Master of Science in Mining Engineering

Research Thesis

Strategic mine planning optimization for the potential transition from open-pit to underground mining at Akyem Gold Mine, Ghana

Thijs Goense

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Delft University of Technology, The Netherlands
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RWTH Aachen, Germany

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Supervisor(s):

Dr. Ir. M. W. N. Buxton

Dr. Ir. M. S. Shishvan

Ir. K. Huss

Committee Members:

Dr. Ir. M. W. N. Buxton

Dr. Ir. M. S. Shishvan

Dr. Ir. A.A.M. Dieudonné

Abstract

The transition from open-pit to underground mining operations is a frequently occurring procedure for mines around the world. The Akyem Gold Mine in Ghana is an example, where the transition is to be optimized with regard to the significant reduction in yearly throughput. The current processing plant is commissioned to efficiently process up to roughly 7.7 Mt of ore per year. The planned underground mine is likely to start operations in 2022 during the final stages of the remaining open-pit operations, with an estimated yearly production of 1.6 Mt of ore. This implies that a major plant modification is recommended in order to prevent unnecessary high operating costs, once a full transition may occur. However, opportunities are seen to discard the plant modification and instead to add incremental open-pit material, which is currently considered as mineralized waste.

The suggested option will be considered once the analyses prove that a certain incremental open-pit head grade will generate more value in combination with the underground activities. This analysis is based on financial modelling of the potential options, where a major trade-off is conducted between the plant modification for the planned option, and the incremental open-pit mining, processing and tailings storage for the suggested option. The analysis is tested by changing all different decision parameters to indicate the optimum solution. Prior to this head grade determination, blending optimization of the potential sources of ore is conducted, in order to determine the optimum period where the potential plant modification may occur.

Results show that the open-pit mine will directly supply the processing plant until 2023, whereas its stockpiles will be depleted by the end of 2025. For this reason, the suggested plant modification is suggested to take place at the start of 2026. Subsequent results related to the option evaluations are in favor of the full transition to only underground mining. Despite any changes in the set of decision parameters and gold prices, the incremental open-pit head grade will not be possible to obtain. This results from the fact that the calculations indicate a deficiency of required gold ounces based on the remaining open-pit reserves.

It may be concluded that the potential transition from open-pit to underground operations with a reduction in yearly throughput, is optimized through a full transition to underground mining. The planned option, with the cohering plant modification, is proven to be optimum and is currently considered as the solution for future mining at Akyem Gold Mine.

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Delft University of Technology, The Netherlands Thijs Goense
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Acronyms

NPV Net Present Value

OP Open-pit

UG Underground

GA Genetic Algorithm

ACO Ant Colony Optimization

TSF Tailings Storage Facility

NGC Newmont Goldcorp Corporation

ASZS Akyem Shear Zone System

STP Short Term Planning

MTP Medium Term Planning

LOM Life-of-Mine

LTP Long Term Planning

SMU Smallest Mineable Unit

ROM Run Of Mine

SAG Semi-Autogenous

CIL Carbon-In-Leach

GTC Grade-Tonnage Curve

DCF Discounted Cash Flow

BP Business Plan

UPL Ultimate Pit Limit

MILP Mixed Integer Linear Programming

Part I

Introduction

Chapter 1

Introduction

1-1 An introduction to the Akyem Gold Mine

Mining is the process of extracting a beneficial naturally occurring resource from the earth (Ben-Awuah et al. (2016)). This process may be conducted in several ways depending on the characterization of the resource deposit. The main methods of extraction can be associated with either open-pit or underground operations. Open-pit operations are likely to be conducted in case of shallow deposits, whereas underground operations are practised for deeper deposits.
Many deposits have the possibility of transitioning from open-pit to underground operations or vice versa (Bakhtavar et al. (2009)). As some active open-pit operations deepen; the costs of mining are likely to reach a certain economic threshold, where the transition to underground operations prove to be more feasible (MacNeil and Dimitrakopoulos (2017)).

The Akyem Gold Mine, in Ghana, is an example of an open-pit mine which is nearing depletion of its current open-pit reserves. Future mining activities are currently being evaluated by Newmont Goldcorp Corporation, having full ownership. The mine has been successfully running open-pit mining activities from 2013 onward but will potentially transit to underground operations in the near future. The mine is located northwest of the country capital, Accra. Figure 1-1 shows the geographical location of the mine together with a site overview.

This potential transition from open-pit to underground mining will require substantial large capital expenditures related to the underground operations. Due to the necessity of these expenditures, the utilization of existing mine infrastructure and equipment will be highly beneficial. In return, the access to incremental gold reserves will extend the Life-of-Mine (LOM) benefiting both company share- and stakeholders.

The unique features of each deposit will require a specific solution depending on the goal and objective of the project. For this reason, Newmont Goldcorp Corporation (NGC) has provided the opportunity for the conduction of this Masters thesis to analyse and evaluate the possibilities related to the transition in mining activities. A site visit has been made from March, 2019 until June, 2019. This has allowed for a general appreciation of the project

