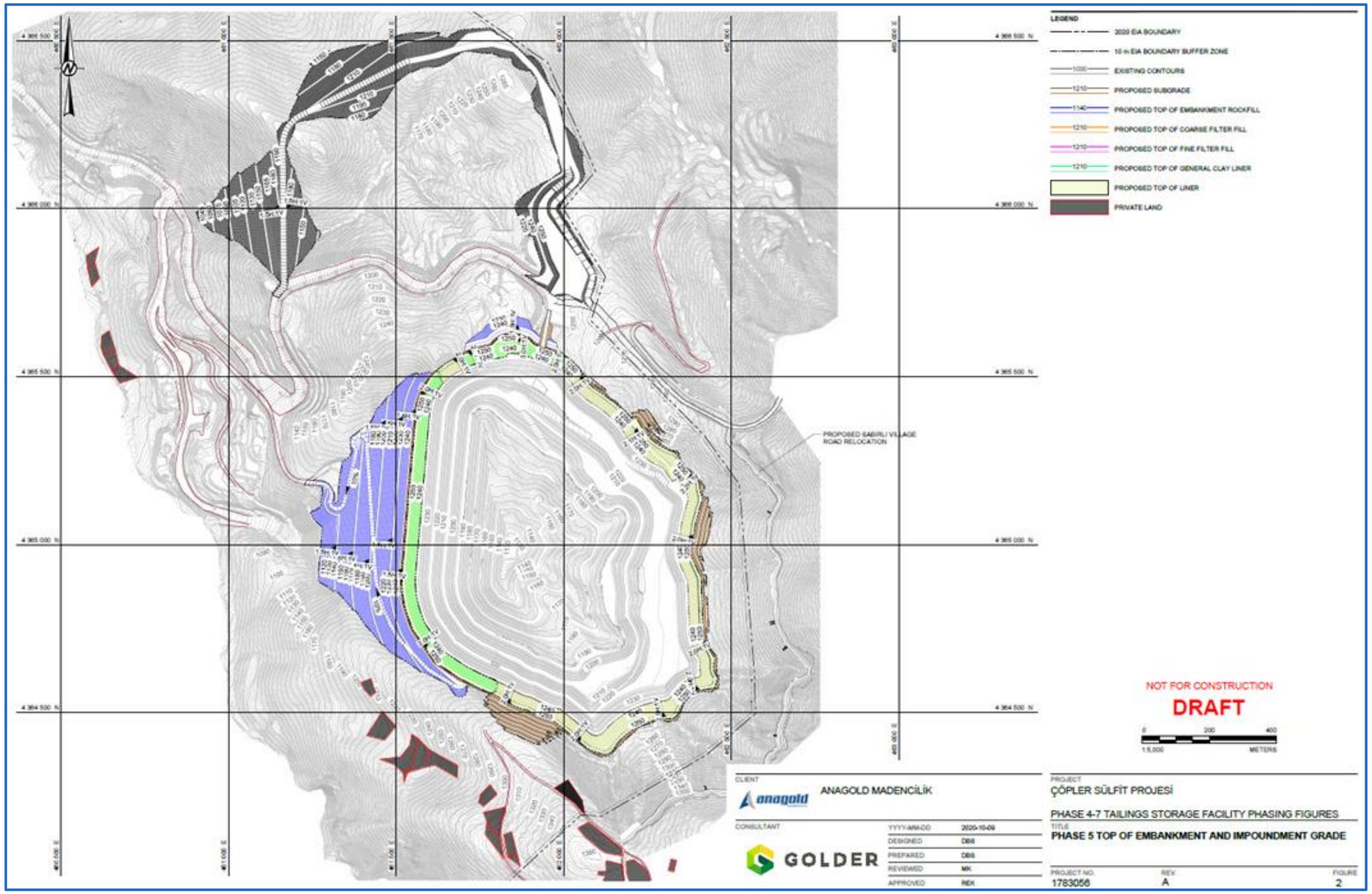
The facilities are classified in accordance with the Canadian Dam Association (CDA) guidance (2013 Edition) as 'High' for the operational and post-closure phases. The 'High' classification is the third lowest in terms of risk with the dam classes being from least risk to greatest risk: Low, Significant, High, Very High, and Extreme.

Figure 18.3 Phase 4 – Top of Embankment and Impoundment Grade



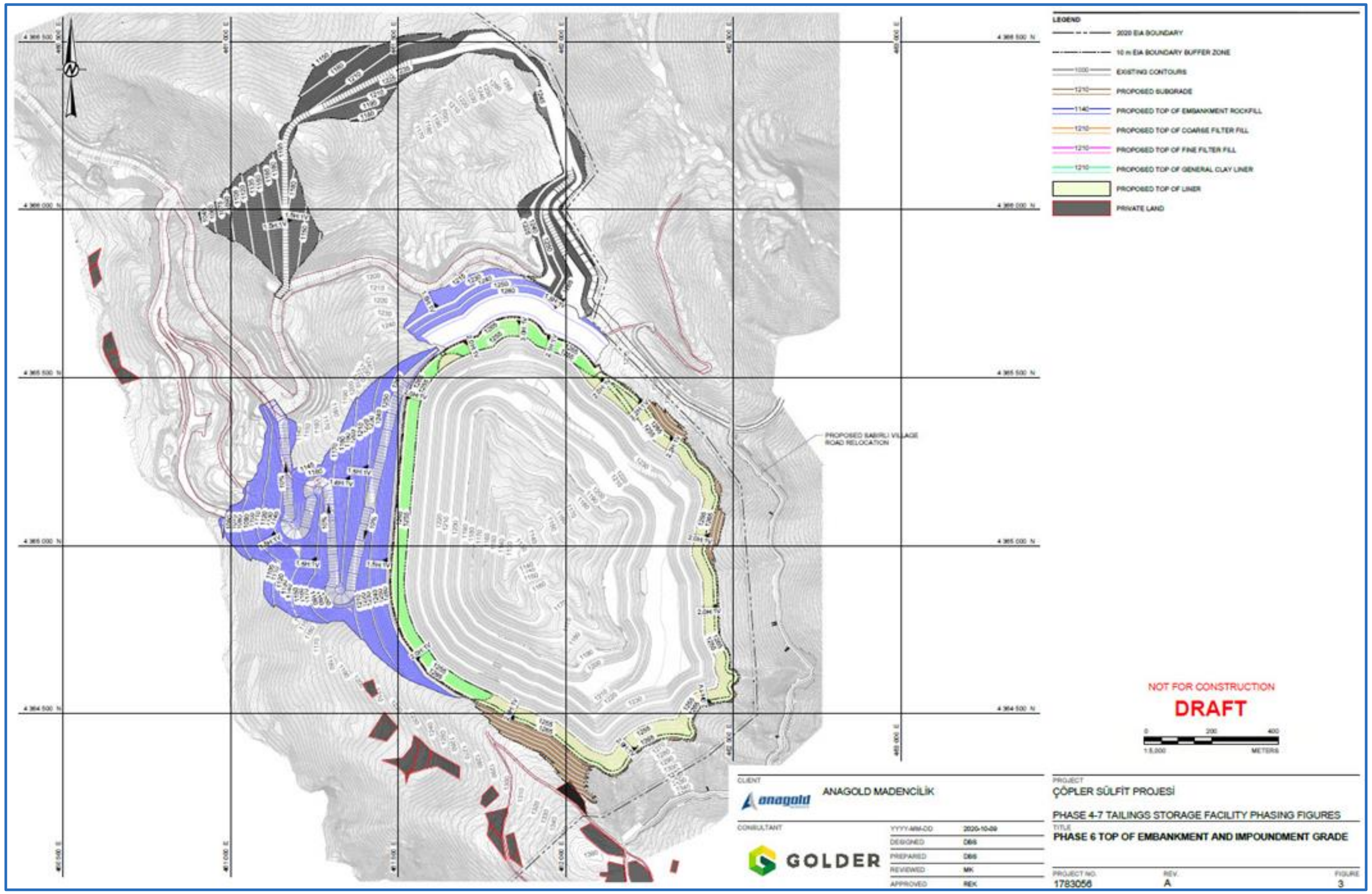
Anagold, 2020

Figure 18.4 Phase 5 – Top of Embankment and Impoundment Grade



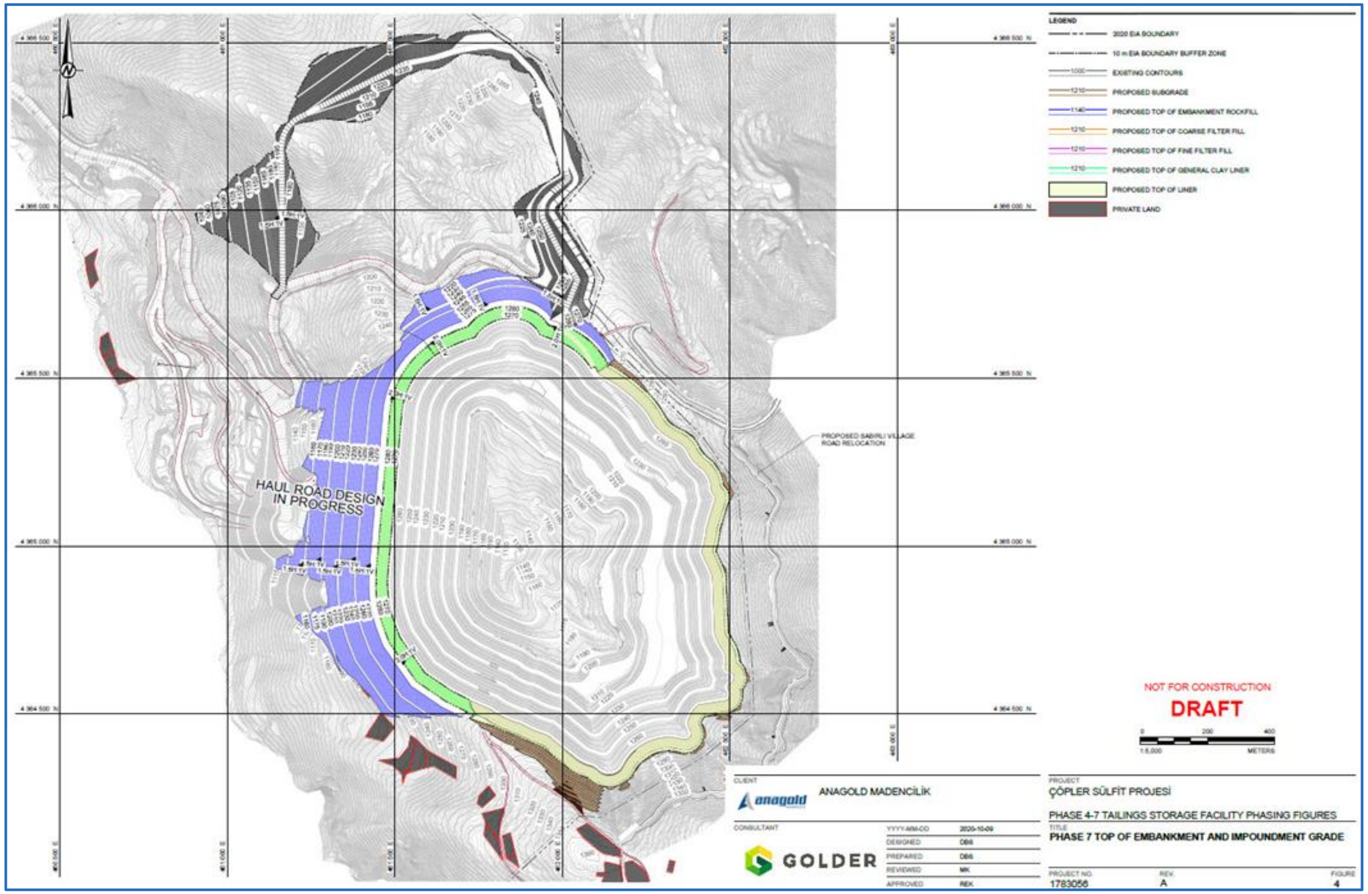
Anagold, 2020

Figure 18.5 Phase 6 – Top of Embankment and Impoundment Grade



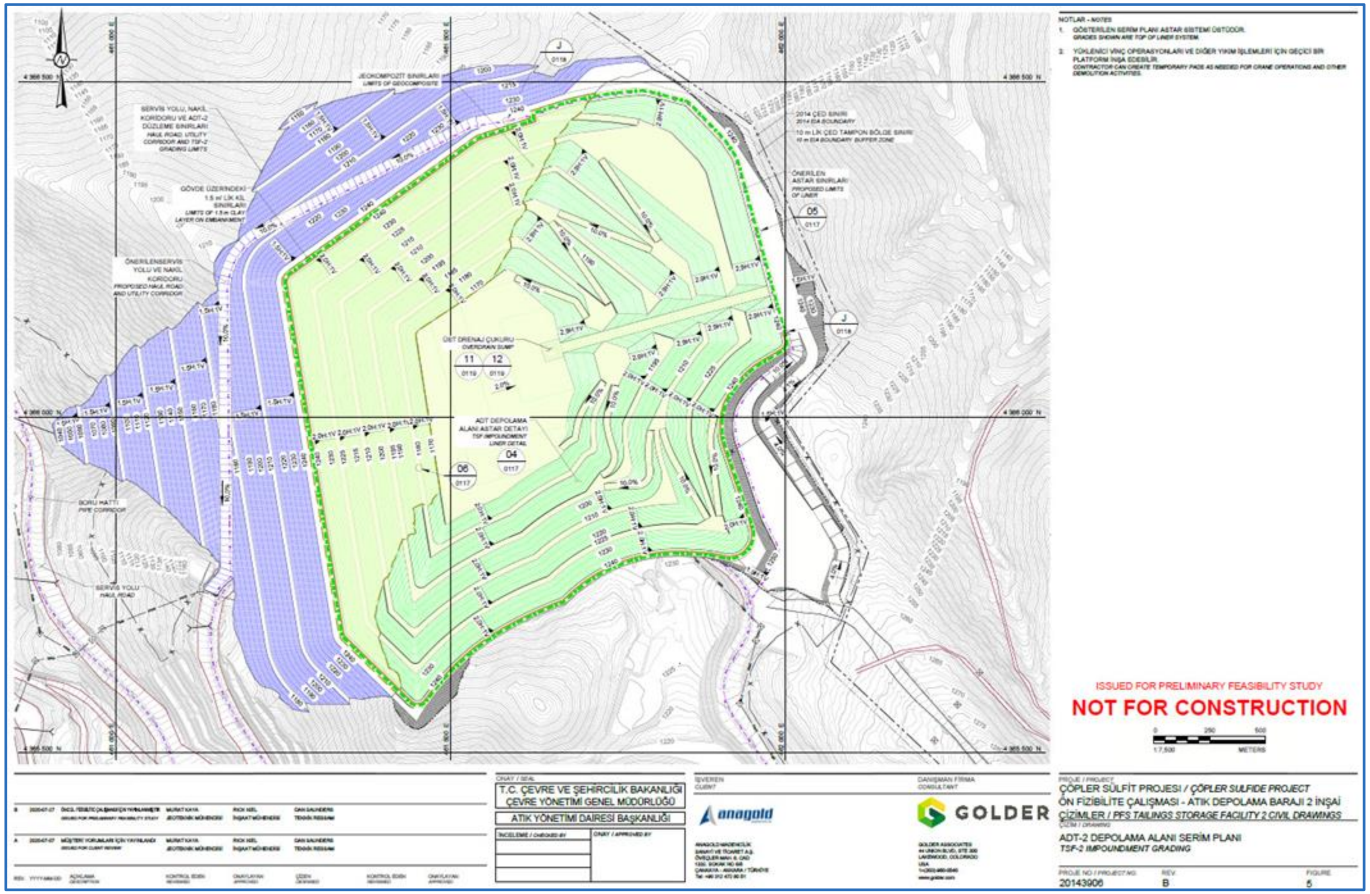
Anagold, 2020

Figure 18.6 Phase 7 – Top of Embankment and Impoundment Grade



Anagold, 2020

Figure 18.7 TSF 2 Impoundment Grading



Anagold, 2020

18.10.3 Monitoring and Inspection

An Operational, Maintenance, and Surveillance (OMS) Plan was prepared by Golder with input and support from Anagold. The OMS Plan was prepared in accordance with the Turkish mining regulations (MoEU 2017) with additional guidance published by the Mining Association of Canada (MAC 2019). The OMS Plan is a 'living document' that is updated on an annual basis. In addition to providing the basic guidance for the management of process fluids, the OMS Plan does the following:

- Summarises the roles and responsibilities of Anagold personnel.
- Presents a description of the facility and pertinent design details.
- Provides maintenance and surveillance parameters and procedures.
- Outlines abnormal operating conditions.
- Details emergency preparedness and response protocols.
- Presents a conceptual closure plan.

The OMS Manual provides a documented framework for action, as well as a sound basis for measuring performance and demonstrating due diligence. It is intended to be a dynamic document that is reviewed and revised by site personnel and the Engineer of Record (EoR) on an annual basis and as operating conditions require. The OMS Manual includes a requirement for the annual dam safety inspection prepared by the EoR which includes a series of inspections at site that is documented in an annual Dam Safety Inspection Report. The first annual inspection for TSF 1 was conducted in Q4'19. The results of the inspection and data review indicated that the Çöpler TSF 1 is in good condition and operating in general accordance with the intended design of the facility. A review of the instrumentation indicated normal data trends and no unanticipated abnormal readings or 'triggering events' observed. Of the action items included in the report, none were considered serious in nature or otherwise a concern to the safety of the Çöpler TSF.

The TSF is inspected daily for signs of stress or damage. Daily and monthly operating data is collected on-site and provided in a monthly report. The report estimates the settled solids volume in the TSF based on estimated bulk densities and provides for a comparison of actual tailings and water pool elevations compared to estimates made by Golder using data from the mine and tailings production plans and from the consolidation model that predicts settlement of the tailings. The difference between the actual tailings elevation and predicted elevations have shown close agreement generally less than 1 m.

In addition, members of the Anagold's HSSER team also inspect the TSF monthly. The TSF is subject to fortnightly external official audits by the Erzincan Provincial Environmental Directorate. The authorised hydraulic structures inspection company, Hidro Dizayn, is always on-site during construction, on behalf of the MoEUCC. The TSF design and engineering consultant is also on-site during construction to ensure quality and conformance to design.

Anagold has established an Independent Tailings Review Board (ITRB), as per leading international best practices, to review tailings facilities as part of the review and oversight process. The ITRB reports directly to the senior management at a corporate level.

18.10.4 TSF Design

The TSF at Çöpler is a downstream, mass filled, dam. The technical specifications for the construction of the TSF conform with both Turkish national requirements and accepted good practice standards for tailings facilities, including; World Bank Standards, Canadian Dam Association Safety guidelines, Mining Association of Canada (MAC) Guide and the International Commission on Large Dams (ICOLD) to the Management of Tailings Facilities.

Both the TSF 1 and TSF 2 designs consist of fully lined impoundments, including a compacted earth and rockfill embankment. The TSF 1 and TSF 2 designs include the following primary components:

- A compacted earth and rockfill embankment with a zoned upstream granular filter protection system. Both facilities will have 1 m of freeboard under their crest elevations and are designed to contain the PMP storm event. The downstream face of the ultimate embankments will be constructed at a composite slope of 1.7H:1V. The upstream face of the embankment will be constructed at a slightly shallower slope with slopes of 2.0H:1V to facilitate placement of the filter layers and liner system and a resultant composite slope on the order of 2.6H:1V after considering the operational benches. The filter layers and low-permeability soil layers are designed to be 1.5 m thick, as measured perpendicular to the slope. Measured horizontally, the layers are designed at 3.3 m wide each.
  - TSF 1 is a downstream raise construction which will consist of seven phases (six raises)
  - TSF 2, if constructed, is a downstream raise construction and is currently designed to be constructed in one phase.
- A composite liner system consisting of a 2 mm thick, double-sided, textured high-density polyethylene (HDPE) geomembrane and geosynthetic clay liner (GCL) over a low-permeability soil (i.e., clay) liner system that provides an equivalent protection to that provided by 5 m of a geologic barrier with k <10-9 m/s. A GCL is also substituted with low-permeability clay on select slopes steeper than 3H:1V as allowed by Turkish regulations.
- An impoundment gravity flow underdrain system for collection and monitoring of naturally occurring seeps and springs.
- An impoundment overdrain system for the collection and management of tailings seepage water through natural consolidation and drainage of excess process water.
- Perimeter roads and benches within and around the impoundment area for access and tailings distribution / reclaim water pipes.
- Tailings delivery and distribution system.
- Reclaim Systems.

18.10.5 Seismic Deformation Evaluation

The current deformation model provides the deformations under seismic loading conditions for a TSF 1 with 1,264 m crest elevation, which corresponds to phase 6 in the current design. Based on the average predicted deformations and the expected levels of liner strain, the TSF 1 phase 6 embankment is expected to remain stable when subjected to the design strong motion events. Simple deformation analysis by Bray and Travasarou (2007) was performed to assess the magnitude of earthquake induced movements on the phase 7 TSF 1 Embankment.

No deformation analysis was performed for TSF 2 considering it is a smaller dam and has a lower embankment height than TSF 1 and because of the similarities in design and foundation conditions. TSF 2 deformations are expected to be smaller than TSF 1 and in the acceptable deformation range as per the design criteria.

18.10.6 Tailings Consolidation and Capacity

Golder updated the tailings consolidation modelling to include the TSF 1 and TSF 2 joint operations and to account for the tailings characteristics obtained from 2020 laboratory tests on POX and Flotation tailings. The updated consolidation model also included the current mine plan. In the model, TSF 1 was first filled to elevation of 1,219 m (to the limits of phase 3 with a crest elevation of 1,220 m allowing for 1 m freeboard) and then tailings deposition was switched to TSF 2 and tailings in the TSF 1 was let to rest until TSF 2 is filled for a period of approximately 3.4 years. The rest period in TSF 1 increases the tailings density from 0.85 t/m^3 to 1.08 t/m^3 due to the natural consolidation and results with an average settlement on the order of 7 m which results in a capacity gain of 3.2 Mm^3 in TSF 1. The model results show that with the current mine plan and tailings characteristics TSF 1 and TSF 2 would have approximately 76 Mt and 14 Mt tailings capacity, respectively, over approximately 19.2 years of TSF 1 filling time.

The tailings tonnage estimate requires the sulfide plant feed to be adjusted to allow for the limestone added during processing for pH control. The limestone reacts with the acid to form gypsum. The applicable factor is 1.146. Commissioning of the Flotation Plant commenced in January 2022, once operational will also directly contribute to the tailings placed.

Based on the updated consolidation analysis and assumptions on the mine plan, tailings characteristics, and operational plans as stated herein, approximately 90.6 Mt of tailings can be stored in TSF 1 and TSF 2 combined. The average dry tailings density expected at end of filling is 1.17 t/m^3 and 0.89 t/m^3 in TSF 1 and TSF 2, respectively.

18.10.7 TSF Schedule Assumptions

The key assumptions related to the ongoing construction and expansion of TSF 1 as follows:

- Phase 4:
  - There is a parcel of private land located east of Gully B that has not yet been purchased. If the private land cannot be purchased, contingent measures are in place to allow phase 4 to be constructed.
  - Construction of the new Sabirli Road is required for Phase 4 to be completed. Construction commenced in Q3'21 and at the time of this report is on schedule for timely completion

* Phases 5 to 7:
  - The design of the access roads and utility corridor for phases 5 to 7 considered construction of the haul road developed as part of the TSF 2 design which requires nominally 4 Mm^3 of rock fill. This design has been shown starting with phase 5. If TSF 2 were not developed, the access road and utility corridor could be further optimised depending on the extent of development. If only phase 5 were to be developed, the access could be provided by a much smaller ramp. If phase 6 and/or phase 7 are developed, then a route like that shown would be required.
* Schedule:
  - The current mine plan and schedule provides capacity within phase 3 through Q1'23, which generally required that construction of phase 4, would start in 2021. Phase 4 embankment works were progressed approximately 66% during 2021 and embankment and impoundment works are scheduled to continue through 2022. As an ongoing option, the TSF 2 design and approval activities were also substantially progressed during 2021

Sabirli road completion is scheduled for completion in time to achieve phase 4 TSF construction works.

18.10.8 Further Work

There are opportunities that may offer significant reduction in capital costs with consideration of the following:

* Alternative TSF Considerations – The dam capacity to fill ratio for TSF 2 was approximately 1:25, which is significantly lower than TSF 1 due to the narrow and small valley where it is located. Several other options were identified in the CDMP21TR Siting Study that would provide for reduced capital costs. Sites identified as TSF 4 and TSF 7 were determined to have dam capacity to fill ratios of 1:3.2 and 1:1.9, respectively based on conceptual designs only. Of the other sites considered in the CDMP21TR Siting Study, TSF 4 was ranked second behind TSF 7 based on several environmental and social considerations namely due to its proximity and location with the Bağıştaş area, however, TSF 4, provides a significantly greater potential storage capacity with less fill required. TSF 7 would be highly visible to the Sabirli community but on the opposite side of Sabirli creek.
* Waste Rock Encapsulation – There are opportunities to consider encapsulation of potentially acid-generating (PAG) waste rock within portions of the downstream embankment within the limestone. A study commenced in Q3'21 to evaluate this potential

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

The markets for gold and silver doré are readily accessed and available to gold producers. Currently, 100% of the gold and silver is delivered to the Istanbul Gold Refinery. Copper precipitate is currently produced from the sulfidisation, acidification, recovery, and thickening (SART) plant and sold into local markets in Turkey. The sulfide plant does not currently include a copper circuit. Provisions have been made in the plant design to include the copper circuit in the future if market conditions warrant.

19.2 Contracts

Anagold contracts the mining operations to a Turkish mining contractor. The contract contains provisions for escalation / de-escalation of fuel prices, foreign exchange rates, haul grade and distance and Turkish inflation. The terms and prices for the mining contract are within industry standards for mining contracts.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Material Impacts

The Çöpler mining and processing operations involve open pit mining from multiple pits, construction of multiple waste dumps to accommodate mined materials, processing of oxide ores and placement on a heap leach pad, and processing of sulfide ores with placement of tailings in a tails storage facility (TSF). These activities and facilities are carried out on treasury, pasture, and forestry lands, including some private lands.

In addition to the direct impacts on the involved lands, the operations impact on the surrounding lands and the local communities. Physical impacts may include changes to local surface and groundwater (including potential pollution), air quality impacts particularly from dust, and increased noise and vibration from mining and processing operations.

Operation of the Çöpler mining and processing facilities, and subsequent mining at Çakmaktepe, has been investigated and authorised by means of a series of Environmental Impact Assessments (EIAs), with positive decisions obtained from the Turkish Ministry of Environment, Urbanisation, and Climate Change (MoEUCC). These EIAs include specific actions designed to address all material impacts of the mining and processing operations. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project.

The original 2008 EIA obtained on 16 April 2008 included three main open pits (manganese, marble contact, and main zones), five waste rock dumps (WRDs), a heap leach pad, a processing plant, and a TSF. The 2008 project description involved only the oxide resources.

The Çöpler project started its open pit and heap leach operation in 2010 and first gold was poured in December 2010. Additional EIA investigations have been submitted and approved, as required, to support ongoing mining and processing operations, including:

- EIA to allow operation of a mobile crushing plant approved 10 April 2012.
- EIA to allow waste dump capacity expansion, oxide capacity expansion to 23,500 tpd and a sulfidisation, acidification, recovery, and thickening (SART) plant approved 17 May 2012.
- EIA to allow the sulfide plant and heap leach area expansion approved 24 December 2014.
- EIA to allow the Çakmaktepe satellite pits expansion approved 26 January 2017.
- EIA to allow a Çakmaktepe capacity increase approved 9 August 2018.
- EIA to allow a second capacity expansion, including heap leach pads 5 and 6. TSF expansion and operation of a flotation plant approved 7 October 2021.

In addition, pending EIA processes include:

- EIA to allow Çakmaktepe second capacity increase to include initial mining from Ardich with EIA description file. The EIA project description file was submitted in October 2020 and a Public Hearing was held in November 2020. All public institutions gave positive feedback regarding the report and the approval process is ongoing with the MoEUCC.

- A review and evaluation meeting was held on the 15 December 2021 for Çakmaktepe EIA and the approval process is ongoing.

After the EIA positive decisions, additional permits and licences were required to be issued by government agencies consistent with the Turkish governing laws and regulations. These include land access permits (pasture and forestry); environmental permits and licences; workplace opening and operating permits; and licences and certificates. The status of project permits and operating licences is documented in Section 3 of this report.

In the period following the receipt of the 2008 EIA permit, Anagold has conducted further technical studies to supplement the Turkish EIA studies and to establish plans and procedures to manage potential project impacts and meet IFC requirements. Significant operational management plans established as a result of these prior and ongoing studies include:

- Non-mining Wastes Management Plan
- Mining Waste Management Plan
- Water Resources Management Plan
- Biodiversity Management Plan
- Soil Management Plan
- Air Quality and Emissions Management Plan
- Mine Closure and Rehabilitation Plan
- Environmental Management System Framework
- Environmental Noise and Vibration Management Plan
- Hazardous Substances Management Plan
- Mine Closure Framework
- Resource Efficiency and Pollution Prevention Management Plan
- Cyanide Management Plan

20.2 Physical Features

The project site is in a transition region between Central and Eastern Anatolian climates. The region has a continental climate, where summers are hot and dry, and winters are cold and relatively humid. Owing to the mountain ranges bordering Erzincan Province on all sides, the region has a milder climate than the neighbouring provinces.

The long-term annual average precipitation for the project site is 367 mm, including snow in the winter months. The annual average wind speed is 2.6 m/s. Maximum wind speeds are observed in spring. The prevailing wind direction is south.

The project site is in a rural area with no significant commercial or industrial air pollution sources. Scattered slag piles and ore extraction sites remain from the former manganese mining operations.

The ambient air quality monitoring programme on-site indicated that SO_2 and NO_2 levels, and particulate matter (PM10) and dust deposition levels in ambient air are well below the limit values defined in Turkish Air Quality Standards. Heavy metal concentrations in dust were well below the limit values defined by European Commission (EC), World Health Organisation (WHO), and Turkish standards.

The railway and the İliç-Kemaliye Road passing near the Euphrates River are the mobile sources of noise in the area. The Euphrates-Karasu River is the largest surface water body near the project; it borders the northern edge of the project area. Peak flow rates are observed in April and May following the snow melt and rainfalls. All other streams in the vicinity of the project area are intermittent, flowing between March–June.

The surface water quality within the site was investigated at various water sampling locations throughout the site. Water quality is classified from class I (very good quality) to class IV (highly polluted, poor quality water). Sampling has indicated class IV water quality for Sabırlı and Çöpler Creeks, and Karabudak Stream. Similarly, the Euphrates-Karasu River is classified as a class IV water resource. For all streams, metal concentrations, including aluminium, iron, copper, and arsenic are high, especially in the drainage from Sabırlı and Çöpler creek catchments. Elevated metal concentrations in these catchments are attributed to natural metallic enrichment from the surrounding geology.

20.2.1 Land Use

The prevalent land use and cadastral information for the Project and its environs is presented in Figure 20.1. The land use patterns are based on maps produced by the General Directorate of Rural Services. As observed in Figure 20.1, most of the project area consists of pastureland, treasury, and forest. The Land Use Capability Classes (LUCC) for the project area and environs is given in Figure 20.2.

Under the LUCC system, there are three main categories and eight classes (ranging between I and VIII).

- The first category covers classes I through IV and describes lands, which are suitable for cultivation and animal husbandry. This category has few limitations, except for class IV, which requires very careful management because of its greater limitations.
- The second category covers classes V through VII, which are unsuitable for cultivation, but which can support perennial plants when intensive conservation and development practices are applied. Under controlled conditions, this land may also support grazing and forestry. The soil type included in class VII has severe limitations, preventing the growth of cultivated plants due to characteristics such as the formation of steep slopes (which are exposed to medium to severe erosion) and shallow soil layers, possessing stony, salty, and sodic texture. As such their utilisation for agricultural purposes is very limited.
- The third category contains only the class VIII, which is suitable only for wildlife, sports, and tourism-related activities.

As shown in Figure 20.2, the project area has VI, VII, and VIII classes of LUCC. The land use types in the project area and its vicinity are:

- Degraded forest lands and coppice
- Barren forest lands
- Agricultural lands
- Settlements

The project area and surroundings are generally of low-land use capability and not suitable for agricultural activities. Although the agricultural activities are limited in the area, there are several small gardens which belong to the local villagers.

The forests in the area are under stress due to high grazing and illegal land use practices; pasture lands are used for the purpose of grazing, but it is illegal to use forestry lands for grazing. In general, the local soil has poor fertility due to its nature and elevation such that it only supports limited species of vegetation.

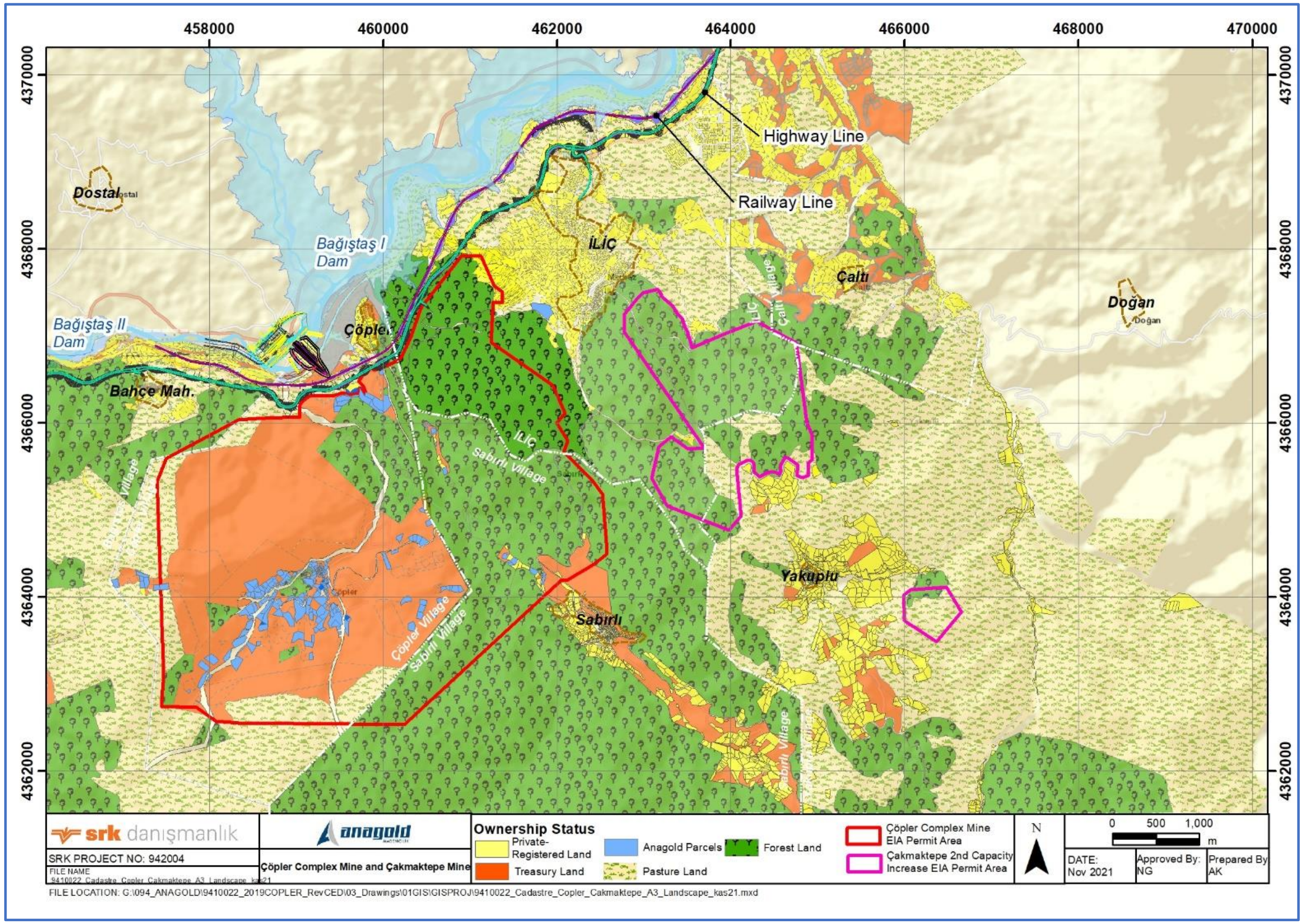
20.2.2 Biological Features

Floral species from the Irano-Turanian and Mediterranean phytogeographic regions are dominantly observed at the site. Most of the flora species are identified in the dry meadow habitats in the project area. Ruderal habitat (such as roadsides etc.) and rocky areas follow dry meadow habitats with respect to the floristic species diversity.

Flora and fauna surveys were conducted in the framework of the 2005–2007 EBS by specialists from Hacettepe University. Biodiversity of the site has been updated by the specialists from Gazi University and Hacettepe University via three seasonal surveys during 2011–2012. A Biodiversity Action Plan (BAP) was prepared, and a BAP Report has been provided as an appendix of the Environmental and Social Impact Assessment (ESIA) Report for the Sulfide Expansion Project. The flora species were classified according to their thread status with respect to Turkish Red Data Book of Plants and the International Union for Conservation of Nature (IUCN) and European Red List (ERL) Categories and Criteria.