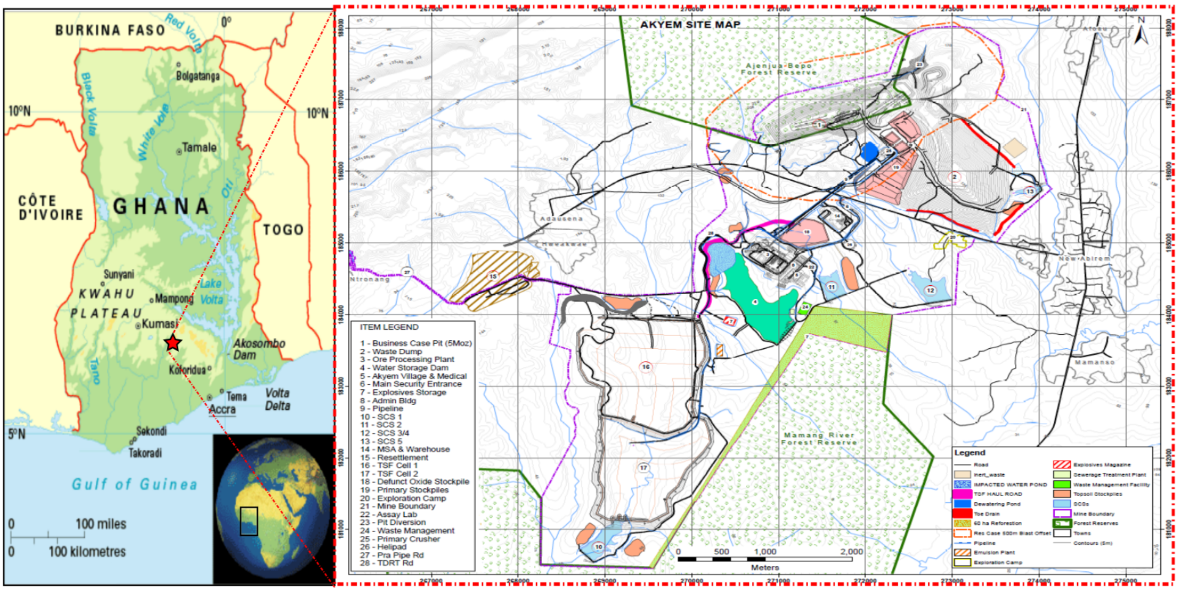


Figure 1-1: Geographical Location of Akyem Gold Mine With Schematic Site Overview
Consult Appendix B-1, for an enlarged copy of the image. *Modified from Survey Department*

and provisional insight to the transitional options. Such a transition of operation will require specific points of evaluation different to any other mine. Therefore, a set of unique evaluation methods will determine the optimum solution for future mining activities at the Akyem Gold Mine in Ghana.

1-2 Problem Definition

Open-pit mining operations are estimated to cease in 2023. The low-grade stockpiles have another two or three years before being exhausted. Meanwhile exploration teams have shown the potential of expanding into underground mining operations. According to current findings of different stages of feasibility studies, these underground operations could possibly go into full production by 2024.

The mine is currently facilitated to enable high production rates of relatively low-grade open-pit ore. However, the transition to underground operations would imply a significant reduction in annual ore productivity with higher gold grades. For this reason, the potential transition from surface to underground operations involves a great deal of strategic mine planning.

Issues regarding the utilization of present on-site equipment require extensive analyses. Potential options for the transition from open-pit to underground mining must be evaluated accordingly in order to present the trade-off between the viable options. Each specific option will be related to major changes within the operation. These options are related to a change in the total financial cost model and must be examined.

One of the challenges is related to the decrease in annual throughput of ore. The underground mine is estimated to produce up to roughly 1.6 Mt of ore annually. This is significantly lower compared to the current open-pit production of roughly 7.7 Mt of ore annually. Therefore, plans are being constructed to modify the processing plant that would seemingly optimize its configuration to allow for the maximum annual underground throughput. As a result the mine aims to lower its financial burden related to the direct and indirect operating costs.

On the other hand, possibilities are considered for processing incremental open-pit mineralized waste together with the forecasted underground material. The blend between these two sources of ore could potentially ensure the current maximum throughput capacity. Hence, there will be no excessive operating costs for maintaining current processing plant configurations.

Figure 1-2 presents a schematic illustration of the rudimentary options to be evaluated. The figure illustrates the three sources of ore and their potential routes to the processing plant. The uncertainties lie with the depletion of the open-pit ore and whether at that point the plant modification should be realised.
Options are visualized for a blend between incremental open-pit ore (orange) and underground material towards the current processing plant with potential throughput option (green). It is important to consider the trade-off that must be made between the required capital investment related to the plant modification, against the capital investment for additional Tailings Storage Facility (TSF) capacity. The other main option will be to solely process underground material through a modified plant, which will eventually be determined by the value generated.

It is evident that several options remain to be evaluated regarding changes in the processing plant and its subsequent throughput options. It will be essential to analyze and evaluate the optimum combination of decision parameters to search for a substantial solution regarding the transition in mining operations.