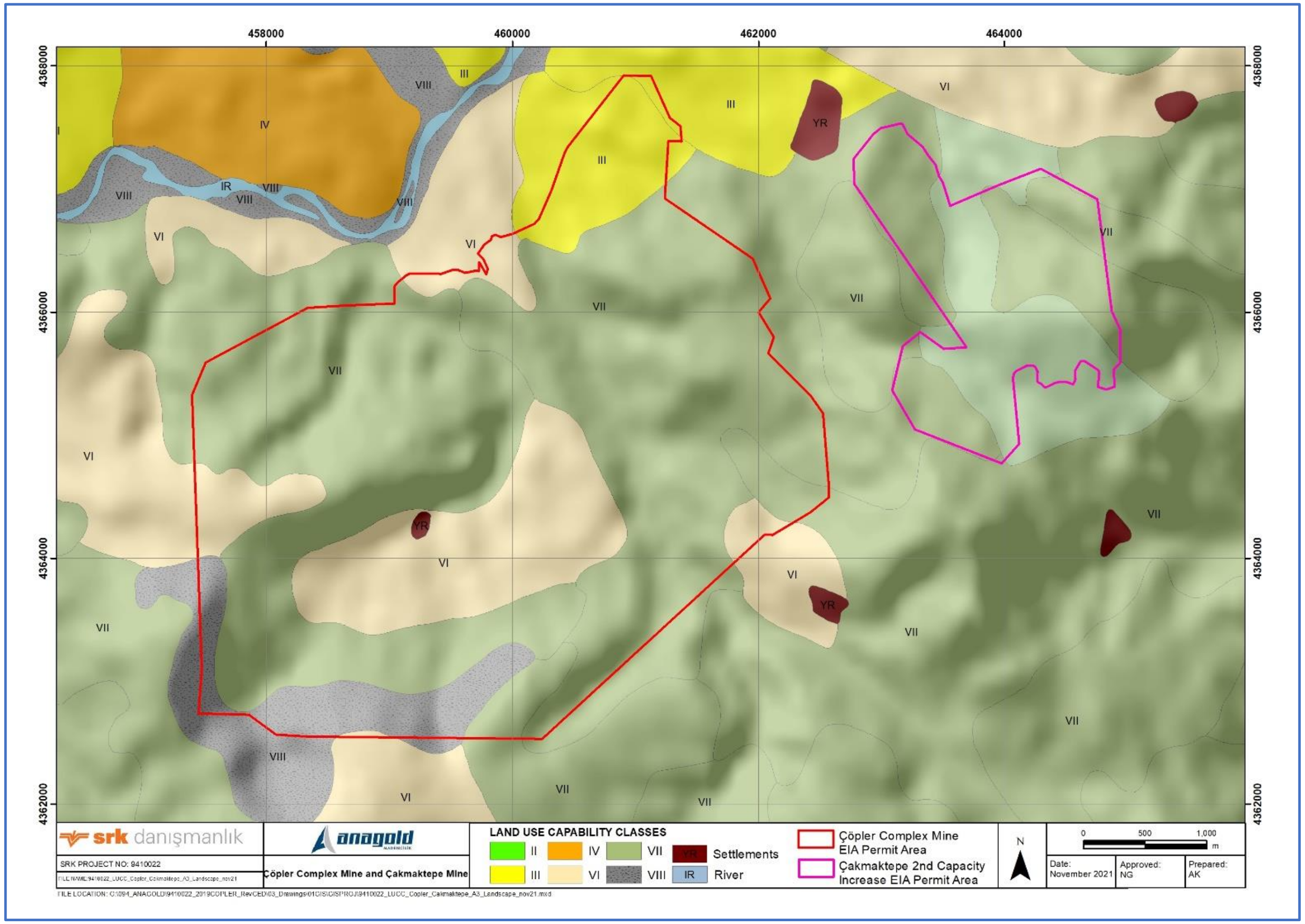
There are four main vegetation types in the area namely: Quercus petraea subsp. pinnatiloba; Quercus libani and Quercus brantii forests; Irano-Anatolian steppe vegetation; and wooded steppes and rock habitat, while the rest of the site is designated for main mining activities. The faunal composition of the site is considered weak.

Figure 20.1 Current Land Use Types and Cadastral Map for the Çöpler Project



Anagold, 2021

Figure 20.2 Land Use Capability Classes (LUCC)



Anagold, 2021

20.3 Social and Community Plans

The EIA studies are conducted according to the format stipulated by the Turkish EIA Regulation. The scope of the Turkish EIA studies differs from the scope of international ESIA studies (as established by the International Finance Corporation's (IFC)'s Environmental and Social Performance Standards), especially in terms of social impacts and public disclosure processes. While the social impact assessment and public disclosure processes are also parts of the Turkish EIA studies, they are treated less rigorously than in IFC standards.

Anagold has conducted further investigations to supplement the Turkish EIA studies, initially to support the original project establishment and, then subsequently, to monitor the social and community attitudes and the impacts of ongoing mining operations on the adjacent communities. The fundamental data to assess social impact is derived from direct survey of the local community members in villages impacted by the mining operation. Significant (primary) surveys have included:

- Initial survey of 51 households in three villages (Sabirli, Bagistas and Dostal) presented collectively as part of the 2009 Çöpler Gold Project Social Impact Assessment (SIA) by KORA.
- Survey of 153 households in six villages (Çöpler, Bagistas, Bahcecik, Dostal, Yakuplu and Sabirli) presented individually performed by Middle East Technical University (January 2013).
- Survey of six villages performed by UDA Consulting (December 2014).
- SIA by SRK (2015).
- Survey by TANDANS Company (2017).
- Çöpler Mine phase 2 SIA Peer Review Report by Intersocial Company.
- Çakmaktepe 2nd Expansion Project SIA Works by SRK (Ongoing).
- Survey by TANDANS Company (Ongoing).

Anagold has considered the outcomes from the community surveys and SIA assessments as a key input to establish and monitor the social action plans associated with the project. These are also the basis to develop a strategic and planned approach to community investment and development programmes. Some significant social and community plans and policies developed as a result of these investigations address the following:

- Community health and safety
- Local employment
- Local procurement
- Community development fund (SKF)
- Donations
- Stakeholder engagement and community relations
- Grievance management
- Environmental and social sustainability

- Training management
- Cultural Heritage
- Land access and resettlement
- Communications

The performance and effectiveness of social and community plans are monitored, reviewed, and updated, as required, to meet changing community needs and expectations.

20.4 Mine Closure

Mine rehabilitation and closure obligations are prepared and updated annually for the Çöpler project. Scheduling and costing of the closure tasks are made in accordance with the Anagold mine plan.

Cost estimates rely on data from mine operations including labour and equipment rates, material costs, groundwater well inventories, and electronic topography data.

Closure costs are estimated using the Standardised Reclamation Cost Estimator (SRCE). The SRCE is an industry standard tool developed to facilitate accuracy, completeness, and consistency in the calculation of costs for mine site reclamation.

SRCE utilises lengths, areas, volumes, flow rates, quantities, etc., provided or estimated by the user (based on the reclamation or closure actions). Some actions require crews and fleets with productivities either provided by the SRCE default settings or those provided by Anagold to estimate the time it takes to perform the work. Where available, these times are then multiplied by labour and equipment rates provided by Anagold.

The Heap Leach Draindown Estimator (HLDE) model is another industry standard tool used for estimating heap leach pad draindown curves for reclamation bonding purposes. The HLDE inputs are derived from site-specific data.

20.4.1 Closure Cost Estimate Assumptions – Waste Rock Dumps

All slopes on the WRDs will be regraded to 2.5H:1V to prepare them for covering, scarification, and revegetation. The sequence of costs in the schedule corresponds to the assumption that reclamation will occur as soon as each WRD reaches final configuration.

Anagold plans to encapsulate all potentially acid-generating (PAG) waste rock within the WRDs as part of mining operations, leaving no PAG material on the surface or outer portions of the WRDs at closure. Therefore, although some PAG cells are currently exposed, costs for construction of a buffer layer encapsulating PAG waste rock are accounted under operational costs and no additional costs for mitigation of current configurations are included in the ARO estimates.

Per the EIA Report, waste rock management will be carried out to allow for the construction of a buffer layer to prevent degradation of seepage and these costs are accounted under operational costs. The seepage collection ponds active during the operations period will be reclaimed during closure. Seepage from the WRDs will not be monitored during closure and post-closure.

20.4.2 Closure Cost Estimate Assumptions – Pits

Berms will be constructed around the perimeter of the pit to discourage public access. There are no other physical reclamation measures assumed for the pit walls.

Rapid refilling of the pits with water is the preferred method for the western part of the pit. Costs for pit refilling by pumping flow of 66 litres per second (L/s) for four-years are included in the ARO estimates.

Some PAG rock will remain exposed in the pit walls after formation of a pit lake; therefore, some reclamation work will be necessary to address the requirement (legal obligation) to cover remaining PAG materials exposed in the pit after mining ceases.

It is assumed that areas within the pit where PAG materials are exposed will be covered with 1 m of non-PAG (or non-acid-generating – NAG) material. The PAG materials exposed within the pit walls are assumed to be located on gentle or nearly-flat slopes. Additional measures (e.g., reduction of pit wall slopes in exposed PAG areas to facilitate cover placement) are not taken into consideration at this time. No PAG cover will be required below the final pit lake elevation.

20.4.3 Heap Leach Pad

All slopes on the heap leach pads will be regraded to 2.5H:1V or flatter to establish a geotechnically stable closure configuration. Following regrading, the areas will be covered, scarified, and revegetated. The ARO estimates reflect the requirement per the EIA report that identifies 2–3 m of cover placement on the heap leach pad followed by growth medium placement after the reduction of heap and pond fluid inventory.

Although not a requirement in the EIA plan, there is a provision for extending half of the heap leach pad perimeter liner to contain heap material regraded beyond the existing liner during reclamation.

East and west buttresses are considered part of the heap leach pad area. The physical reclamation of this area by growth media placement and revegetation is included as a WRD.

The 2014 EIA discusses rinsing of the heap with fresh water with no subsequent fluid management. Rinsing of heap leach pads has been shown to be typically unnecessary and potentially detrimental to long-term chemical stability of gold heap leach.

Per the approach of the HLDE model mentioned above, heap drain-down will be initially managed for inventory reduction via recirculation and active evaporation, followed by active evaporation only. Active evaporation will continue until drain-down flows are reduced to a rate amenable to management with passive evaporation.

Following active solution management, when the heap drain-down flow rate decreases to a level where it can be managed exclusively within available emergency and process pond via passive evaporation, the two ponds will be converted to evapotranspiration (ET) cells. To convert process ponds to ET-cells, the ponds will require relining followed by backfilling with select material and revegetation.

Conversion costs are calculated based on experience from multiple Nevada sites.

In scheduling costs, the cost of construction of ET-cells is included at a time when drain-down rates reach a level that will allow fluid to be managed through the evapotranspirative capacity of ET-cells.

20.4.4 Tailings Storage Facility

Anagold submitted an EIA in 2014 that included TSF 1 and TSF 2. The current designs for TSF 1 and TSF 2 are within the 2014 EIA boundaries, except for a small portion of TSF 1 phase 7. TSF 1 phase 3 has been constructed and approved for use in October 2021 by the MoEUCC. The current mine plan only requires construction of TSF 1. Long-term management costs are included in the estimate and proportioned for the size of the TSF construction.

Reclamation of the life-of-mine (LOM) TSF includes the following actions:

- Reclamation of the TSF surface by placing a traffic layer and growth media followed by revegetation.
- Reclamation of the final TSF embankment.
- Fluid management including managing drainage from the TSF and removal of water ponding on the TSF surface due to consolidation of the tailings.

The estimate includes costs for placement of a traffic layer over the tailings material in addition to the growth media layer. The starter embankment is built at 1.5H:1V with the final embankment at 2.0H:1V. The costs of placing 1 m cover over the embankment are also included.

Costs are included for tailings fluid management crews, pumping for recirculation and forced evaporation, as well as removal of the supernatant in the period soon after the TSF operations end.

20.4.5 Other

SRCE estimates costs to demolish buildings using productivities in conjunction with building volumes, wall areas, and slab volumes. Decontamination costs are included in the estimate for a decontamination crew to pressure-wash the plant site over a nominal number of weeks.

Production wells are assumed to be closed at the end of operation of the sulfide plant and monitoring wells are assumed to be abandoned at the end of the post-closure monitoring period.

20.4.6 Monitoring

The water quality and flow monitoring schedule during the operation, closure and post-closure monitoring period includes numbers of samples, frequencies, and durations for each closure phase. The monitoring locations include the groundwater monitoring wells around the heaps, WRDs, TSF and springs as well as pit lake water quality once the rapid filling begins.

20.4.7 Closure Planning

Closure planning costs are typical industry costs for development of closure plans and studies, reporting and preparation of closure designs and engineering.

20.4.8 Construction Management

Construction management costs include one supervisor during active reclamation. Costs are included for road maintenance, which will be carried out with a water truck and grader during active reclamation.

20.4.9 Human Resources

Closure personnel include a closure general manager, environmental manager, environmental technician, security, and surveyor for whom terminal benefits are included. Under the LOM schedule, the closure general manager would be present during the years of active reclamation and closure. Camp costs are included under general and administration costs.

For solution management, the cost of the heap drain-down management crew is assumed to be shared with those of the TSF. The annual Asset Retirement Obligation reports for EOY'20 and EOY'21 have been completed.

20.4.10 Closure Schedule

The EOY'20 closure was scheduled separately for the oxide and sulfide projects according to the mine plan and is consistent with the long-term management obligations expected for the TSF.

Heap drain-down management starts at the end of heap leaching operations in the mine plan. Ore will be sent to the leach pad until the end-of-2030, although at a reduced rate after 2020. Management and reclamation on the heap will take place while other components of the Çöpler sulfide project continue to operate, with the active closure period starting after the end of deposition in the TSF.

20.4.11 Further Work

There may be an opportunity to utilise the heap drain-down solution in the sulfide circuit rather than disposing of it by forced evaporation, potentially reducing costs. This will require changes to the design of the evapotranspiration cells included in the current estimate.

Further studies and design work are required for the mitigation of PAG materials exposed in the pits to verify whether the proposed 1 m of non-PAG cover is practical and effective to implement.

The growth media inventory and expected amount to be recovered over the course of the project should be compared to the sum of the growth media requirements of the project facilities. Further work is required to determine the most sustainable revegetation covers to be employed.

20.5 Sustainability

Anagold aims to provide sustainability governance that not only meet or exceed the requirements of Turkish legislation, but also align with the expectations of ICMM (International Council of Mining & Metals) guidance and International Finance Corporation (IFC) Performance Standards, and the World Gold Council. The Anagold approach to policy development is to identify the most stringent standards and integrate them into project policy.

Çöpler project policies are supplemented by site-specific environmental and safety standards, management plans and procedures that are specifically tailored to the unique environmental and social challenges and permitting regulations of the site. These plans are certified to the requirements of international standards including ISO14001: 2015 and ISO45001.

Anagold maintains annual sustainability reporting for the project, the report is produced to be in accordance with GRI Standards. The 2020 Sustainability Report has been completed and is publicly available. The 2021 Sustainability Report is currently under development.

Anagold has a dedicated Environmental, Health, Safety and Sustainability (EHS&S) Committee. The EHS&S Committee oversees, monitors, and reviews practice and performance in areas of safety, health, stakeholder relationships, environmental management, and other sustainability issues.

Sustainability is also a key responsibility for group level executives and site teams. The approach to sustainability is underpinned by the principle of collective responsibility and a belief that every employee must contribute to our sustainability performance – particularly on issues of health and safety and reporting of incidents.

20.5.1 Stakeholder Engagement

At the Çöpler project, Anagold has a wide-ranging stakeholder engagement programme which sets out the ways in which Anagold engages with stakeholders and ensures regular communication with stakeholder groups.

During 2021 stakeholder consultations included meetings with shareholders, analysts, local communities, local and national authorities, contractors, government representatives, NGOs, universities, political parties, and trade union officials. Some of the key topics discussed included the Mine Expansion Project, Social Development Fund, exploration activities, cyanide and environmental awareness, local procurement, local contracting opportunities, training, and job creation.

The grievance mechanism is an important part of the Anagold local stakeholder engagement programme and the overall governance of sustainability. The community grievance mechanism has been developed to meet the requirements of both Turkish regulations and the IFC Performance Standards. The mechanism is designed to be widely accessible and there are access points available throughout each of the affected communities. There is also a dedicated access point for suppliers.

20.5.2 Health and Safety

Health and Safety Policy is guided by two key goals. First, to eliminate fatalities and serious injuries from our operations, and second, to continually reduce the number of minor injuries occurring on-site. To fulfill these goals on the ground we implement:

- Robust systems and plans
- Risk assessment and controls
- Employee engagement
- Training

Anagold measures safety performance by tracking a range of leading and lagging safety indicators, the safety statistics reported also include exploration activities. All significant incidents are investigated and, based on findings, corrective action plans are developed to prevent recurrence.

20.5.3 Training and Development

The approach to the development of people is to strategically and continuously invest in staff training to ensure the business and operational needs both now and in the future are met. The development opportunities provided include technical skill development, leadership and business literacy skills, procedures and standards, and career development for staff. Çöpler has a specialised training centre with a capacity of 150 trainees.

Anagold carries out training and capability development programmes for our neighbouring community. Training is directed to future roles with the project, while other training is focused on general skills development to enable people to seek gainful employment in other industries and locations throughout Turkey. This will help to broaden the economy and skills base in the Iliç District.

20.5.4 Industrial Relations

The workforce has no restrictions on union representation. Approximately 60% of the workforce at the Çöpler project are union members and have collective agreements in place. There have been no instances of industrial action.

20.5.5 Diversity and Inclusion

Anagold does not set diversity or gender quotas for the workforce. Personnel are appointed based on merit and have specific objectives in place to ensure that the candidate pools for any position available throughout the company are made up of a range of qualified and diverse candidates. Women are paid equal with men in similar positions. The Anagold Diversity Policy commits the project to provide:

- An environment in which all employees are treated with fairness and respect; and
- Equal access to opportunities - regardless of race, gender, sexual orientation and/or religious beliefs.

The approach to recruitment is to first look to local communities with appropriate skills. If unsuccessful, this is followed by recruiting from the wider region, followed by nationally, before finally looking internationally. The Anagold commitment to employing and developing local and national workers is reflected by the targets set for the Çöpler project:

- 90% of unskilled workers to be drawn from communities impacted and affected by Anagold operations.
- 80% of semi-skilled worker to be drawn from impacted and affected communities.
- 80% of skilled workers to be Turkish citizens.

Suppliers are also encouraged to employ local workers whenever possible.

Local supply chains are preferred. Where supplier skills are lacking Anagold work with the suppliers to build capacity by providing training and mentoring.

20.5.6 Sustainable Community Development

To promote economic development in the communities neighbouring the Çöpler mine a Social Development Fund (SDF) was established in 2018. The SDF provides a structure under which Anagold will work in partnership with communities neighbouring the Çöpler mine, applicable Government agencies, third-party development partners and other relevant stakeholders, with the objectives of:

- Ensuring Anagold's SDF funding of community programmes and projects is managed and distributed in a fair, transparent, and equitable manner.
- Building capacity within the local communities to participate in the benefits afforded by the mine and related regional economic and social development more actively.

- Moving away from donations type community relations expenditure by developing sustainable projects and programmes which address agreed social and community development priorities in the areas of agriculture, health, education, non-mine related income generation, and empowerment of underrepresented and disadvantaged groups.
- Where appropriate, reviving and promoting traditional customs and practices.
- Promoting independence from Anagold operations and assisting the communities to prepare for life beyond mining.
- Where appropriate, community relations expenditure by developing sustainable projects and programmes which address agreed social and community development priorities and/or benefit of public such as infrastructure, renovation, sponsorship, and construction.