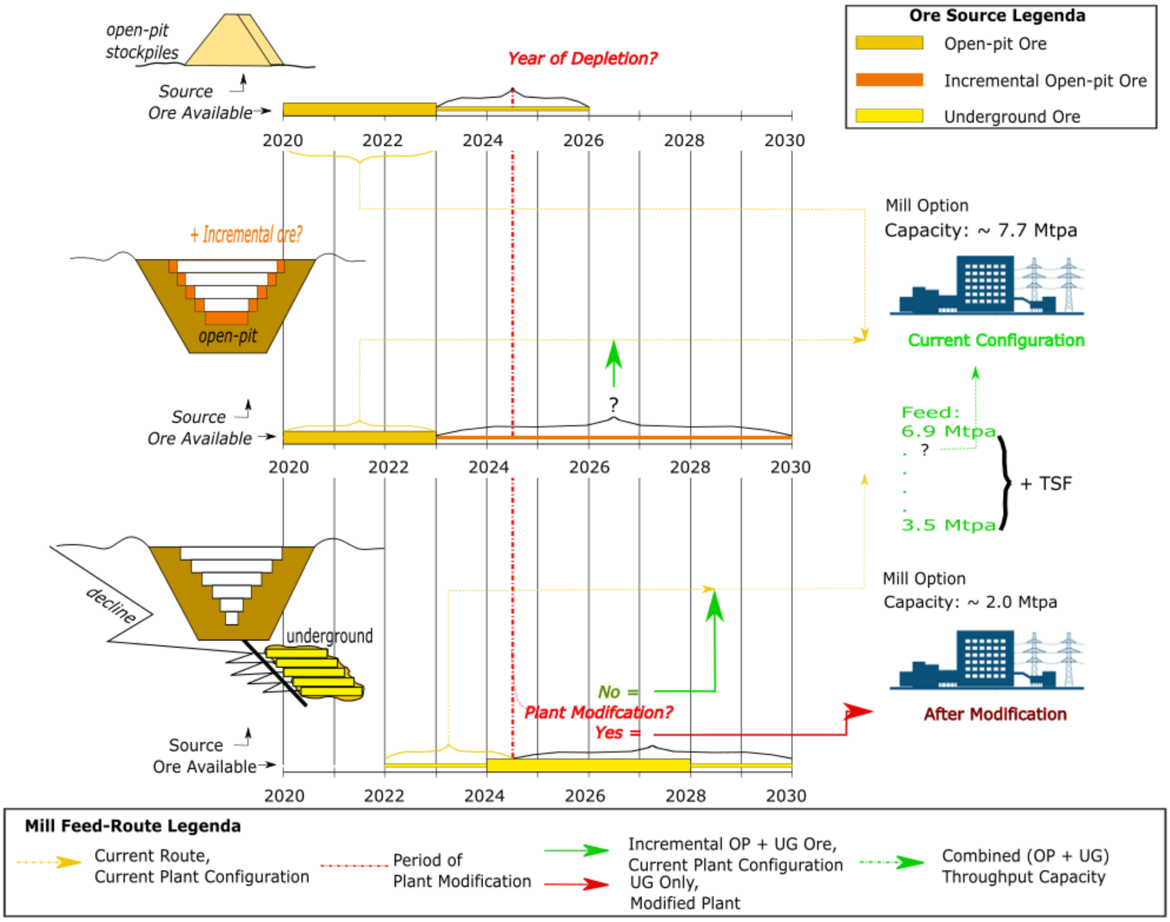


Figure 1-2: Schematic Illustration of Main Potential Options Regarding the Transition from Open-pit to Underground Operations

1-3 Goal and Objectives

The strategic mine planning will serve as the backbone of this thesis. It will present and support solutions for the questions related to the goal and objectives stated below.

The goal:

> Determine the optimum solution for the transition from open-pit to underground mining operations at an operating gold mine with respect to the significant reduction in yearly throughput.

Essentially, the objective of this thesis is to determine a strategic mine plan capable of yielding the optimum economic value for the potential transition from open-pit to underground mining activities. The focus is based on the utilization of the present processing plant where the trade-off between different decision parameters are to be presented.

The hypothesis:

> A reduction of the mill feed rate complemented by a correct blend of incremental open-pit and underground ore will generate more value, than modifying the mill and exclusively processing underground ore.

The hypothesis is tested by giving answers to the following research questions:

1. What is the optimum blend of open-pit and underground ore based on the planned production forecasts?
2. When would the potential plant modification be conducted?
3. What open-pit cut-off grade would obtain a head grade, which once blended with underground ore, would enable a higher profit compared to processing only underground ore?
4. What is the relationship between the throughput rates, head grades and recoveries?

The following sub-objectives that must be achieved in order to provide a solid foundation to support the answers to the research questions are:

1. Investigate if there will be enough mineable incremental ore to be stockpiled and blended.
2. Investigate the impact of capital investments related to the different expansion options.
3. Evaluate all options with regard to the time value of money.

1-3-1 Scope and Assumptions

The dynamic nature of mining activities result in a continuous change of available data sets. Data has been obtained through different departments each focused on different aspects of the project. The majority of the data is based on information reviewed within the Business Plan (BP)19.
The mining physicals and financial data are based on rudimentary high-level (long-term) available estimations and serve as a guidance for future prospects. Assumptions made for specific data estimations will be given beforehand and minor alterations are provided prior to each given result.
The following main data sets have been obtained:

- Open-pit Mining Physicals - Ore + waste tonnes, average grades and contained gold ounces
- Underground Mining Physicals - Ore tonnes, average grades and contained gold ounces
- Open-pit Block Model - Block model physicals from April, 2019, until the bottom of current open-pit reserves (Ultimate Pit Limit (UPL))
- Metallurgical Throughput and Recovery models - Hourly throughput and recovery percentages
- BP19 Total Costs- Combination of cost centres related to mining/processing/refining and site support other costs
- Underground Mining Costs - Combination of operating and capital expenditures related to underground mining activities
- Underground Processing Costs - Processing plant modification costs including operating costs
- TSF Project Expansion Costs - Capital development costs

Table 1-1: Research Scope

In-Scope	Out of Scope
Blending Optimization (OP and UG Recoverable Ounces)	OP Operations (Scheduling and Pit Limits)
Scenario Optimization (OP Head Grade Determination)	UG Operations (Scheduling)
Cut-off Grade Optimization	OP to UG transition depth optimization
Net Present Value Optimization	Processing Optimization

1-4 Thesis Outline

The thesis is divided into five parts with the following chapters:

1. **Introduction**

   Chapter 1: Introduction
   An introduction to the topic with an overview of previously conducted different methods for mine planning optimization studies.

   Chapter 2: Akyem Gold Mine
   An introduction to the Akyem Gold Mine including general information of the site and operations.

2. **Theory**

   Chapter 3: Literature Review
   A general overview of previously conducted strategic mine planning optimization studies, with short discussions on shortcomings relative to this thesis.

   Chapter 4: Cut-off Grade Determination Methods
   Brief introduction to three widely used methods for cut-off grade determination, in order to obtain a general appreciation of the main objective.

3. **Methodology**

   Chapter 5: The Applied Method
   Extended explanation on executed steps required for the blending procedure, Incremental cut-off grade determination and financial modelling.

4. **Results and Discussion**

   Chapter 6: Data Used
   Extended overview of exact obtained data with minor discussions on the reliability and assumptions made.

   Chapter 7: Results
   Presentation of results according to the structure elaborated in the methodology.

   Chapter 8: Discussion
   Discussion related to the methodology applied to the obtained results and produced results with their trends and feasibility aspects.

5. **Conclusion and Recommendations**

   Chapter 9: Conclusion
   Conclusion on the overall findings related to the scope of this thesis.

   Chapter 10: Recommendations
   Several recommendations regarding the path forward that can be provided based on the scope of this thesis.

Chapter 2

Akyem Gold Mine

2-1 Introduction

Prior to the literature review on mine plan optimization techniques, an introduction to the operations at the Akyem Gold Mine will be provided.
This chapter includes general information regarding the following aspects of the mine:

- The mine site
- Geology of the region and deposit
- Mining Operations
- Processing Operations
- Tailings Storage Facility

2-2 General Site Information

The mine is located in the Birim North District part of the Eastern region of Ghana located roughly 124 km northwest of the country capital, Accra. See figure 1-1. The total area of the site covers roughly 2000 ha of which NGC has acquired all necessary licences and leases, provided by Ghana's Mineral Commission by 2010. As such NGC holds the mineral rights to carry out its mining activities on the assigned land. The land surface is still owned by the lawful inhabitants and landowners, to whom NGC is responsible to compensate with a share of the profit being made as result of mining activities.