Anagold will work with the community and other development partners in a manner that reflects the core values and principles of the SDF which include:

- Fairness and Equality – Impartial administration of the SDF, with all sectors of the SDF communities treated equally.
- Transparency – Clear, publicly available processes for how the SDF is managed, and timely and fulsome reporting of decisions that are made, including financial reporting everyone has access to the same information.
- Cooperation and Partnership – Anagold working with the Community to focus on agreed development priorities. The SDF will not initiate programmes that are not requested by the community and in which the community do not have active and meaningful participation.
- Mutual respect – Everyone has a right to be heard and their opinion considered.
- Sustainability – Focusing on what counts over the long term and preparing for life beyond mine closure.
- At all times being fully compliant with relevant Turkish and International laws and conventions, and Anagold corporate policies and commitments.

While recipients of the SDF expenditure are the communities neighbouring the Çöpler mine, Anagold will retain ownership and governance control over all aspects of Anagold's financial and in-kind contributions to the SDF. Anagold's contribution to the SDF includes direct financial support, managerial/administrative support, and limited technical support.

Direct financial support has been approved by Anagold's partners (SSR and Lidya) for ongoing annual funding to the SDF of $2 per ounce of gold produced from the Çöpler orebody. The SDF will replace a substantial proportion of Anagold's existing discretionary community expenditure and direct funding towards development proprieties which are agreed with the community. The continuation of Anagold's support to the SDF is at Anagold's discretion, and will be influenced by, among other things, the success of the SDF and the community's participation in ensuring the objectives of the SDF are achieved.

Managerial and administration support will be provided to the recipients of the SDF and Anagold's policies, procedures, and management plans. Anagold will also cover the costs associated with stakeholder communication and consultation during the roll-out of the SDF, including support for the first three years in establishing a help-desk facility for SDF applicants to receive assistance in preparing their applications.

While support to the SDF applicants on how to apply and administer their applications and projects will be available through a dedicated SDF help-desk, where appropriate, and where relevant skills exist within the Company (and timing permits), Anagold will also support the SDF applicants with limited ad-hoc technical support as projects are being developed, and during the implementation phase. However, where a project requires specific and ongoing technical support, project applicants must ensure this is identified and resourced appropriately using third-party technical resources.

Anagold's intensions for the SDF initiative are based on good-will and respect for its neighbouring communities, however, Anagold acknowledges that other individuals, organisations, and government agencies may be more skilled and adept at identifying and implementing social and community development programmes and projects. As such it is Anagold's desire that the SDF be implemented in such a way that third parties are attracted to participate in supporting community based SDF initiatives. In this way, the SDF can realise a greater funding base as well as attract leading skills in social and community development programme implementation. Third-party partners can include organisations providing development support or financial support including Government agencies, NGOs, or other credible development organisations. The SDF will not be used to fund third-party projects outside the approved SDF catchment area. Where a third-party partnership is part of an SDF application, the working relationships between Anagold, project applicants, and third-party partners must be clearly detailed in the project application. Details of these relationships will form part of the application review process and be thoroughly scrutinised with respect to Anagold's FCPA policy.

While Anagold's annual contribution to the SDF is substantial, not every project will receive funding. The SDF will be established to focus on participatory needs-based development priorities which support the above-mentioned purpose. It is proposed that development priorities will be re-assessed every three-years.

20.5.7 Environmental Management

Anagold's commitment to responsible environmental management is set out in the Environmental Policy, which complies with in-country legislation, the IFC Performance Standards, and the Equator Principles. The Çöpler Environmental Management System (EMS) is certified to the international ISO14001: 2015 standard. The latest ISO14001: 2015 external audit was completed successfully in December 2019.

20.5.8 Water Risk

The Çöpler project is in a high desert region in Eastern Turkey near the culturally significant Euphrates River. All water used at Çöpler is governed by strict permitting rules regarding abstraction and discharge under Turkish regulations. The approach to water management is to use water as efficiently as possible and to only draw as much needed and allowed within permitted limits. All water abstract is groundwater. Water used on-site is recycled and re-used in the process plant. Water is not discharged to the environment.

20.5.9 Energy and Climate Change

All the electricity the Çöpler project uses is drawn from the Turkish national grid. Approximately 41% of Turkey's national grid capacity comes from hydropower stations. The treatment of sulfide ore requires a more energy and CO2 intensive process than the oxide ore process that was previously the only ore treated at the Çöpler project. Anagold plan to use 2019, 269 GWh, as the baseline year for electricity use and efficiency, and to set targets based on 2019. The greenhouse gas emissions are published in the Anagold sustainability report.

20.5.10 Tailings Dam Management

Tailings produced by the Çöpler project are classified as class II non-hazardous. All tailings are sent to a carefully engineered TSF. Anagold has procedures in place to ensure that all parts of the TSF life cycle from construction to closure align with international best practice standards.

The TSF at the Çöpler project is a downstream mass filled dam. It became fully operational during the final quarter of 2018 with the start-up of the sulfide plant. The technical specifications for the construction of the Çöpler project TSF conforms with both Turkish national requirements and accepted good practice standards for tailings facilities, including:

- World Bank Standards
- Canadian Dam Association Safety Guidelines
- ICOLD (International Commission on Large Dams) Bulletins
- Turkish Hydraulic Works' Technical Codes
- Mining Association of Canada (MAC) Guide to the Management of Tailings Facilities.

The Çöpler project TSF has been designed to withstand significant earthquakes up to a magnitude of 7.5 on the Richter scale. Modelling showed that even in the most severe seismic event, the wall of the TSF will heave with minimal risk of altering facility location or strength. There are no communities living directly downstream of the Çöpler project TSF.

The TSF uses a combination of technology, regular inspections and external oversight and audits to monitor the Çöpler project TSF (see Section 18.10.3).

In addition to stability designs and monitoring, Anagold also has three groundwater monitoring wells in place both above and below the Çöpler project TSF, to monitor for signs of groundwater contamination. It was designed to meet the best in class requirements for class I (hazardous) waste, even though all tailings are classified as class-II (non-hazardous).

20.5.11 Water Management

The process of removing ore from the ground and extracting gold creates significant non-hazardous and some hazardous waste, which must be appropriately dealt with over the long- and the short-term. Ensuring all waste is responsibly dealt with is crucial to protecting the health of the local environment and neighbouring communities.

To ensure that all waste, whether hazardous or non-hazardous, is reduced and dealt with in a safe and responsible manner, the Çöpler project has a detailed and comprehensive waste management plan. This is underpinned by the goal to reduce the amount of waste generated and to maximise the proportion of waste sent for recycling.

The bulk of the waste created at the Çöpler project is waste rock. All the waste rock created by the Çöpler project is carefully disposed of in engineered waste rock dumps. The design and management of all waste rock dumps is overseen by geotechnical engineers to ensure they have safe slope angles, maximum structural stability and management of any potentially acid forming materials are conducted appropriately by mine operations and thus meet the requirements of Turkish national regulations, industrial best practices and the IFC Performance Standards.

20.5.12 Cyanide Management

The use of cyanide is a critical part of the gold mining process. However, if not handled correctly, cyanide can have significant impacts on both environmental and human health. The use of cyanide at the Çöpler project is governed both by the requirements of Turkish national laws and regulations and aligned with industrial best practice. All employees and contractors who handle, transport, or dispose of cyanide are required to undertake specialised training in cyanide handling.

20.5.13 Biodiversity

The size, scale and location of mining operations means they can have a negative impact on local biodiversity. Failure to manage these risks and minimise the impacts on biodiversity could affect the social licence to operate and reputation. The Anagold aim is to restore sites (both operational and exploratory) and repair any damage done to the extent practicable. To do this, detailed records of the full range of biodiversity present as part of feasibility studies of any project or expansion. These studies form the basis for a Biodiversity Action Plan (BAP). The BAP sets out how impacted ecosystems are to be restored to their original state (or as close as possible) at the time of closure. Both the Çöpler project, its associated TSF and prospects have Biodiversity Action Plans in place. Anagold also conducts biodiversity monitoring studies each quarter with experts from Gazi and Hacettepe Universities.

20.5.14 Air Quality

There is a potential for dust to be generated across many parts of the operation, including blasting, crushing, and milling, and the movement of large vehicles on haul roads. Dust management is a key focus across all facets of the operation. Air quality and the presence of dust is an important factor for local communities and workers. Ensuring management air quality for workers and communities is an important part of environmental management. Anagold has put in place a dust management plan at the Çöpler project to minimise the levels of dust in the air and ensure they fall within Turkish and IFC guideline limits. There are several monitoring stations across site and in the local communities. These stations record levels of airborne particulate matter and dust fall out. The results from the monitoring stations are reported to the relevant national authorities, and to local communities.

21 CAPITAL AND OPERATING COSTS

Capital and operating cost estimates have been developed based on the current project costs, the mine and process designs, and discussions with potential suppliers and contractors. The estimated capital costs are to a feasibility level of accuracy and include a contingency of 10%.

21.1 Capital Costs

Growth capital costs in the Reserve Case includes costs for:

- Ardich establishment and mine development
- Heap leach phases 5 and 6
- Road relocation, studies, and project management
- Explosives magazine relocation

Sustaining capital in the Reserve Case includes costs for:

- Tailings storage facility (TSF)
- Project team
- Technical services
- Administration
- Assay laboratory
- Mining
- IT
- Sulfide processing
- Oxide processing
- Environment
- Mineral / lands rights
- Health & safety
- Security
- Supply chain
- Reclamation

Capital costs assumptions to the end of 2021 and for the life-of-mine (LOM) are shown in Table 21.1.

Table 21.1 Capital and Reclamation Costs

Description	Unit	Total LOM
Oxide		
Growth	$M	69
Sustaining	$M	29
Sulfide		
Growth	$M	–
Sustaining	$M	413
Reclamation and Other		
Reclamation	$M	114
Working and Other	$M	–37
Total	$M	**588**

21.2 Operating Costs

Operating costs were estimated based on current site cost performance and contract costs including actual operational costs for labour, consumables, contracts and the Anagold budget assumptions. The projected LOM unit operating cost estimate is summarised in Table 21.2 and the average costs are shown in Table 21.3.

Table 21.2 Average Operating Costs Unit Rates

Activity	Unit	LOM Average Unit Cost
Mining	$/t mined	1.62
Processing – Heap Leach	$/t HL processed	14.45
Processing – Sulfide	$/t sulfide processed	35.91
Site Support and Office	$/t ore processed	5.21

Table 21.3 Summary of LOM Average Operating Costs

Cost	Total LOM ($M)	5-Year Average per year ($/t ore)	LOM Average per year ($/t ore)
Mining	766	14.98	10.15
Process	2,225	27.79	29.49
Site Support and G&A	473	7.14	6.27
Operating Costs	3,464	49.91	45.91

Mining costs include waste stripping costs

21.3 Mining Cost Summary

The mining costs were applied to the financial model as operating costs or capital costs. In the mining cost model, costs are broken down into specific areas including drill and blast, load and haul and rehabilitation.

Mining operations for the mine are currently contracted to a Turkish mining contractor. No capital cost is included for mining equipment or facilities. All such costs are built into the unit rate for mining operations included in the operating cost estimate.

Mining operating costs include:

- Drill and blast
- Load and haul
- Labour
- Dewatering
- Other indirects

Mining capital costs include:

- Fixed equipment
- Mobile equipment
- Office and supply
- Mine rehabilitation
- Studies

21.4 Processing and Infrastructure Cost Summary

The following has been included in the costs for processing:

- Oxide processing
- Sulfide processing
- Waste management
- TSF
- Utilities and services
- Reagents
- Plant infrastructure
- Plant mobile equipment

The following has been included in the capital costs for infrastructure cost estimates:

- Bulk services
- Site preparation

- Buildings and structures (new and refurbished)
- Communications
- IT hardware and software
- Security and access control
- Site costs
- Mobile equipment
- Services contracts
- Community support

The following has been included in the operating cost estimates:

- Plant consumables
- Crusher consumables
- Screens
- Grinding media
- Filters
- Packaging plant bags
- Plant reagents
- Plant mobile equipment
- Plant maintenance
- Power
- Labour
- Production and dispatch
- Plant and infrastructure day work services
- Plant technical services
- Shift maintenance
- Laboratory service level agreement
- TSF water treatment

21.5 General and Administration Cost Summary

The General and Administrative (G&A) costs include costs not directly attributable to operational output such as the mining and processing operations. The following costs have been included in total G&A cost:

- Office and general expenses
- Site support costs
- Off-site Anagold offices

- Internal and external consultants
- Maintenance and inspection contracts
- Equipment and sundry
- Fuels and utilities
- Rentals and leases
- Insurance and insurance taxes
- IT hardware and software
- Personnel transport
- Communications
- Licences and land fees
- Labour
- Accommodation and messing
- Medical support
- Flights
- Light vehicles
- Environmental, community development and engagement
- Banking and audit fees
- Legal

22 ECONOMIC ANALYSIS

22.1 Reserve Case Economic Analysis Results

The Reserve Case production includes 22.5 Mt at 1.69 g/t Au oxide ore processed by heap leaching and 52.9 Mt at 2.33 g/t Au processed in the sulfide plant. Total production is 75.4 Mt at 2.14 g/t Au. Total gold production is 4.4 Moz. Mining at the Çöpler pit is completed in 2029 and at Ardich in 2034. Oxide heap leach stacking is completed in in 2034, while sulfide processing will continue from stockpiles until 2042.

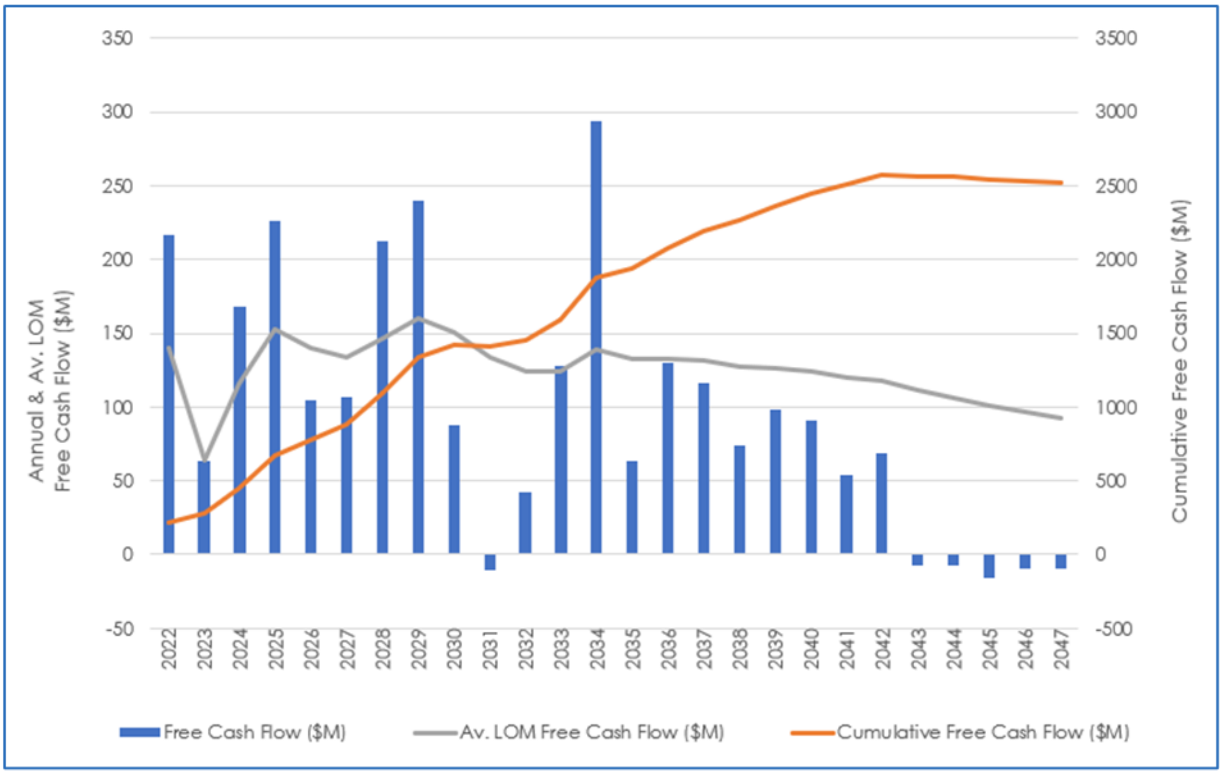
The Reserve Case results include:

- After-tax NPV at a 5% real discount rate is $1.73 billion.
- Mine life of 21 years.

An IRR is not reported as the operation is cash positive in each year of the mine plan until closure. The Reserve Case average all-in sustaining cost (AISC) is $966/oz gold. Key results of the Reserve Case economic analysis are shown in Table 22.1.

The after-tax cash flow is shown in Figure 22.1. The NPV results for before and after-tax over a range of discount rates is shown in Table 22.2. The sulfide and oxide production profiles are shown in Figure 22.2 and gold production in Figure 22.3. Cash costs are shown in Table 22.3.