The site is located 3 km west of the district capital, New Abirem. Adjacent to the site amongst others are mainly houses, farmlands and several nature reserves including the Ajenjua Bepo Forest Reserve which the main pit slightly crosses its borders with. Appendix B-1 can be

Figure 2-1: Picture of Main Pit with Focus on Northern Footwall and Processing Facility
Left: Main pit with major slope failure. Toyota Land Cruiser (white 'dot') for scale indication. Right: processing plant with primary stockpile on right side with feed from gyrotary crusher.

consulted for a general appreciation of the surrounding Akyem mine site features, including all general on-site facilities.

These features indicate one of many challenges being faced by the company regarding space restriction for future on- and off-site expansion options and additional prospecting in the region. Figure 2-2 indicates permitted prospecting areas in blue, at both Akyem and Ahafo mines obtained by NGC, where Appendix B-1 shows the restrictive areas surrounding the mine site.

The site is adequately equipped with the necessary installations and facilities for efficient and responsible open-pit mining, processing and tailing treatment including on-site housing and recreation facilities. Power is mainly provided through a 161 kV overhead power line generated from fossil fuels. The site even includes a sustainable power source of solar energy through a 120 kW plant, powering the housing and mess facility throughout day times.

During the construction, which commenced in 2012, NGC and its contractors employed approximately 1300 people, which has gone down to roughly 800 once the mine went into commercial production by late 2013. The majority of the employees have been recruited and trained from the surrounding communities, to help develop the region.

A significant growth of the surrounding villages during the start of the mine shows the impact the mine has on the current local economy. Maximizing value together with extending mining operations would therefore be beneficial for all company stakeholders. The company continuously seeks solutions that enhance a sustainable future in order to maintain a prosperous and healthy future within the surrounding communities.

2-3 Geology

2-3-1 Regional Geological Setting

The key region of gold mineralization in Western Africa is found within the Ashanti Belt. The region is located in the southwest of Ghana which strikes SW to NE and is covered by lithologies of the so-named volcanic-sedimentary Birimian Supergroup and the overlying clastic sedimentary Tarkwaian Group. Extensive deformational folding and metamorphism, under conditions related to the greenschist facies during the Eburnean tectothermal event around 2.1 Ga, have been the predominant geological defining activities of the region Oberthuer et al. (1996).

The Ashanti Belt is one of five belts as part of the parallel northeast striking structural features of the folded meta-lavas Perrouty et al. (2012). The belts are bounded by regional tipping fault systems divided by a set of folded basins containing predominantly isoclinically folded pyroclastic and meta-sedimentary basins Dzigbodi-Adjimah (1993).

According to Oberthuer et al. (1996), the gold within the region can be associated to four different mineralizations:

1. Mesothermal steeply dipping quartz veins in shear zones mainly in Birimian sedimentary rocks;
2. Sulfide ores with auriferous arsenopyrite and pyrite, spatially closely associated with the quartz veins;
3. Sulfide disseminations and stockworks in granitoids;
4. Paleoplacers of the Tarkwaian Group.

The deposit associated to the Akyem Gold Mine is related to the first type. Where the gold is primarily hosted by lithologies within the regional shear and foliation zones formed under mesothermal conditions as a result of metamorphism and deformation at the time.

2-3-2 Akyem Deposit

The following data is based on the field data collected and interpreted by Henrichsen and Smithson (2010), on behalf of Newmont Mining Corporation, as part of initial feasibility studies prior to production.

The Akyem deposit is situated along the crustal scale Akyem Shear Zone System (ASZS). The ASZS is a structural discontinuity between the lowermost greenschist facies turbidite sediments on the Birim basin (Hanging Wall) and the sub-lowermost greenschist facies intermediate to mafic volcano-sedimentary rocks of the Ashanti Belt (Foot Wall). The system is situated within a thrust duplex characterized by a series of 070 trending primary shear zones and a set of 050 and 090 trending secondary shear zones that collectively accommodate 2-3 km of vertical displacement. The primary shear zone has resulted in the major Akyem Carbon Fault which initially separated the two major rock units.

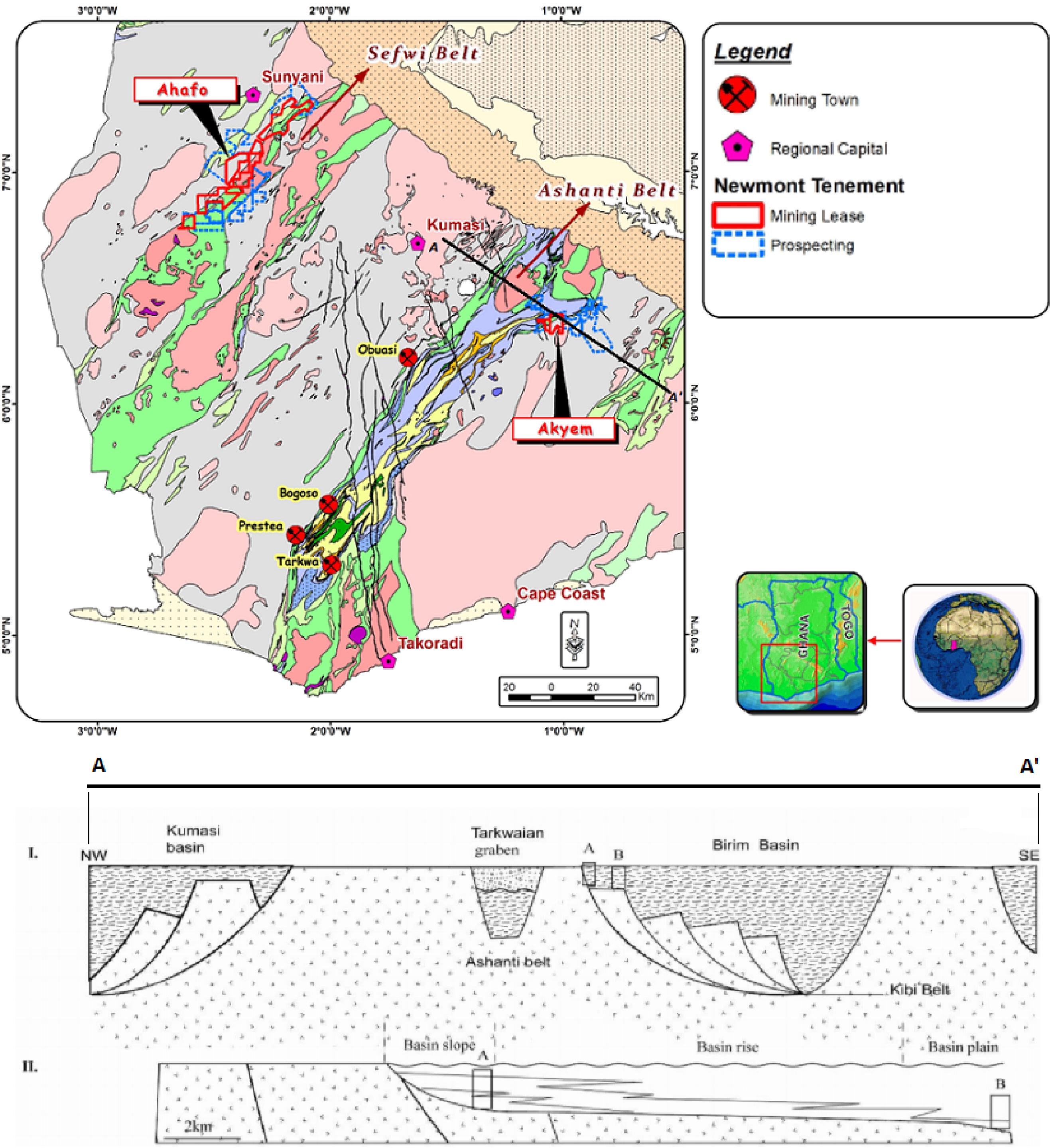


Figure 2-2: Geographical Location of Akyem Gold Mine Illustrated on Schematic Geological Map of Southern Ghana

Above: illustrating the regional geology and NE - SW striking feature of the Ashanti Belt.
Below: I) Cross-section through Akyem deposit of interpreted location of the hanging wall and footwall superimposed on the Ashanti gravity interpretation. II) Lateral facies changes within the interpreted back-arc basin depositional environment.
Modified from Henrichsen and Smithson (2010).

Comprehensively, the deposit is located within the hanging wall on the boundary of the 2050 m wide Akyem Carbon Fault. Gold mineralization is mainly concentrated in assemblages of brecciation, silicification and veining in the meta-sediments as part of the ASZS. Less defined zones of mineralization occur in multiple zones further into the hanging wall. Figure 2-3 shows a schematic cross-section of the UPL of initial 5 MOz reserve. The drill-holes have been added to indicate the potential for underground expansions, showing additional gold resources below the current pit. Furthermore, descriptions have been added of the mineralizations for both Metavolcanics and Metasediments lithologies.

The mineralogical alterations within the hanging wall of the Akyem Carbon Fault are characterized by strong to intense silica - sericite - pyrite - iron carbonate alterations. Four patterns of alteration assemblages, mineral zonation and gold grades are recognized at Akyem. Alterations range from assemblages close to the fault with abundant quartz, sericite, iron-carbonate, and pyrite to assemblages further away from the fault characterized by high amounts of chlorite, biotite, and green mica. Appendix C-2 can be consulted for a description of the lithologies of Akyem deposit.

Geotechnical Complicities

The occurrence of meta-volcanic and meta-sedimentary lithologies suggest relative competent rock types that allow for relative steep pit slope angels. The stratigraphic column shows the presence of saprolite, as a chemical weathered rock suggesting unstable transitional zones towards the primary rocks. These zones are critical to consider with further exposure as a result of waste stripping activities.

Geotechnical drilling prior to and during mining have provided the required information for the current slope design. The presence of the Graphitic Shear zone imply zones of lower stability, requiring additional precautionary measures during mining and monitoring. These weak zones are predominantly adjacent to the northern footwall requiring local adjustments to the slope design.

The northern highwall has experienced some instability throughout the years including a major failure in December, 2018 (see figure 2-1). Additional waste stripping activities help stabilize the slope, preventing similar events from occurring and were completed in June, 2019.

Besides the local adjustments to the high risk parts of the pit, the final slope configuration is defined by an overall slope angle of 45 degrees and an inter-ramp angle of 53 degrees. The slopes are divided into 8 m high benches with a batter angle of 70 degrees and a berm width between each bench of 9.5 m (pers.com: E. Blankson; 2019). These configurations are illustrated in figure 2-4.

2-4 Mining

Mining started full production in 2013 resulting in two open-pits; a larger main pit, which is still in production and a fully exhausted satellite pit east of the main pit (appendix B-1).

The main pit was initially estimated at 5 million ounces of gold. This had initially led to a daily production of roughly 80 kilo tonnes of ore with an average grade of 1.41 g/t. This