Figure 22.1 CDMP21TR Reserve Case After-Tax Cash Flow



OreWin, 2022

Table 22.1 CDMP21TR Reserve Case Results Summary

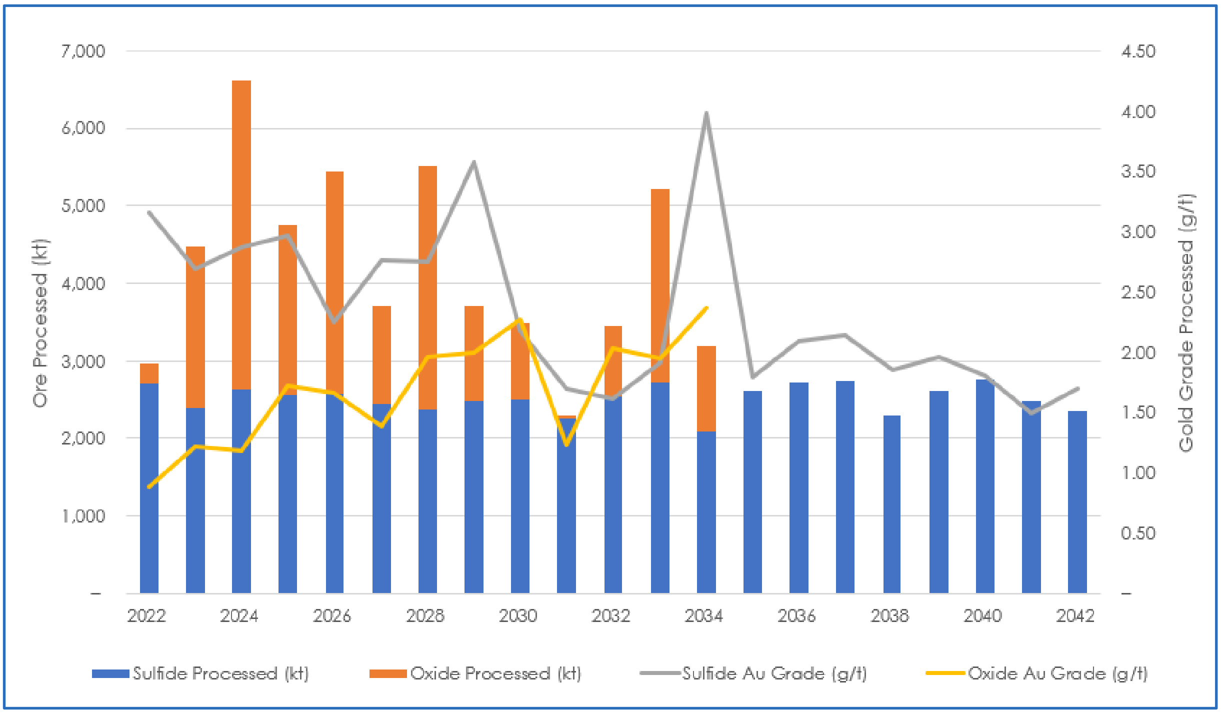
Item	Unit	Reserve Case
Oxide Processed		
Heap Leach Quantity	kt	22,557
Au Feed Grade	g/t	1.69
Sulfide Processed		
Quantity Milled	kt	52,892
Au Feed Grade	g/t	2.33
Total Processed		
Processed	kt	75,448
Gold Feed Grade	g/t	2.14
Total Gold Produced		
Oxide – Gold	koz	765
Sulfide – Gold	koz	3,604
Total – Gold	koz	4,369
Oxide – Gold Recovery	%	61
Sulfide – Gold Recovery	%	91
5-Year Annual Average		
Average Gold Produced	kozpa	278
Free Cash Flow	$Mpa	158
Total Cash Costs (CC)	$/oz gold	880
All-in Sustaining Costs (AISC)	$/oz gold	1,071
Key Financial Results		
LOM Total Cash Costs (CC)	$/oz gold	803
LOM All-in Sustaining Costs (AISC)	$/oz gold	966
Site Operating Costs	$/t treated	45.91
After-Tax NPV5%	$M	1,732
Mine Life	years	21

5-Year Annual Average is for the period 1 January 2021 through 31 December 2025

Table 22.2 CDMP21TR Reserve Case Before and After-Tax NPV

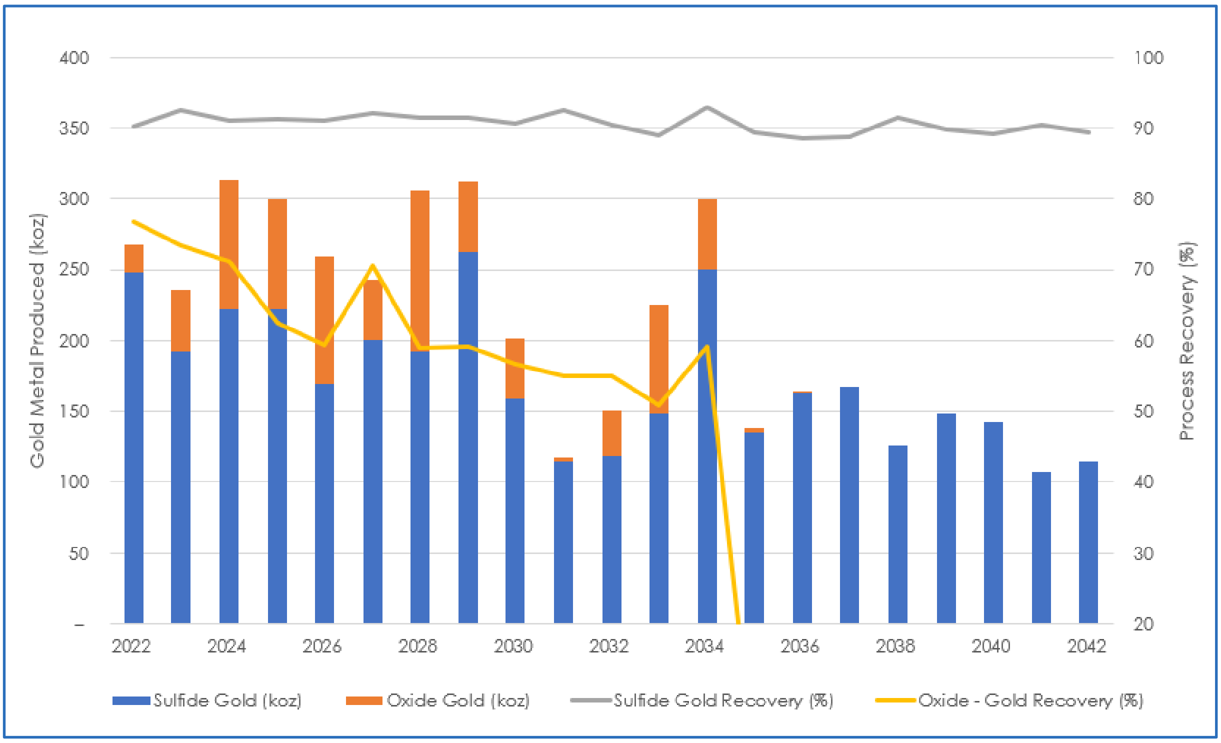
Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	2,729	2,555
5%	1,824	1,732
10%	1,322	1,268
12%	1,185	1,140

Figure 22.2 CDMP21TR Reserve Case Processing



OreWin, 2022

Figure 22.3 CDMP21TR Reserve Case Gold Production and Recovery



OreWin, 2022

Table 22.3 CDMP21TR Reserve Case Cash Costs

Description	Units	Reserve Case
Mining and Rehandle	$M	766
Process, Freight, and Refining	$M	2,031
Site Support	$M	393
Royalties	$M	353
Total Production Costs	$M	3,543
Total Cash Costs (CC)	$/oz gold	803
Sustaining Capital	$M	442
Fixed Lease Payments	$M	192
Site G&A	$M	81
All-in Sustaining Costs (AISC)	$M	4,257
All-in Sustaining Costs (AISC)	$/oz gold	966

Process, Freight, and Refining includes by-product credits and excludes fixed lease costs.
Royalties are calculated in the period incurred and applied to cash flow in the subsequent year.

A financial model was prepared using the Reserve Case production schedule and operating and capital assumptions on an annual basis. The assumptions for taxes and royalties were provided by SSR. The corporate tax rate in Turkey is 23% in 2022 but will revert to 20% from 2023. The royalty rate for precious metals under Turkish Mining Law is variable and tied to metal prices. As Çöpler ores are processed on-site, the applicable royalty rate for POX processing is subject to a further 40% reduction for certain qualifying operating costs. The average royalty calculated as a proportion of gross revenue in the Reserve Case is approximately 4.9%.

Metal prices were estimated after analysis of consensus industry metal price forecasts and metal prices used in other comparable studies. The prices used for the economic analysis are shown in Table 22.4.

Table 22.4 CDMP21TR Reserve Case Metal Price Assumptions

Metal Price	Units	2022	2023	2024	2025	Long-Term
Gold	$/oz	1,800	1,740	1,710	1,670	1,600
Silver	$/oz	24.00	23.00	22.00	21.00	21.00
Copper	$/lb	4.00	3.80	3.80	3.80	3.40

The estimates of cash flows have been prepared on a real basis with a base date of Q4'21 and a mid-year discounting is used to calculate NPV. All monetary figures have a base date of Q4'21 with no allowance for escalation and are expressed in US dollars (US$) unless otherwise stated.

The after-tax NPV sensitivity to metal price variation is shown in Table 22.5 for gold prices from $1,000–$2,000/oz. Cost sensitivity is shown in Table 22.6.

Table 22.5 CDMP21TR Reserve Case Gold Price Sensitivity

After-Tax NPV ($M)	Long-Term Gold Price ($/oz)							
Discount Rate	1,000	1,200	1,350	1,400	1,600	1,750	1,800	2,000
Undiscounted	967	1,546	1,974	2,104	2,555	2,890	2,985	3,398
5%	769	1,115	1,370	1,447	**1,732**	1,939	1,998	2,252
10%	645	866	1,029	1,079	1,268	1,405	1,444	1,611
12%	608	796	935	977	1,140	1,257	1,291	1,435
15%	563	712	822	856	987	1,082	1,110	1,226
18%	525	646	735	762	870	948	970	1,066
20%	504	610	687	711	806	875	894	979

Table 22.6 CDMP21TR Reserve Case Cost Sensitivity

			Change from Base NPV5% ($M)				
Variable	**Units**	**Base Value**	**–20%**	**–10%**	**0%**	**10%**	**20%**
Capital Cost	$M	575	1,790	1,761	1,732	1,702	1,671
Mining Cost	$/t mined	1.62	1,833	1,782	1,732	1,681	1,630
Processing Cost	$/t treated	29.49	1,957	1,845	1,732	1,618	1,503
Site Operating Cost	$/oz Au	108	1,785	1,758	1,732	1,705	1,679
Gold Royalty	$/oz Au	81	1,769	1,751	1,732	1,713	1,694

22.1.1 Project Cash Flow

The after-tax cash flow and average LOM AISC unit cost is shown in Table 22.3.

The annual revenue, operating cost and capital costs and net cash flow is tabulated in Table 22.7.

Table 22.7 CDMP21TR Reserve Case Cash Flow

Item	TOTAL	Year																						
		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044+
	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M
Gross Revenue	**7,154**	**488**	**419**	**549**	**505**	**422**	**392**	**496**	**503**	**325**	**189**	**243**	**365**	**482**	**222**	**262**	**267**	**202**	**238**	**229**	**172**	**184**	**–**	**–**
Realisation Costs																								
Freight & Refining	19	1	1	1	2	1	1	2	1	1	0	1	1	1	1	1	1	0	1	1	0	0	–	–
Royalties	353	23	27	19	29	26	20	17	27	27	14	4	8	18	29	7	10	11	7	9	8	5	6	–
Total – Realisation Costs	**373**	**24**	**28**	**21**	**30**	**28**	**21**	**19**	**29**	**28**	**15**	**5**	**9**	**19**	**29**	**8**	**11**	**11**	**8**	**10**	**9**	**5**	**6**	**–**
Operating Costs																								
Mining	766	47	64	94	79	79	80	82	74	40	41	41	34	10	–	–	–	–	–	–	–	–	–	–
Processing – Heap Leach	326	13	24	53	32	42	18	45	18	14	0	13	36	16	1	1	–	–	–	–	–	–	–	–
Processing – Sulfide Plant	1,899	115	105	113	88	89	85	82	86	87	79	88	94	73	90	95	95	80	91	96	86	82	–	–
Site Support	393	32	32	31	27	27	27	27	23	23	19	19	19	19	9	9	9	9	9	9	9	9	–	–
G&A	81	5	5	5	5	5	5	5	5	5	5	5	5	5	2	2	2	2	2	2	2	2	–	–
Total – Operating Costs	**3,464**	**213**	**230**	**296**	**231**	**241**	**215**	**242**	**206**	**169**	**144**	**166**	**188**	**123**	**102**	**106**	**105**	**90**	**101**	**106**	**96**	**92**	**–**	**–**
Operating Surplus	**3,318**	**251**	**161**	**233**	**244**	**153**	**156**	**235**	**269**	**128**	**30**	**72**	**168**	**340**	**90**	**148**	**151**	**100**	**129**	**113**	**67**	**86**	**–6**	**–**
Total – Capital Costs	**588**	**25**	**81**	**65**	**12**	**49**	**49**	**12**	**12**	**29**	**29**	**19**	**25**	**25**	**25**	**12**	**14**	**14**	**12**	**11**	**6**	**1**	**1**	**58**
Net Cash Flow Before Tax	**2,729**	**226**	**80**	**168**	**232**	**104**	**107**	**223**	**257**	**98**	**0**	**53**	**143**	**315**	**66**	**136**	**137**	**86**	**116**	**101**	**61**	**85**	**–7**	**–58**
Tax	174	6	2	3	4	4	4	7	9	4	–	1	5	12	2	5	25	16	22	19	10	15	–	–
Net Cash Flow After Tax	**2,555**	**220**	**78**	**164**	**228**	**100**	**103**	**216**	**248**	**95**	**0**	**52**	**138**	**303**	**64**	**131**	**112**	**70**	**94**	**82**	**51**	**71**	**–7**	**–58**

Royalties are paid in the period after they are accrued
2044+ covers the period from 2044–2053

22.2 Economic Assumptions

The financial model was prepared using the Reserve Case production schedule, operating, and capital assumptions on an annual basis. The assumptions for taxes and royalties were provided by SSR.

22.2.1 Metal Prices

Metal prices for the economic analysis were estimated after analysis of consensus industry metal price forecasts and compared to those used in other published studies. The metal prices used for the economic analysis, shown in Table 22.8, are considered to be representative of industry forecasts.

Table 22.8 CDMP21TR Economic Analysis Metal Price Assumptions

Metal Price	Units	2022	2023	2024	2025	Long-Term
Gold	$/oz	1,800	1,740	1,710	1,670	1,600
Silver	$/oz	24.00	23.00	22.00	21.00	21.00
Copper	$/lb	4.00	3.80	3.80	3.80	3.40

22.2.2 Taxation

The Turkish government implemented a temporary rate increase from 20% to 22% for the periods of 2018-2020. From 2022 onwards, the effective tax rate is expected to return to 20%.

For tax purposes, a 20% accelerated depreciation rate is applicable for both the oxide and sulfide capital. The depreciation period is 10 years for general mining equipment, if not specifically defined by the tax office.

Investment incentive certificates (IIC) are available for investments that promote economic development. IIC's can be classified as strategic in specific circumstances, thereby providing additional incentives. Anagold received a strategic IIC for the sulfide process plant. An IIC generates credits that offset corporate income taxes generated by the investment. The amount of investment credits generated from the IIC is based on eligible capital expenditures. These investment credits reduce the corporate tax rate to a minimum of 2% in a given tax period until the last quarter of 2023, thereafter it is assumed subsequent non-strategic IIC's will be available and the minimum rate will be 4%. Incentive tax credits can be carried forward to future tax periods indefinitely until exhaustion. Incentive tax credits and other tax pools are determined in the local currency, Turkish Lira, and subject to devaluation and revaluation as fluctuations against the US dollar occur. The cash flow model is prepared on a constant Turkish Lira basis.

Value-added tax (VAT) in Turkey is levied at 18% and the project is eligible for the Turkish exemptions for mining projects and mining equipment purchases. In the CDMP21TR assumes the cash flows are not subject to VAT.

Import duties are not included in the capital cost estimate for mining related imported equipment because they are exempted in the IIC's.

22.2.3 Royalties

Under Turkish Mining Law, the royalty rate for precious metals is variable and tied to metal prices. The Çöpler project is subject to a mineral production royalty which is based on a sliding scale to gold price and is payable to the Turkish government.

Table 22.9 details the current prescribed royalty rates applicable to heap leach production (revised September 2020). The royalties are calculated on total revenue with deductions allowed for processing and haulage costs of ore. As the Çöpler project produces silver and copper as by-products of the process of treating gold ore, revenue from by-products is included in the total revenue used for royalty calculations. Royalty rates are reduced by 40% for ore processed in country, as an incentive to process ore locally. The Çöpler project POX production is eligible for a 40% reduction to the royalty rate.

Table 22.9 Gold Royalty Rates

Metal Price ($/oz Gold)		Prescribed Royalty Rate (%)	Royalty After 40% In-Country Processing Incentive (%)
From	To		
0	800	1.25	0.50
800	900	2.50	1.00
900	1,000	3.75	1.50
1,000	1,100	5.00	2.00
1,100	1,200	6.25	2.50
1,200	1,300	7.50	3.00
1,300	1,400	8.75	3.50
1,400	1,500	10.00	4.00
1,500	1,600	11.25	4.50
1,600	1,700	12.50	5.00
1,700	1,800	13.75	5.50
1,800	1,900	15.00	6.00
1,900	2,000	16.25	6.50
2,000	2,100	17.50	7.00
2,100	+	18.75	7.50

The Çöpler project effective life-of-mine (LOM) royalty rate based on the financial model metal price assumptions and applicable deductions is approximately 4.9%.

Other than the royalty payments, there are no other known back-in rights, payments, or other agreements and encumbrances to which the property is subject.

23 ADJACENT PROPERTIES

There are no adjacent properties that are applicable to the CDMP21TR.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Çöpler Resource Initial Assessment

The Initial Assessment Case is a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

To support the 2021 Çöpler Mineral Resource estimate an Initial Assessment has been prepared to analyse the impact of changes in processing method for the Çöpler Mineral Resource. The project currently has two processing methods:

- Sulfide process plant
- Heap leach oxide processing facility

The sulfide plant includes the crushing, grinding, flotation and pressure oxidation to produce gold and small amounts of silver The heap leach facility produces gold and small quantities of silver and copper.

The scenario for the Initial Assessment Case analysis includes additional processing options to recover copper from the sulfide Mineral Resource. The two processing options are:

- Copper Concentrator producing a copper concentrate and a pyrite concentrate.
- Sodium Hydrosulfide (NaSH) copper recovery circuit to be installed in the current sulfide plant.

The copper concentrator would make a copper concentrate for sale to smelters and a pyrite concentrate to be fed into the autoclaves in the sulfide plant. The pyrite concentrate would have a high gold content and provide sulfur as a source of fuel for the autoclaves. The copper concentrator capacity is 1.8 Mtpa.

The Çöpler Mineral Resource has been selected for the Initial Assessment analysis because the other Mineral Resources at the project do not have significant amounts of copper.

Implementation of the copper recovery options will require capital expenditures and will also provide opportunities for operational cost and productivity improvements. The Initial Assessment Case shows the results of a shorter term analysis using the Reserve Case metal prices and the impact of the estimated capital and potential cost savings from economies of scale and reallocation of shared and fixed costs.

For the Initial Assessment economic analysis the Ardich and Çakmaktepe Mineral Reserves have been included in the cash flow analysis without change from the Reserve Case. This is to allow the analysis to quantify the impact of the copper concentrator and NaSH circuit and demonstrate the potential of the additional Mineral Resources.

The Initial Assessment Case production is oxide of 41.8 Mt at 1.26 g/t Au, 59.7 Mt at 2.45 g/t Au of sulfide, and an additional 24.9 Mt at 0.50 g/t Au and 0.2% Cu amenable to concentrator treatment for a total of 126.4 Mt at 1.67 g/t Au. The gold production in the Initial Assessment Case is 5.4 Moz and 164 Mlb of copper. Copper is produced from all three processing streams. The impact of including the copper concentrator as a processing facility is to expand the Çöpler pit, which ceases mining in 2043. Additional production in the Initial Assessment Case comes from feed of 1.8 Mtpa to the copper concentrator and also from additional sulfide and oxide processing feed that is exposed when the pit gets deeper. Total capital including contingency of 25% for the copper concentrator and the copper recovery circuit in the sulfide plant is $218M. The capital costs have an accuracy of ±50%.

The Initial Assessment Case results include:

- After-tax NPV at a 5% discount rate of $2.00 billion.
- Mine life of 22 years.

The initial Assessment Case shows an average AISC of $924/oz gold.

Key results of the Initial Assessment Case economic analysis are shown in Table 24.1. The after tax annual and cumulative cash flow is shown in Figure 24.1 and the before and after tax NPV at a range of discount rates is shown in Table 24.2.

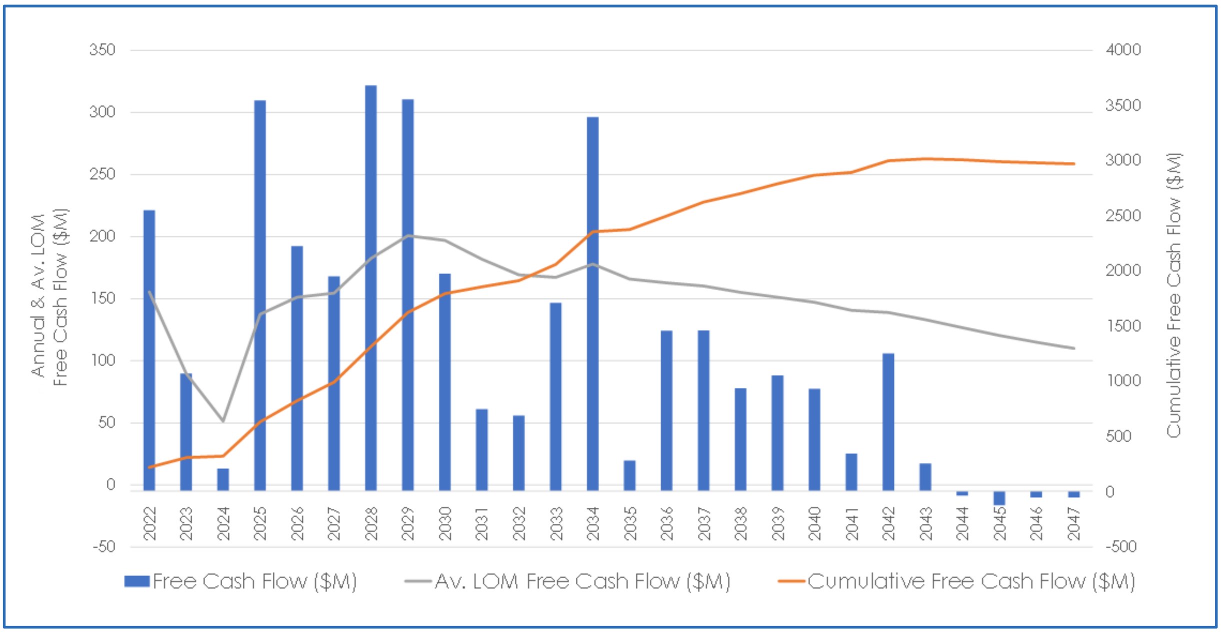
The Initial Assessment has been prepared to demonstrate economic potential of the Mineral Resources at the Çöpler Deposit. The Initial Assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have modifying factors applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that this economic assessment will be realised.

Table 24.1 Çöpler Initial Assessment Case Results Summary

Item	Unit	Initial Assessment Case
Oxide Processed		
Heap Leach Quantity	kt	41,792
Au Feed Grade	g/t	1.26
Sulfide Processed		
Quantity Milled	kt	59,654
Au Feed Grade	g/t	2.45
Cu Concentrator Processed		
Quantity Milled	kt	24,939
Au Feed Grade	g/t	0.50
Cu Feed Grade	%	0.20
Total Gold Produced		
Oxide – Gold	koz	1,068
Sulfide – Gold	koz	4,078
Cu Concentrator – Gold	koz	222
Total – Gold	koz	5,368
Total Copper Production	Mlb	164
5-Year Annual Average		
Average Gold Produced	kozpa	300
Free Cash Flow	$Mpa	165
Total Cash Costs (CC)	$/oz gold	761
All-in Sustaining Costs (AISC)	$/oz gold	938
Key Financial Results		
LOM Total Cash Costs (CC)	$/oz gold	783
LOM All-in Sustaining Costs (AISC)	$/oz gold	924
Site Operating Costs	$/t treated	43.79
After-Tax NPV5%	$M	2,004
Mine Life	years	22

5-Year annual average is for the period 1 January 2022 through 31 December 2026

Figure 24.1 Çöpler Initial Assessment Case After-Tax Cash Flow



OreWin, 2022

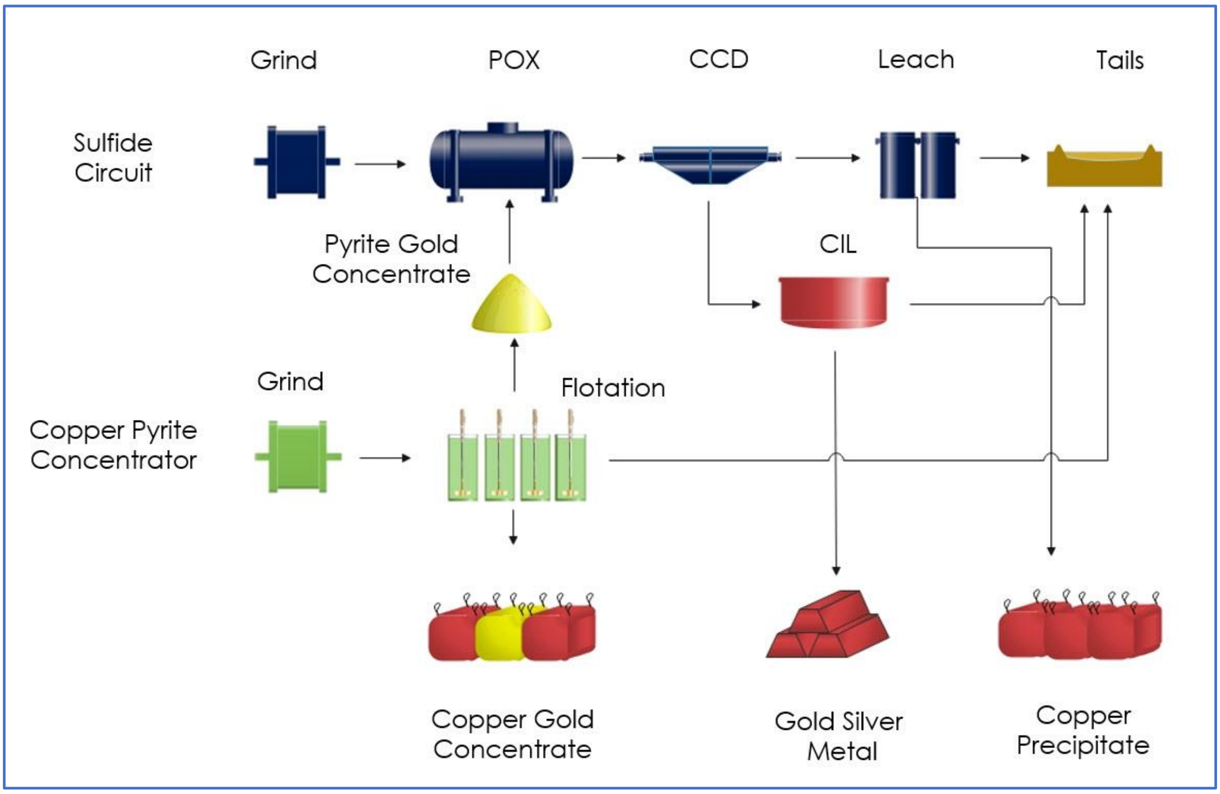
Table 24.2 Çöpler Initial Assessment Case Before and After-Tax NPV

Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	3,301	2,958
5%	2,194	2,004
10%	1,571	1,457
12%	1,398	1,304

24.2 Copper Recovery Processing Assumptions

A simplified process flow diagram after the addition of the copper concentrator and NaSH circuit is shown in Figure 24.2.

Figure 24.2 Çöpler Project Initial Assessment Simplified Process Flow Diagram



OreWin, 2022

24.2.1 Copper Concentrator

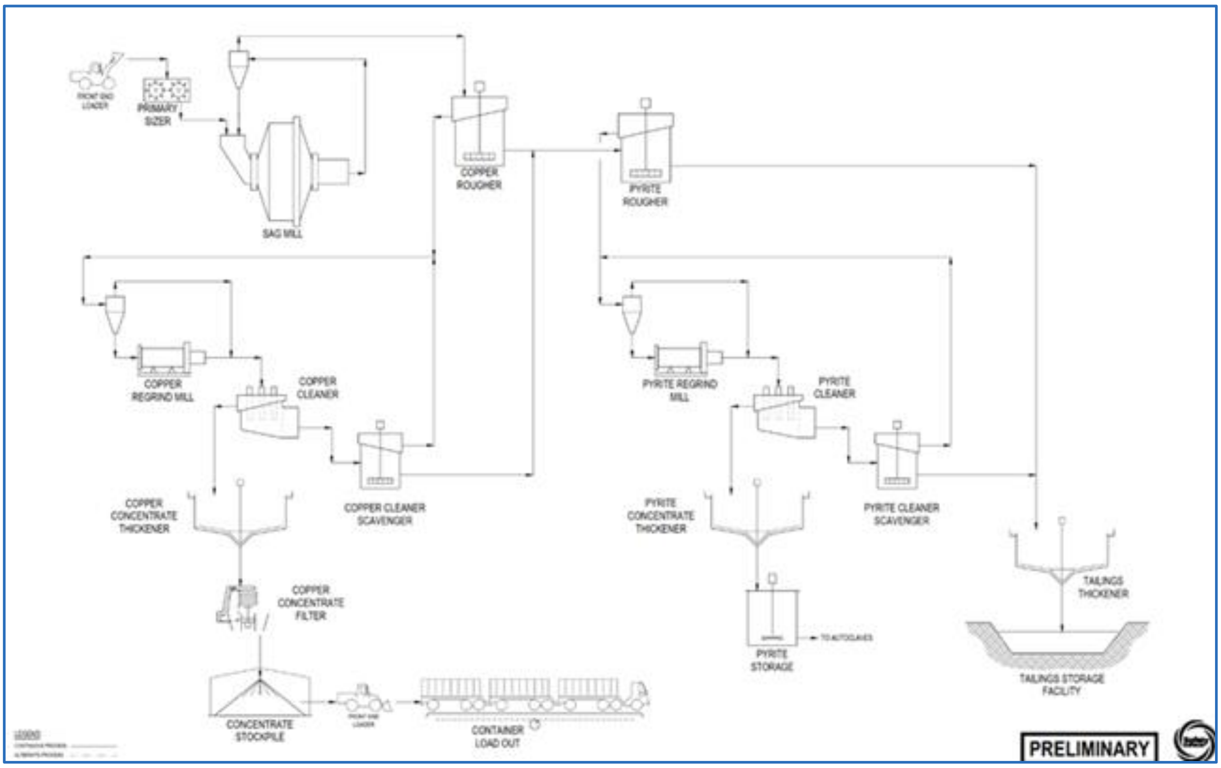
A preliminary design was developed for a concentrator to process copper and pyrite bearing ore at the Çöpler mine site. The design throughput rate was assumed to be 1.8 Mtpa of ROM ore producing saleable copper concentrate with gold and silver and a sulfur-rich pyrite concentrate containing gold for fuel in the autoclave.

The copper concentrator plant includes a ROM pad and a comminution circuit consisting of a sizer and single stage SAG mill. The concentrator consists of a copper flotation circuit inclusive of roughers, copper concentrate regrind, and cleaner flotation to produce a copper concentrate. The copper concentrate is thickened, filtered, and concentrate loadout into bulk bags for shipment to a smelter. Figure 24.3 shows the copper concentrator process flow diagram.

The pyrite flotation circuit treats the copper flotation tails and consists of roughers, a concentrate regrind mill, and cleaner flotation to produce a pyrite concentrate. The pyrite concentrate is thickened and stored in agitated tanks and pumped to the existing gold POX circuit as required to supplement autoclave sulfur requirements.

Final Tailings are thickened and pumped to the TSF. The circuit is inclusive of associated plant services including reagents, air, and water.

Figure 24.3 Çöpler Preliminary Copper Concentrator Process Flow Diagram



Ausenco, 2021

24.2.2 NaSH Copper Recovery Circuit

When originally constructed, the POX circuit did not include extraction of copper. A new circuit assuming precipitation from the pregnant leach solution by sodium hydrosulfide (NaSH) sulfidisation would be expected to achieve high copper recoveries.

Allowance was made in the existing plant for space to install a copper recovery circuit in the future.

24.2.3 NSR Inputs and Cut-off Grade Calculations

The cut-off grades for the Mineral Reserves are presented as a gold only cut-off grade because the majority of the model cell value is derived from gold. For the analysis of the copper recovery, the copper proportion of the plant feeds have a significant revenue, therefore it is necessary to determine the cell values from the three revenue elements: gold, copper, and silver. For this reason, a NSR was calculated for each cell in the Mineral Reserve model for the Initial Assessment for the sulfide and copper concentrator scenario.

Three processing material types were defined for the Initial Assessment. These three processing material types are shown in Table 24.3.

Table 24.3 CDMP21TR Initial Assessment Case Process Types

Ore Type	Defining Criteria
Oxide	S% < 2
Sulfide	S% >= 2
Copper Concentrator	Cu% >= 0.1

24.2.3.1 Initial Assessment Processing Recoveries

The processing parameters used for the calculation of the NSR and cut-off grades are shown in Table 24.4 to Table 24.7.

Table 24.4 Heap Leach Recoveries

Item	Units	Amount
Au Recovery	%	62.3%–78.4%

Table 24.5 Copper Concentrate Recoveries

Item	Units	Amount
Au Recovery	%	55
Ag Recovery	%	45
Cu Recovery	%	84
Mass Pull = (2 x Cu + 0.15) / 100		

Table 24.6 Pyrite Concentrate Recoveries

Item	Units	Amount
Au Recovery	%	15
Ag Recovery	%	15
Cu Recovery	%	2
Mass Pull = (SS + 0.4) / 100		

Table 24.7 POX Plant Recoveries

Item	Units	Amount
Au Recovery	%	91
Ag Recovery	%	10
Cu Recovery	%	98

24.2.3.2 Costs

Costs have been estimated using actual costs from the project, review of plans for productivity and cost savings, previous capital estimates for a NaSH circuit, and the copper concentrator scoping study.

The accuracy of the estimates are within ±50%. A 25% contingency has been added to the direct capital costs of the copper concentrator and NaSH cost estimates.

Operating and capital costs are shown in Table 24.8 to Table 24.10.

Table 24.8 Operating Costs

Item	Units	Cost
Heap Leach Oxide	$/t oxide $Mpa	9.26 8.32
Sulfide Plant	$/t processed	34.67
Copper Concentrator	$/t processed	7.29
Site Support	$Mpa	26.78 to 19.31 8.55 after mining is completed
G&A	$Mpa	5.00
Mining Costs	$/t mined	1.49 to 3.53

Table 24.9 Copper Concentrator and NaSH Capital Cost

Item	Factor	Cost $M
Cu Concentrator Sulfide Cu Recovery Circuit	– –	100.2 33.1
Direct Costs	–	133.3
EPCM Owner's Costs Contingency	18% 20% 25%	24.6 26.7 33.3
Total Capital	–	217.9

Table 24.10 Other Capital Costs

Item	Units	Amount
Closure	$M	114
Heap Leach Sustaining	$/t	0.15

24.2.3.3 Metal Prices and Selling Costs

Metal prices for the economic analysis were estimated after analysis of consensus industry metal price forecasts and compared to those used in other published studies. The metal prices used for the economic analysis, shown in Table 24.11, are considered to be representative of industry forecasts.

Reserve cut-off metal prices are lower than the long-term forecasts and represent a conservative view of the long-term gold price. Metal prices used in cut-off's for Mineral Resources were selected to be higher than the long-term consensus prices and are in line with other price estimates used for Mineral Resources.

Realisation assumptions are shown in Table 24.12 and Table 24.13.

Table 24.11 CDMP21TR Economic Analysis Metal Prices Assumptions

Metal	Units	2022	2023	2024	2025	Long- Term
Gold Price	$/oz	1,800	1,740	1,710	1,670	1,600
Silver Price	$/oz	24.00	23.00	22.00	21.00	21.00
Copper Price	$/lb	4.00	3.80	3.80	3.80	3.40

Table 24.12 Transport and Treatment Charges

Item	Units	Amount
Concentrate Moisture	%	12
Concentrate Transport	$/t wet	25
Concentrate Treatment	$/t concentrate	80

Table 24.13 Payable Metal Assumptions

Item	Units	Amount
Payable Au	%	97.5
Payable Ag	%	90.0
Payable Cu	%	96.0

24.2.3.4 Çöpler Cut-off grades

Cut-off grades in the Initial Assessment for oxide used gold cut-offs of 0.47–0.59 g/t Au.

In the sulfide Mineral Resource an NSR was calculated using the parameters discussed above.

For the copper concentrator, the cut-off applies to Mineral Resource with a Cu > 0.10%. The cut-off used was as follows:

$7.68/t NSR + (Pyrite Mass Pull) x $34.88/t NSR

For the remaining sulfide Mineral Resource, the cut-off used was $34.88/t NSR.

24.3 Initial Assessment Case Mining

Mining in the Initial Assessment is planned to be the same as the current operation. A plan and section showing the Initial Assessment pit shell and the Reserve Case pit design are shown in Figure 24.4 and Figure 24.5

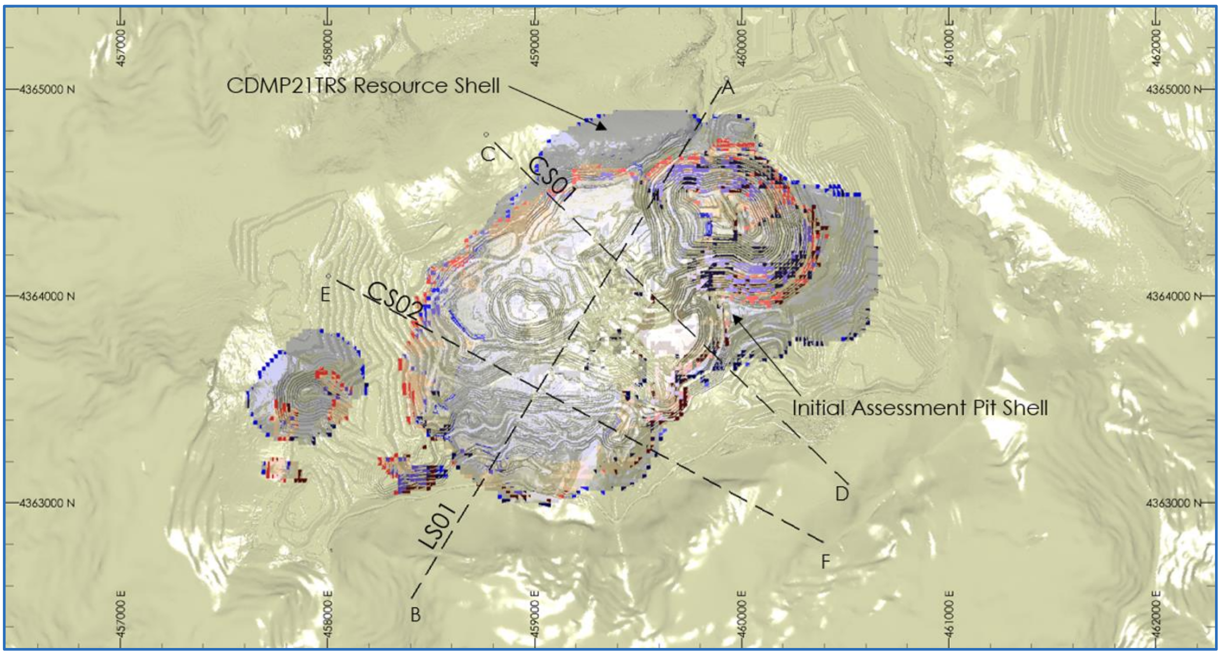
Pit optimisation was prepared using the assumptions described above to generate pit shells. Mineral Resources classified as Measured, Indicated, and Inferred were used in the optimisation. Previous work on the Çöpler pits prepared by OreWin produced pit shells that were used for designs on the Çöpler pit. The pit optimisation work generated pit shells that were considered a close match to the Reserve Case designs. Based on this experience it was considered reasonable to use the pit shells from the optimisation work for production analysis in the Initial Assessment.