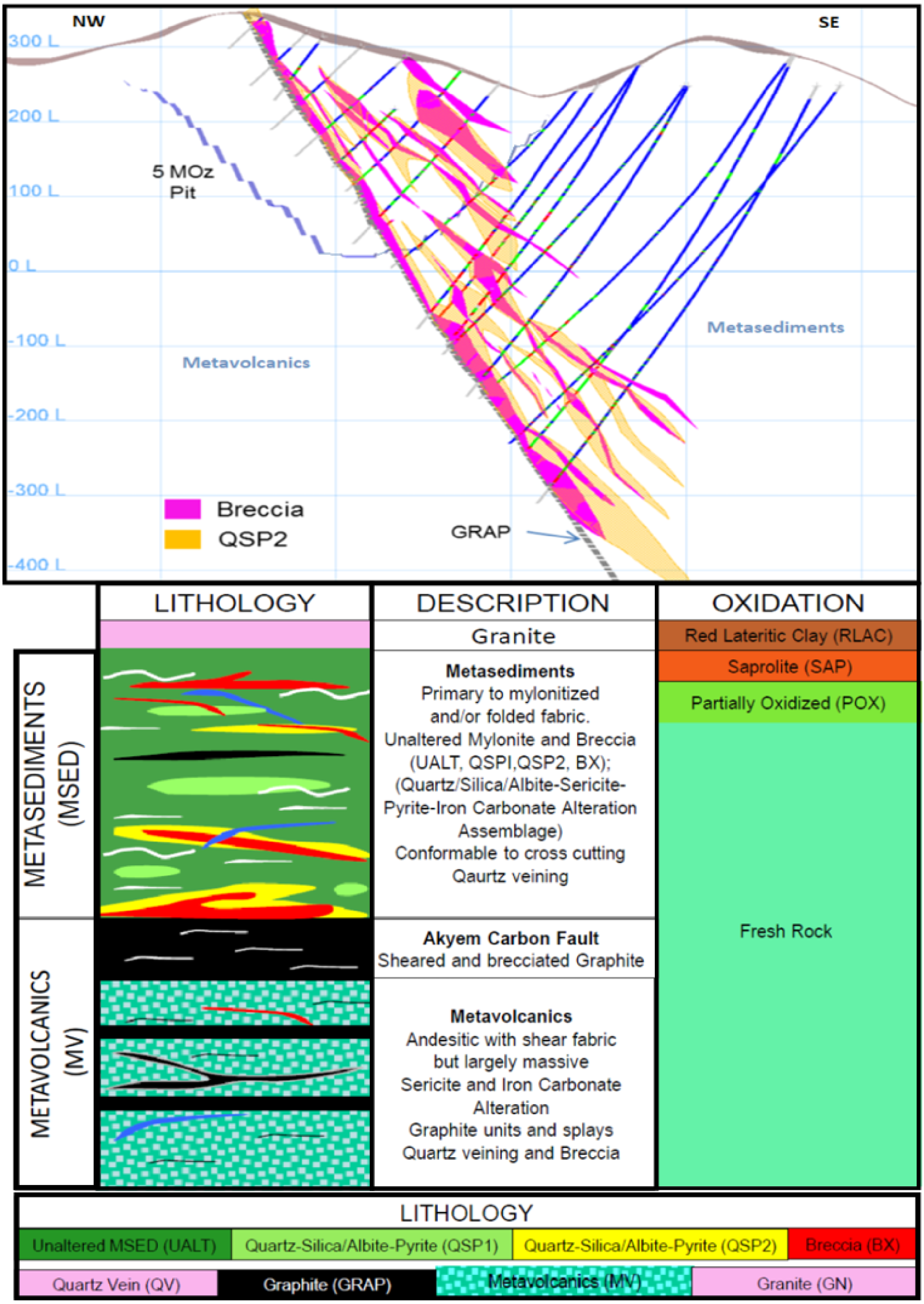


Figure 2-3: Cross-section of Akyem Deposit with Main Mineralization Zones and Akyem Carbon Fault (GRAP) Including Lithostratigraphic Description.
Modified from Geology Department.

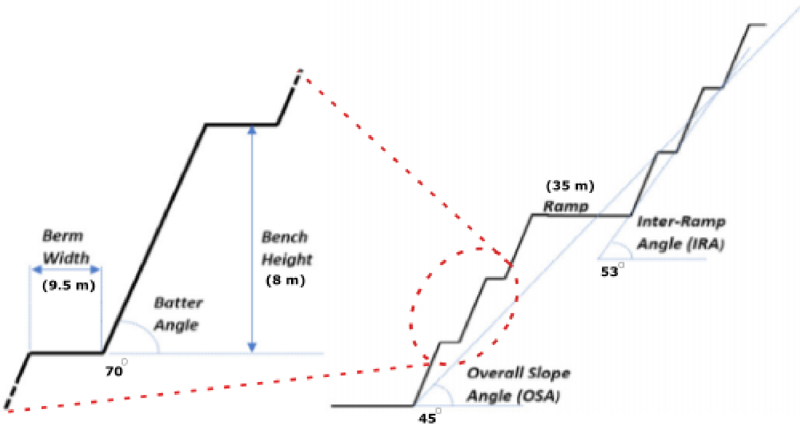


Figure 2-4: Schematic Illustration of the Slope Configuration of the Akyem Main Pit
Modified from Poniewierski (2018)

productivity was achievable due to the presence of the saprolitic material containing gold, which is currently exhausted. Current production is set at roughly 7.7 million tonnes of ore per year (21 kilo tonnes daily).

The dimensions of the current pit amount to roughly 1.6 km in length; 500 m in width and has a reduced level of 96 m (pers.com: R. Juati; 2019).

Open-pit mining activities are done according to the traditional cycle of planning, drilling, blasting, loading, hauling and crushing activities.

2-4-1 Planning

Depending on the budgeted forecasts and the geotechnical constraints, either waste or ore blocks are being implemented in the Short Term Planning (STP) schedules. These plans follow the physicals that are presented by the Medium Term Planning (MTP) group which is mainly responsible for the design of the actual pit.

The physicals are a result of the actual mineable polygons obtained by the MTP which fall within the ultimate pit limit, designed by the Long Term Planning (LTP) team. The LTP team conducts high-level mine planning optimization based on the geological block models provided by the geology department through Vulcan software.

The blocks in the geological block models have dimensions of 12×12×8 m and account for the Resource and Reserve estimations which divides ore and waste blocks on behalf of traditional break-even calculations (pers.com: F. Peprah; 2019). These blocks determine the tonnes and grade of mineralization which are presumably mineable.

Ultimate pit limits are based on Lerch and Grossmann (1965) optimization algorithms implemented as result, where the pit that yields the highest Net Present Value (NPV) together with cut-off grade calculations are used for following mine planning activities.