24.4 Initial Assessment Case Production Schedule and Cash Flow

The annual mining production and process production for oxide heap leach, sulfide, and copper concentrator are shown in Table 24.14 and Table 24.15.

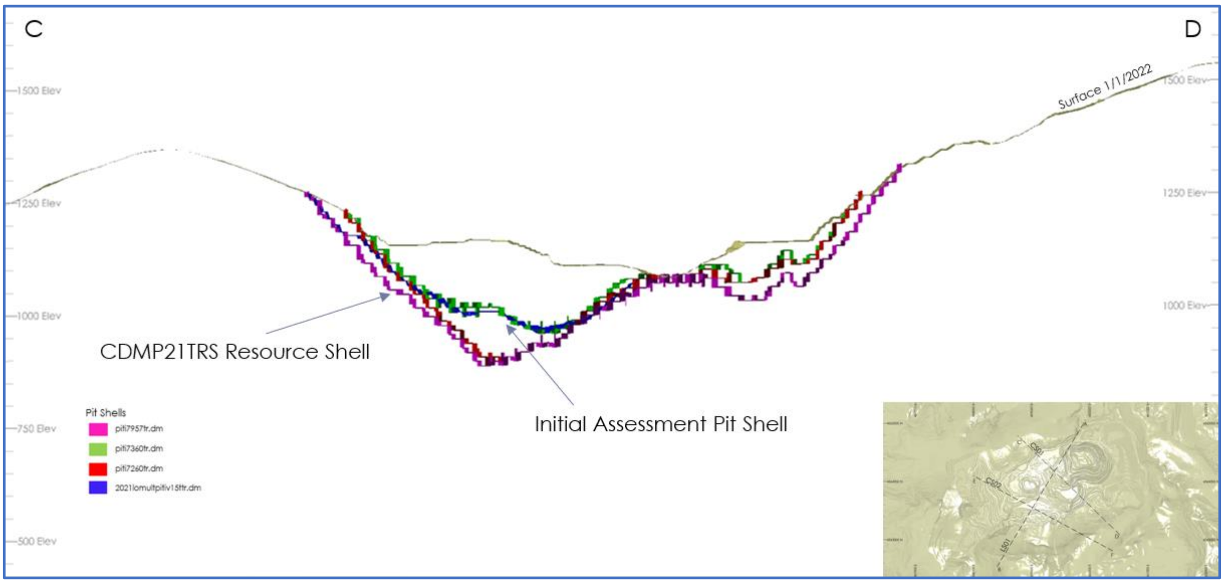
The Initial Assessment Case annual cash flow is shown in Table 24.16.

Figure 24.4 Çöpler Plan Initial Assessment Pit Shell, Resource Shell, and Reserve Pit Design



OreWin, 2022

Figure 24.5 Çöpler Long-section Initial Assessment Pit Shell, Resource Shell, and Reserve Pit Design



OreWin, 2022

Table 24.14 Çöpler Initial Assessment Case Production Schedule

Description	Units	TOTAL	Year																					
			2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
Total Movement	kt	**721,847**	26,798	36,490	52,473	50,053	53,177	51,541	45,392	45,590	46,872	46,719	46,893	41,577	25,116	20,655	22,616	23,239	22,159	21,630	20,758	18,741	2,829	529
Waste	kt	**595,461**	23,828	32,015	44,655	43,362	44,843	45,329	37,456	39,722	39,649	41,003	40,115	33,191	18,062	16,033	16,033	16,033	16,033	16,033	16,033	16,033	–	–
Plant Feed	kt	**126,386**	2,970	4,475	7,818	6,691	8,334	6,212	7,936	5,869	7,223	5,716	6,778	8,386	7,053	4,621	6,583	7,206	6,126	5,597	4,725	2,708	2,829	529
Heap Leach Stacked	kt	**41,792**	**263**	**2,080**	**5,167**	**2,637**	**3,823**	**1,825**	**2,923**	**963**	**2,324**	**1,241**	**2,318**	**3,476**	**1,585**	**325**	**1,955**	**2,562**	**1,920**	**2,639**	**1,766**	**–**	**–**	**–**
Au Feed Grade	g/t	**1.26**	0.88	1.22	1.12	1.67	1.47	1.22	1.99	2.21	1.38	0.66	1.21	1.62	1.88	0.76	0.85	0.77	0.81	0.67	0.72	–	–	–
Ag Feed Grade	g/t	**1.63**	4.90	8.28	0.59	1.11	1.02	0.77	0.47	2.05	1.36	1.46	0.92	0.59	1.22	5.48	2.44	2.38	2.11	1.55	1.65	–	–	–
Cu Feed Grade	%	**0.09**	0.10	0.07	0.04	0.02	0.04	0.07	0.01	0.03	0.09	0.17	0.11	0.07	0.04	0.08	0.16	0.14	0.17	0.18	0.17	–	–	–
Gold Recovered	koz	**1,068**	20	46	98	94	112	55	108	43	64	21	54	96	61	10	36	43	36	40	30	2	0.5	–
Silver Recovered	koz	**607**	11	133	7	31	38	15	14	19	30	18	21	20	19	18	45	58	41	40	29	–	–	–
Copper Recovered	klb	**9,875**	24	87	12	218	460	379	121	85	596	640	777	786	254	96	916	1,048	996	1,418	963	–	–	–
Sulfide Plant Feed	kt	**59,654**	**2,708**	**2,395**	**2,650**	**2,704**	**2,711**	**2,587**	**3,212**	**3,106**	**3,099**	**2,675**	**2,660**	**3,110**	**3,668**	**2,497**	**2,828**	**2,844**	**2,406**	**2,769**	**2,959**	**2,708**	**2,829**	**529**
Au Feed Grade	g/t	**2.45**	3.16	2.69	2.87	3.05	2.55	3.08	2.85	3.54	2.53	2.10	1.80	2.04	3.12	2.07	2.26	2.30	2.08	2.11	1.83	1.51	1.77	1.77
Ag Feed Grade	g/t	**4.08**	3.82	4.71	1.96	3.59	3.70	5.45	4.89	7.57	5.94	3.74	5.89	4.18	3.66	2.95	2.11	1.48	4.12	6.24	3.81	3.88	1.90	1.92
Cu Feed Grade	%	**0.13**	0.08	0.07	0.09	0.07	0.13	0.16	0.14	0.15	0.18	0.23	0.20	0.18	0.13	0.14	0.12	0.11	0.17	0.17	0.10	0.07	0.10	0.11
Gold Recovered	koz	**4,078**	248	192	222	240	187	211	244	301	215	153	128	172	327	140	175	180	134	159	157	119	146	27
Silver Recovered	koz	**645**	10	11	5	31	30	41	43	65	54	27	46	38	38	19	13	8	26	50	36	34	17	3
Copper Recovered	klb	**71,961**	–	–	–	4,083	2,565	2,706	3,747	4,230	6,159	6,567	4,811	6,115	4,143	1,339	918	623	2,617	3,667	5,910	4,187	6,370	1,204
Cu Concentrator	kt	**24,939**	**–**	**–**	**–**	**1,350**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**1,800**	**189**	**–**	**–**	**–**	**–**
Au Feed Grade	g/t	**0.50**	–	–	–	0.69	0.77	0.74	0.71	0.45	0.37	0.43	0.43	0.44	0.40	0.41	0.41	0.41	0.41	0.41	–	–	–	–
Ag Feed Grade	g/t	**2.12**	–	–	–	1.11	1.59	2.84	4.25	1.91	1.93	1.52	1.56	1.90	1.97	2.20	2.20	2.20	2.20	2.20	–	–	–	–
Cu Feed Grade	%	**0.28**	–	–	–	0.39	0.46	0.44	0.40	0.25	0.20	0.24	0.24	0.24	0.20	0.21	0.21	0.21	0.21	0.21	–	–	–	–
Gold Recovered	koz	**222**	–	–	–	17	24	23	23	14	12	14	14	14	13	13	13	13	13	1	–	–	–	–
Silver Recovered	koz	**135**	–	–	–	8	9	9	9	9	11	10	9	9	10	10	10	10	10	1	–	–	–	–
Copper Recovered	klb	**92,339**	–	–	–	5,711	6,208	6,351	6,362	6,418	7,353	6,811	6,175	6,342	6,819	6,769	6,769	6,769	6,769	711	–	–	–	–
Total Feed	kt	**126,386**	**2,970**	**4,475**	**7,818**	**6,691**	**8,334**	**6,212**	**7,936**	**5,869**	**7,223**	**5,716**	**6,778**	**8,386**	**7,053**	**4,621**	**6,583**	**7,206**	**6,126**	**5,597**	**4,725**	**2,708**	**2,829**	**529**
Au Feed Grade	g/t	**1.67**	2.96	2.01	1.71	2.03	1.67	1.85	2.05	2.37	1.62	1.26	1.23	1.52	2.15	1.33	1.34	1.29	1.19	1.37	1.42	1.51	1.77	1.77
Ag Feed Grade	g/t	**2.88**	3.91	6.37	1.06	2.11	2.02	3.32	3.12	4.93	3.47	2.55	3.04	2.20	2.68	2.84	2.23	1.98	2.92	3.89	3.01	3.88	1.90	1.92
Cu Feed Grade	%	**0.15**	0.08	0.07	0.06	0.12	0.16	0.21	0.15	0.16	0.16	0.22	0.18	0.15	0.13	0.17	0.16	0.15	0.18	0.18	0.13	0.07	0.10	0.11
Gold Recovered	koz	**5,368**	268	238	320	351	323	289	375	358	291	188	195	283	401	163	225	236	183	200	187	121	147	27
Silver Recovered	koz	**1,387**	21	144	12	70	77	66	66	93	95	55	76	67	67	46	68	76	77	91	65	34	17	3
Copper Recovered	klb	**174,175**	24	87	12	10,013	9,233	9,436	10,230	10,733	14,108	14,018	11,763	13,242	11,216	8,204	8,604	8,439	10,382	5,796	6,872	4,187	6,370	1,204

Table 24.15 Çöpler Initial Assessment Case Copper Concentrator Processing Schedule

Description	Units	TOTAL	Year																				
			2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
Cu Concentrator	kt	**24,939**	–	–	–	1,350	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	189	–	–	–	–
Au Feed Grade	g/t	**0.50**	–	–	–	0.69	0.77	0.74	0.71	0.45	0.37	0.43	0.43	0.44	0.40	0.41	0.41	0.41	0.41	0.41	–	–	–	–
Ag Feed Grade	g/t	**2.12**	–	–	–	1.11	1.59	2.84	4.25	1.91	1.93	1.52	1.56	1.90	1.97	2.20	2.20	2.20	2.20	2.20	–	–	–	–
Cu Feed Grade	%	**0.20**	–	–	–	0.23	0.19	0.19	0.19	0.19	0.22	0.20	0.19	0.19	0.20	0.20	0.20	0.20	0.20	0.20	–	–	–	–
Cu Concentrate	kt	**137**	–	–	–	8	9	10	10	10	11	10	9	10	10	10	10	10	10	1	–	–	–	–
Au Feed Grade	g/t	**50.30**	–	–	–	62.97	80.92	76.39	73.76	46.65	34.51	42.52	45.93	45.73	39.11	40.89	40.89	40.89	40.89	40.89	–	–	–	–
Ag Feed Grade	g/t	**173.26**	–	–	–	81.98	137.10	240.96	359.75	160.96	147.06	122.35	134.96	161.46	158.22	177.84	177.84	177.84	177.84	177.84	–	–	–	–
Cu Feed Grade	%	**29.50**	–	–	–	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	29.50	–	–	–	–
Pyrite Concentrate	kt	**169**	–	–	–	11	15	15	14	12	11	11	12	12	11	11	11	11	11	1	–	–	–	–
Au Feed Grade	g/t	**11.15**	–	–	–	13.13	13.46	13.16	13.38	10.46	9.21	10.16	10.20	10.29	9.93	10.15	10.15	10.15	10.15	10.15	–	–	–	–
Ag Feed Grade	g/t	**46.94**	–	–	–	20.90	27.88	50.73	79.76	44.11	47.95	35.73	36.64	44.43	49.08	53.99	53.99	53.99	53.99	53.99	–	–	–	–
Cu Feed Grade	%	**0.68**	–	–	–	0.66	0.50	0.52	0.55	0.68	0.84	0.74	0.67	0.68	0.78	0.76	0.76	0.76	0.76	0.76	–	–	–	–

Table 24.16 Initial Assessment Case Cash Flow

Cash Flow Statement ($M)	TOTAL	Year																						
		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044+
Heap Leach - Gold Revenue	1,737	36	80	167	157	180	87	173	68	102	34	86	154	98	16	58	69	57	65	47	3	1	–	–
Sulfide Plant - Gold Revenue	6,999	447	334	379	429	338	375	427	505	363	267	226	298	543	244	302	309	236	256	252	191	234	44	–
By-Product Revenue	637	5	5	5	40	33	33	36	38	50	49	42	46	40	29	31	30	37	22	25	15	22	4	–
Net Revenue	**9,373**	**488**	**419**	**552**	**625**	**550**	**496**	**637**	**612**	**515**	**350**	**354**	**499**	**680**	**289**	**390**	**408**	**329**	**342**	**324**	**209**	**257**	**48**	**–**
Realisation Costs																								
Freight & Refining	68	1	1	1	4	5	5	5	4	4	4	4	4	5	3	4	4	4	2	1	0	1	0	–
Royalties	486	23	27	19	29	36	30	25	36	33	26	14	15	27	39	11	18	19	15	14	13	7	10	1
Total - Realisation Costs	**554**	**24**	**28**	**21**	**34**	**41**	**35**	**30**	**41**	**38**	**30**	**18**	**20**	**32**	**42**	**15**	**22**	**23**	**16**	**16**	**13**	**7**	**10**	**1**
Operating Costs																								
Mining	1,255	47	57	86	74	78	77	70	72	76	77	79	73	48	42	48	51	51	51	50	47	–	–	–
Processing - Heap Leach	550	13	21	64	33	44	25	35	17	30	20	30	41	23	12	27	32	26	33	25	–	–	–	–
Processing - Sulfide Plant	2,084	115	83	91	94	94	89	111	107	107	92	92	107	127	86	98	98	83	96	103	94	98	18	–
Processing - Cu Concentrator	182	–	–	–	10	13	13	13	13	13	13	13	13	13	13	13	13	13	1	–	–	–	–	–
Site Support	453	32	30	29	27	27	27	27	23	23	19	19	19	19	19	19	19	19	19	19	9	9	–	–
G&A	100	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	0	–	–
Total - Operating Costs	**4,624**	**212**	**196**	**275**	**242**	**261**	**237**	**262**	**237**	**253**	**227**	**238**	**259**	**236**	**178**	**211**	**219**	**197**	**205**	**202**	**154**	**107**	**18**	**–**
Operating Surplus / (Deficit)	**4,194**	**252**	**196**	**256**	**350**	**249**	**224**	**345**	**334**	**224**	**93**	**98**	**221**	**413**	**68**	**165**	**167**	**109**	**121**	**106**	**42**	**143**	**20**	**-1**
Capital Costs																								
Growth	357	4	69	192	22	–	–	–	–	–	–	18	18	18	18	–	–	–	–	–	–	–	–	–
Sustaining	458	32	55	52	12	49	49	12	12	30	30	12	12	12	12	12	12	11	11	10	10	10	–	–
Closure	114	–	–	–	–	–	–	–	–	–	–	7	13	13	13	–	2	3	1	1	1	1	1	58
Working & Other	-37	-12	-21	-4	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total - Capital Costs	**893**	**25**	**103**	**239**	**34**	**49**	**49**	**12**	**12**	**30**	**30**	**36**	**42**	**42**	**42**	**12**	**14**	**14**	**12**	**11**	**11**	**11**	**1**	**58**
Net Cash Flow Before Tax	**3,301**	**227**	**93**	**17**	**315**	**200**	**175**	**333**	**322**	**195**	**63**	**62**	**178**	**371**	**26**	**153**	**153**	**95**	**108**	**95**	**30**	**131**	**18**	**–60**
Tax	344	6	3	4	6	8	7	12	11	25	2	6	31	74	6	28	29	17	20	18	5	26	1	–
Net Cash Flow After Tax	**2,958**	**221**	**90**	**13**	**310**	**192**	**168**	**322**	**311**	**170**	**61**	**56**	**147**	**296**	**20**	**124**	**125**	**78**	**88**	**77**	**25**	**106**	**17**	**-60**

2044+ covers the period from 2044–2052

24.5 Initial Assessment Summary Results

The Initial Assessment Case production is 126.4 Mt at 1.67 g/t Au this includes 24.9 Mt of feed to the copper concentrator at 0.5 g/t Au and 0.20% Cu. The metal production in the Initial Assessment Case is 5.4 Moz of gold and 164 Mlb of copper. The Mineral Resource in the Initial Assessment Case contains 27% Inferred Mineral Resource. The production schedule was prepared so that only Measured and Indicated Mineral Resources are processed in the first four years of the production schedule and there is only 10% Inferred Mineral Resource in the process feed fifth year of production. An additional cash flow analysis has been prepared using only Measured and Indicated Mineral Resources to show the impact of Inferred Mineral Resource on the economic analysis.

The increase in total production relative to the Reserve Case is from expansion of the Çöpler pit deeper than the current Reserve pit design. As well as the copper concentrator feed that is captured in the Initial Assessment pit shell additional oxide and sulfide Mineral Resource is exposed resulting in increased feed for both the heap leach oxide and sulfide processing facilities.

The Initial Assessment Case results include:

- After-tax NPV at a 5% real discount rate is $2.00 billion.
- Mine life of 22 years.

The Initial Assessment Case shows an average AISC of $924/oz gold.

The Reserve Case after-tax NPV at a 5% discount rate is $1.73 billion. Incremental analysis suggest that the impact from the addition of the copper concentrator and NaSH circuit to increase the after-tax NPV5% by $273M demonstrates the economic potential of the Çöpler Mineral Resource.