Further divisions of blocks are done to account for the technical parameters including modifying factors as dilution for the extraction of such blocks. Therefore mineable polygons are estimated with help of mine optimization software: MineSight. These mineable polygons are

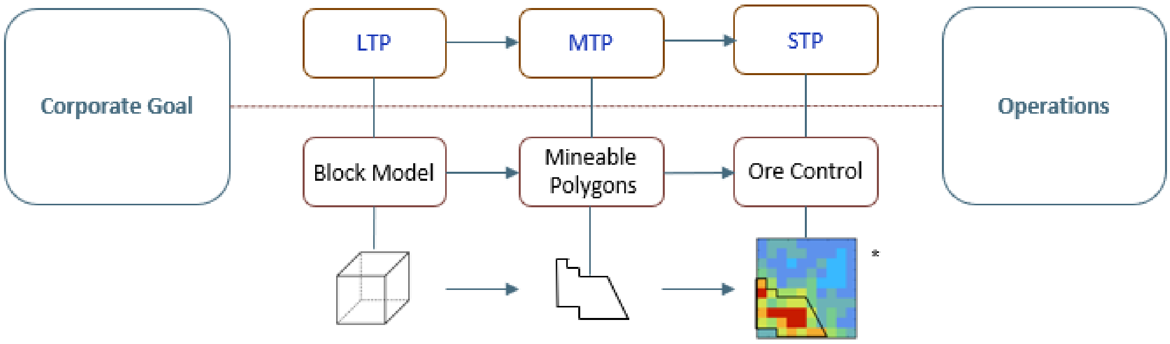


Figure 2-5: Schematic Illustration of Planning Flow Structure
* Ore Control illustration modified from Vann (2005).

generally sized accordingly: 24×26×8 m. The mineable polygons are the physicals used to determine whether production target will actually be met and serve as a guidance for the STP activities (pers.com: F. Kaba; 2019).

Finally, prior to extraction, ore control is carried out by the geologists to help define the actual boundaries between different cut-off grade ranges. This will ensure that the budgeted forecasts will have a higher likelihood of attainment by anticipating the diluted head grade that the mill will receive. These blocks are referred to as the Smallest Mineable Unit (SMU).

Figure 2-5 can be consulted for a schematic overview of the mine engineering planning process.

2-4-2 Drill and Blast

Drilling and blasting is done according to blast patterns designed by STP engineers in co-operation with the blasting engineers from the explosive supplier. Depending on the rock classification, different drill pattern sizes are used.

Production drill rig quantities range from 1 to 6 in BP19 based on the mineable polygons sizes. The current operations are related to primary material only, therefore the drill patterns (Burden * Spacing) range from 4.5 m * 4.5 m in primary ore to 5.0 m * 5.0 m in waste (pers.com: R. Juati; 2019).

In order to maintain the highwall, pre-splitting is carried out near the edges adjacent to the highwalls, on each bench. This controlled blasting techniques prevents excessive ground vibrations and controls the back-break. The spacing is 1.5 m with a dip of 70 degrees for the first 8 meters.

A buffer zone between the pre-split holes and production holes is fired after the production holes. These trim-shots have a pattern of 4.0 m * 4.5 m. This configuration can be consulted in figure 2-6.

2-4-3 Load and Haul

The mine is equipped with a fleet of 19 haul trucks that have a payload capacity of roughly 135 tonnes. The fleet is supplied by three excavators which are either assigned to mine out

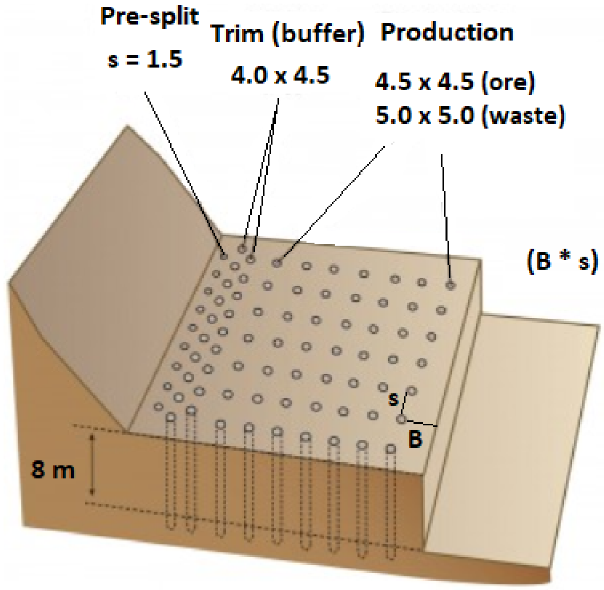


Figure 2-6: Schematic Illustration of Blast Pattern Design for Primary Material
Modified from Mining and Construction, Epiroc

ore or waste, depending on the STP. Trucks will be assigned to different operating excavators depending on orders from the fleet management system. The dispatch system is operated by the operators in the control room and continuously processes real-time data through Leica Geosystems software.

Loaded trucks hauling waste will be assigned to either the waste dump adjacent to the pit or to the TSF, to help support the outer walls and provide additional material for future raises for Cell 2. Ore hauls are mainly assigned directly to the gyratory crusher which is located outside the south-side of the pit. Excessive ore hauls will be stockpiled next to the crusher on a so-called Skyway (see figure B-1 in the Appendix). Stockpiled ore is quickly accessible and may be utilized for blending in order to meet certain production targets. Once ore is fed to the gyratory crusher, it will be transferred via a 2 km long conveyor belt to the primary stockpile located next to the processing plant (figure 2-1).

2-5 Processing

The Run Of Mine (ROM) is characterized by three different ore types, namely primary, transitional and oxide ore. The processing plant was initially designed to process an average of 8.5 Mt of ore per year. This was achievable due to the blend of oxide material, which flushes through the mill, together with the transitional and primary ore. The maximum amount of oxide blend was set at 30 % which basically prevents; the mill weight from overloading, settling rates in thickeners reaching its limits and blockages of inter-tank screens leading to tanks overflowing (pers.com: P. Aboagye; 2019).

The oxide material has been depleted since 2018, thus only primary and transitional material is currently being processed. Continuous changes to the throughput models are being made to account for the ROM; the head grade influences the throughput because with a higher head grade causes more gold to be lost to tailings. Therefore, by decreasing high grade throughput and subsequently increasing the residence time to enhance gold liberation; leaching would result in higher recoveries. A definitive trade-off is being made between higher operating costs for longer residence time with higher recovery, against the lower costs with higher throughput but increased losses to tailings.