The Initial Assessment Case is a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

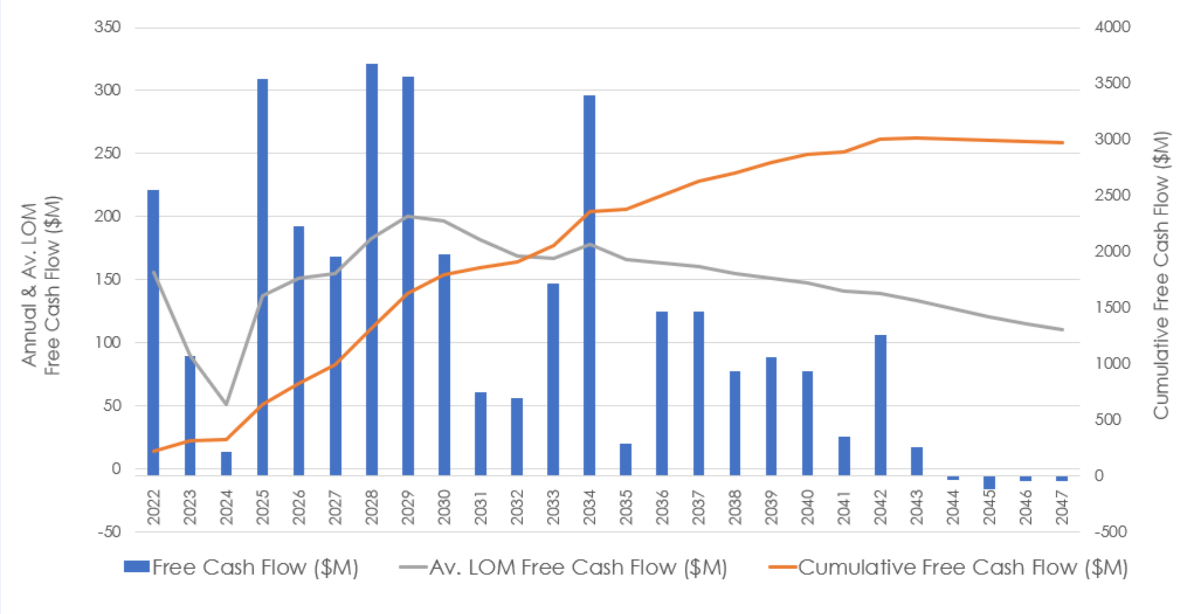
Key results of the Initial Assessment Case economic analysis are shown in Table 24.17. The after-tax cash flow is shown in Figure 24.6. The sulfide and oxide production profiles are shown in Figure 24.7 and gold production is shown in Figure 24.8. The NPV results for before and after-tax over a range of discount rates is shown in Table 24.18.

Table 24.17 Initial Assessment Case Results Summary

Item	Unit	Initial Assessment Case
Oxide Processed		
Heap Leach	kt	41,792
Au Feed Grade	g/t	1.26
Sulfide Processed		
Milled	kt	59,654
Au Feed Grade	g/t	2.45
Cu Concentrator Processed		
Milled	kt	24,939
Au Feed Grade	g/t	0.50
Cu Feed Grade	%	0.20
Total Gold Produced		
Oxide – Gold	koz	1,068
Sulfide – Gold	koz	4,078
Cu Concentrator – Gold	koz	222
Total – Gold	koz	5,368
Total – Copper	Mlb	164
5-Year Annual Average		
Average Gold Produced	kozpa	300
Free Cash Flow	$Mpa	165
Total Cash Costs (CC)	$/oz gold	761
All-In Sustaining Costs (AISC)	$/oz gold	938
Key Financial Results		
LOM Total Cash Costs (CC)	$/oz gold	783
LOM All-In Sustaining Costs (AISC)	$/oz gold	924
Site Operating Costs	$/t processed	43.79
After-Tax NPV5%	$M	2,004
Mine Life	years	22

5-Year annual average is for the period 1 January 2022 through 31 December 2026

Figure 24.6 Initial Assessment Case After-Tax Cash Flow

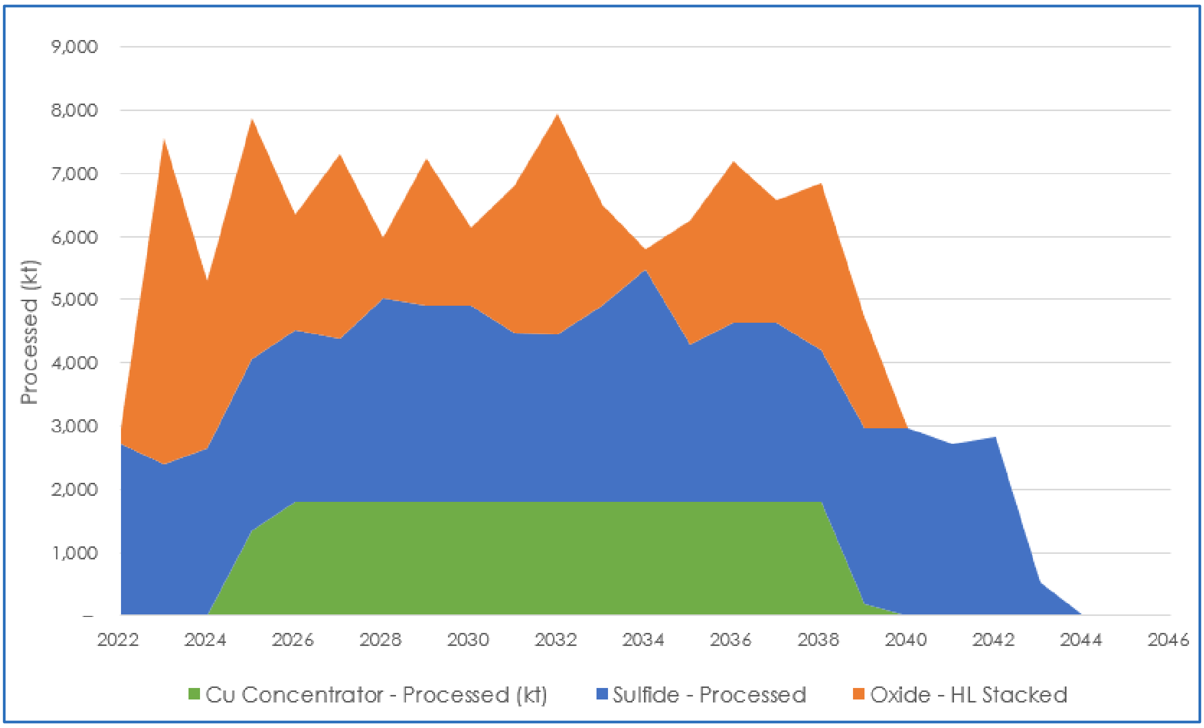


OreWin, 2022

Table 24.18 Initial Assessment Case Before and After-Tax NPV

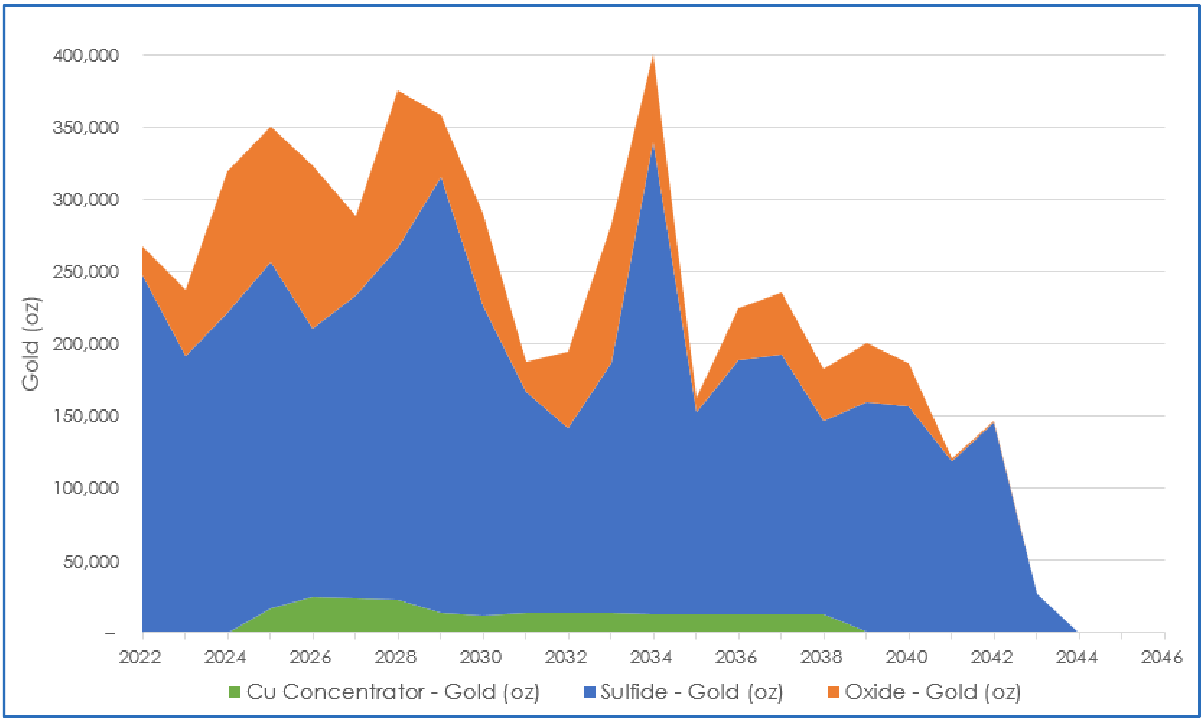
Discount Rate	Before Tax (NPV $M)	After Tax (NPV $M)
Undiscounted	3,301	2,958
5%	2,194	2,004
10%	1,571	1,457
12%	1,398	1,304

Figure 24.7 Initial Assessment Case Processing



OreWin, 2022

Figure 24.8 Initial Assessment Case Gold Production



OreWin, 2022

The Initial Assessment has been prepared to demonstrate economic potential of the Mineral Resources at the Çöpler Deposit. The Initial Assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have modifying factors applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that this economic assessment will be realised.

Costs, metal prices, taxation, and royalty assumptions used in the Initial Assessment Case economic analysis were the same as in the Reserve Case.

The estimates of cash flows have been prepared on a real basis with a base date of Q4'21 and a mid-year discounting is used to calculate NPV.

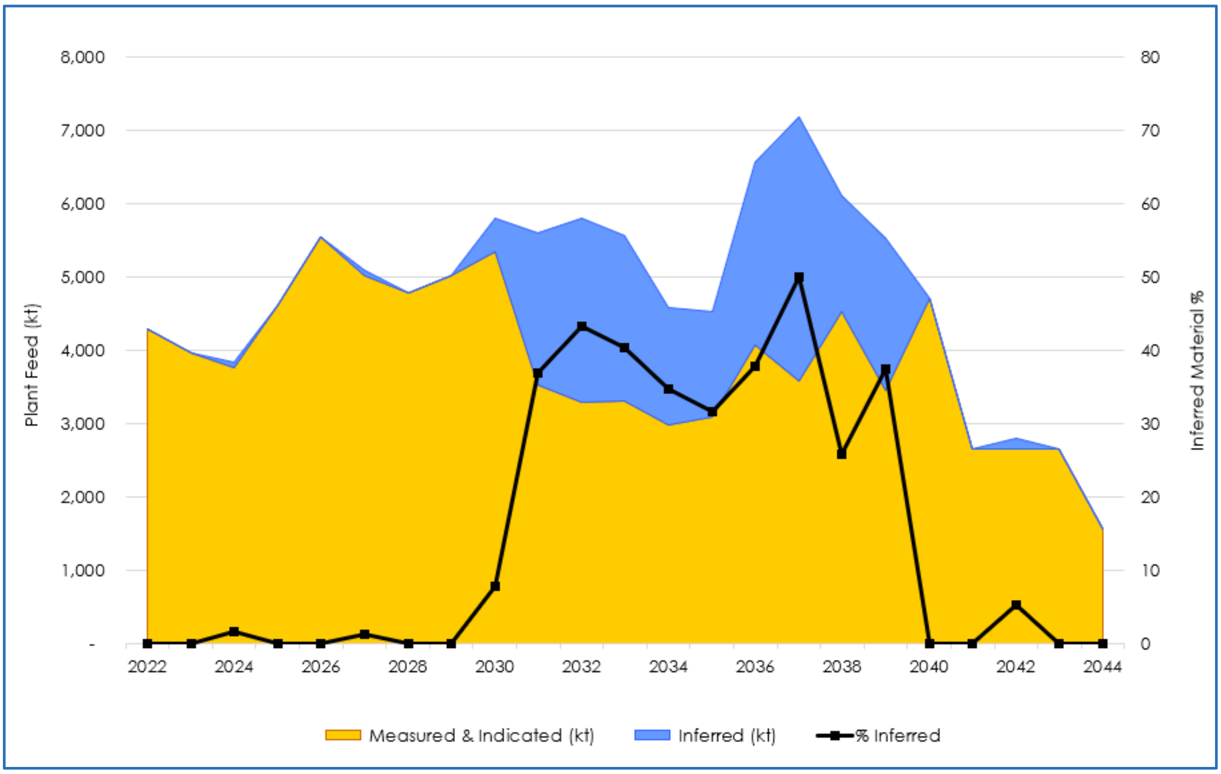
24.5.1 Impact of Inferred Mineral Resources

A separate analysis of the Initial Assessment Case was prepared using only Measured and Indicated Mineral Resources (MI Case). Comparison of the initial years of the Initial Assessment and the MI Case showed only 1.4% of the material processed in the first nine years of the Initial Assessment is Inferred Mineral Resource. Most of the Inferred material is processed in years 10 to 20 and does not exceed 50% of the total processing in any one year. This is shown in Figure 24.9.

The after tax NPV5% for this MI Case only analysis is $1,867M this is a reduction of $137M. To mitigate this additional resource development work and studies will need to done convert the Inferred Mineral Resources to Indicated or Measured Mineral Resources or find an alternative source.

The Initial Assessment has been prepared to demonstrate the economic potential of the Mineral Resources at the Çöpler Deposit. The Initial Assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have modifying factors applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that this economic assessment will be realised.

Figure 24.9 Initial Assessment Case Gold Production



OreWin, 2022

25 INTERPRETATION AND CONCLUSIONS

Mineral Resources and Mineral Reserves in the CDMP21TR meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The CDMP21TR has identified additional Mineral Resources and additional Mineral Reserves when compared to prior studies.

The Initial Assessment Case has demonstrated that there is significant economic potential that may be derived from the copper in the Çöpler Mineral Resource. Given this economic potential it is then concluded that it is valid to report the Mineral Resources using the Mineral Resource metal prices and pit shell.

Further study and analysis will be required to advance the understanding of this potential.

Significant factors that could materially affect the Mineral Resources and Mineral Reserve are:

- Environmental, Permitting Social and Community – the Çöpler project is subject to the laws and regulations of Turkey, the mine has several local communities that are nearby. In order to operate the mine, Anagold must maintain appropriate relations with all the authorities and stakeholders. Social, community and government relations are managed by Anagold and include programmes and engagement with the local communities and both local and national governments. Anagold has remained in compliance with all aspects of the Environmental Impact Assessments (EIA) and operating permits throughout the history of the project.
- Seismic impacts – the Çöpler project is in an area with a history of significant seismic activity that could negatively impact mining operations.
- Metal price impacts – gold is the primary revenue element and silver and copper are produced as by-products. The ore is mined at an elevated cut-off grade and low-grade ore is stockpiled for processing after mining is completed. The use of the elevated cut-off grade serves to mitigate the risks from periods of lower gold prices.
- Geotechnical impacts – slope recommendations have significant impacts on the Mineral Reserve and the continued study will allow the Mineral reserves to be maximised.
- Processing impacts – the processing analysis in the Reserve Case includes incorporation of a flotation circuit into the existing sulfide plant to upgrade sulfide sulfur (SS) to fully utilise grinding and pressure oxidation (POX) autoclave capacity. Continued debottlenecking of the sulfide plant and optimisation of the flotation circuit may improve costs and recoveries, changing cut-off grades and impacting the Mineral Reserve.
- The addition of the flotation circuit to the sulfide plant required new grade control protocols and associated stockpile strategies to be implemented to manage the required sulfide plant feed blend. It is likely that there will need to be ongoing modification of the stockpiling cut-offs and procedures for both short-term and longer term blending as the mine progresses. Measures such as increasing the number of active mining areas, increasing the mining rate, and increasing the size or number of ROM stockpiles.

26 RECOMMENDATIONS

Key recommendations from the CDMP21TR are:

- Continue to update and evaluate the Çöpler District Master Plan as the existing Mineral Resources and Mineral Reserves are updated and as new prospects are advanced.
- Undertake infill drilling at Çöpler and update the copper Mineral Resource estimate.
- Prepare further studies of the copper recovery options.
- Geotechnical review and study of the re-evaluation of the pit re-designs.
- Optimisation of the sulfide flotation circuit, POX, and process operation.
- Metallurgical testwork on future oxide, sulfide, and copper ore sources.
- Optimisation of the oxide heap leach circuit.
- Optimisation of the mining rates to increase gold production.
- Stockpile reconciliation and management studies.
- Review and adapt the ore control and stockpiling strategies to optimise recovery and throughput and maximise gold production.
- Reconcile monthly blend and gold production with predictive modelling.
- Continue drilling at Ardich.
- Geotechnical studies of Ardich.
- Reconciliation studies of Çöpler.
- Update Çöpler and Ardich resource models and estimates.
- Further study of Initial Assessment Case and advance to next stage of study:
  - Geotechnical studies
  - Environmental Impact Assessments (EIA) and permitting
  - Blasting studies
- Metallurgical studies.

26.1 Mineral Resources

Specific recommendations related to the Mineral Resource are:

- Mineral Resource models should be updated on a campaign basis following the completion of planned drilling programmes. Where significant new data has been obtained (either exploration data, or production data), an annual model update roster should be adequate, but only required where warranted by the introduction of new data that has potential to result in a material change in the model (such as by significant modifications to the geological interpretation, or by substantial expansion of the dimensions of the mineralisation).

- The Çöpler model has not been updated since 2016. It is recommended that a new model be developed to incorporate the new exploration data obtained since that time, and to check interpretations relative to grade control data to help hone the interpretation.
- Continue drilling at Ardich.
- An update to the Ardich model is warranted given the quantum of new data that has been obtained since the most-recent update, and the status of the deposit as shown in the Initial Assessment Case.
- Both Çöpler and Ardich are geologically complex deposits with multiple metals that must be tracked along with oxidation type and lithological domains, further complicated by extensive structural disruption. Work on verifying and adjusting resource model domains and parameters should be continued to help facilitate a greater understanding of the deposits, hopefully resulting in improved resource estimates.
- Since the mineralisation locally follows the lithological contacts and structural features, using a search ellipse that follows these trends (dynamic anisotropy) should be evaluated.
- An audit of the databases used to house exploration and grade control data should be undertaken on a reasonably regular basis (e.g., annually). This should include review of all related procedures, monitoring observance to the procedures, and spot checks of the database itself to identify errors and omissions.
- A comprehensive and consistent suite of assays should be collected routinely in exploration drilling. This should be formalised as a requirement across all exploration drilling. Estimation into the resource models should involve all components that may be of future interest.
- The routine collection of in-pit mapping data is encouraged as this information provides invaluable experiential knowledge to inform interpretations based on exploration data.
- Detailed scheduling and design of the sulfide ore stockpiles should be completed. Results from ongoing metallurgical testwork will assist in determining the optimal stockpiling strategy and in reconciliation success.
- Further refinement of the modelled carbonate and sulfide sulfur (SS) grades in the resource model should be completed.
- Further mapping and definition of the local and regional fault structures, alteration types, and other domains should be completed to reduce or realise geotechnical risk in the areas where these structures intersect the pit.

26.2 Mineral Reserves

Specific recommendations related to the Mineral Reserve are:

- Re-design of Çöpler pits at updated metal prices.
- Geotechnical review and study of the re-evaluation of the pit re-designs.
- Optimisation of float circuit pressure oxidation (POX) and process operation including metallurgical testwork on Ardich and Çöpler.

- Review and monitor the stockpiling procedures and criteria to optimise the feed to the plant.
- Optimisation of the mining rates to increase gold production.
- Stockpile reconciliation and management studies.
- Geotechnical studies of Ardich.

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