2-5-1 Flow Diagram Processing

The current plant configuration is facilitated to operate continuously at 7.7 Mtpa. This is optimized accordingly by accounting for the estimated ROM and down-times. The configuration can be consulted on the processing flow diagram presented in figure 2-7.

The crushed ore from the primary ore stockpile is withdrawn through feeders and conveyed to the Semi-Autogenous (SAG) mill. The ore is crushed and grinded through the mill where its discharge is screened. Undersize slurry is pumped to a cyclone a bank and coarse oversize discharge is conveyed to a small bin connected to a pebble crusher and recycled back to the SAG mill.

Cyclone overflow is pumped through trash screens to filter out all unwanted organic materials before added to a pre-leach thickener. The pre-leach thickener will enhance leaching by ensuring that the slurry and carbon will react appropriately, due to similar densities before reaching the eleven Carbon-In-Leach (CIL) tanks. The cyclone underflow is transported to a secondary ball mill. Its discharge circulates back to the cyclone feed sump.

Sodium cyanide is added to the thickened slurry in the first CIL tank and is brought in contact with counter-current flows of activated carbon. The gold in the slurry is solubilized by the sodium cyanide and will bind onto the activated carbon. The loaded carbon moves on to a strip vessel where most of the carbon will be stripped. The stripping of gold is done through acid washing and elution. The stripped carbon is sent to a regeneration kiln and recycled to the last CIL tank for reloading. The gold concentrate as a product of elution will be refined further with help of electrowinning before being smelted and poured into doré bars of roughly 97%.

Tailings from CIL tanks are screened to recover most of the remaining carbon. The tailings are thickened in a cyanide recycle circuit where cyanide is recovered before the tailings are pumped to the TSF. Process solution is recycled by decanting to several reservoirs for purification and returned to the processing facilities.

2-6 Tailings Storage Facility

The tailings are stored in the TSF which is located southwest of the processing plant. The facility is divided into two cells which have a total storage capacity of 89 Mt (see figure 2-8).

Cell 1 has a capacity of 46 Mt and has reached its maximum. Construction of Cell 2 was finished and fully commissioned by March, 2019. The total capacity of Cell 2 is 43 Mt and is currently operating. The TSF is constructed to store tailings resulting from the projected

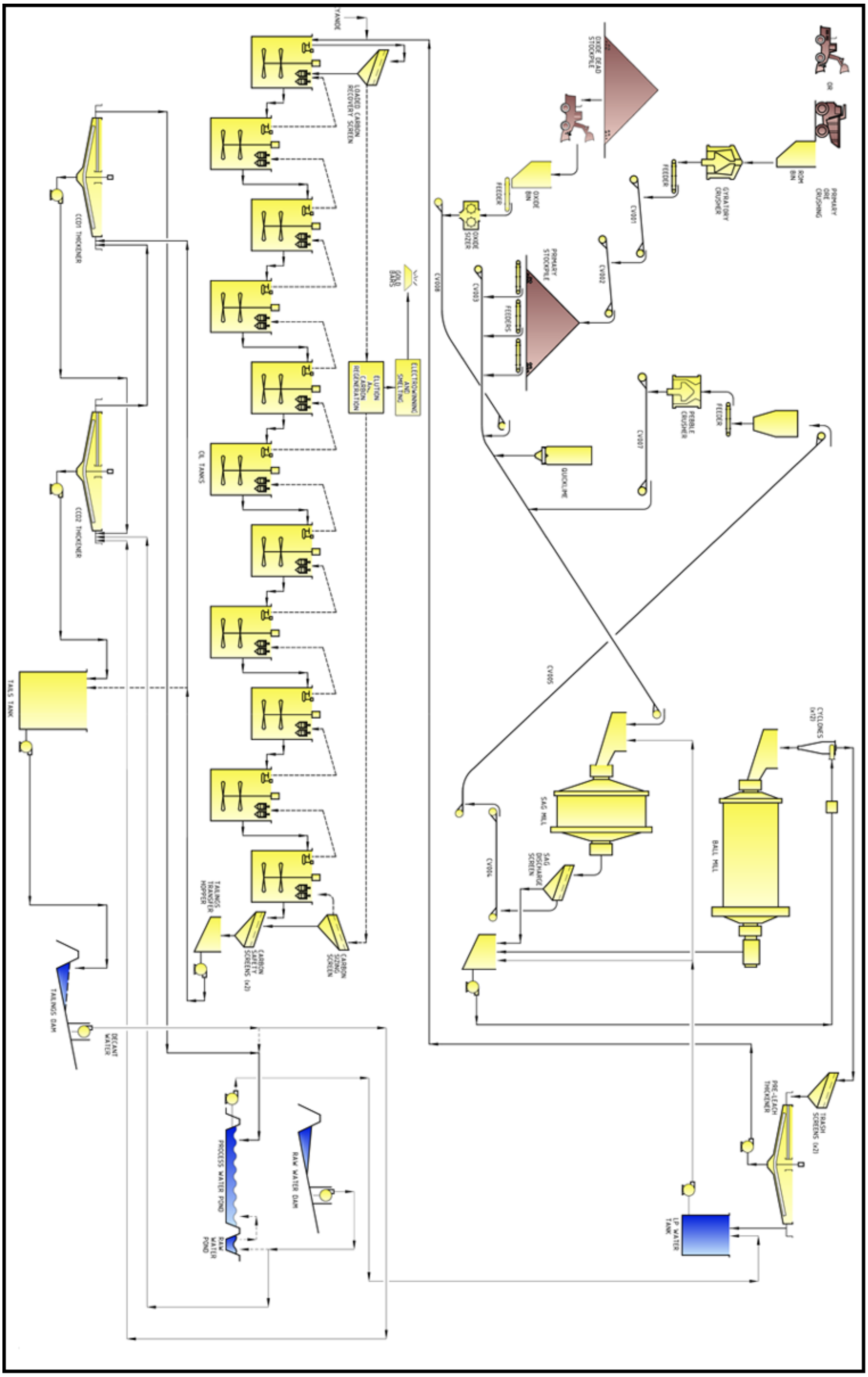


Figure 2-7: Current Processing Flow Diagram

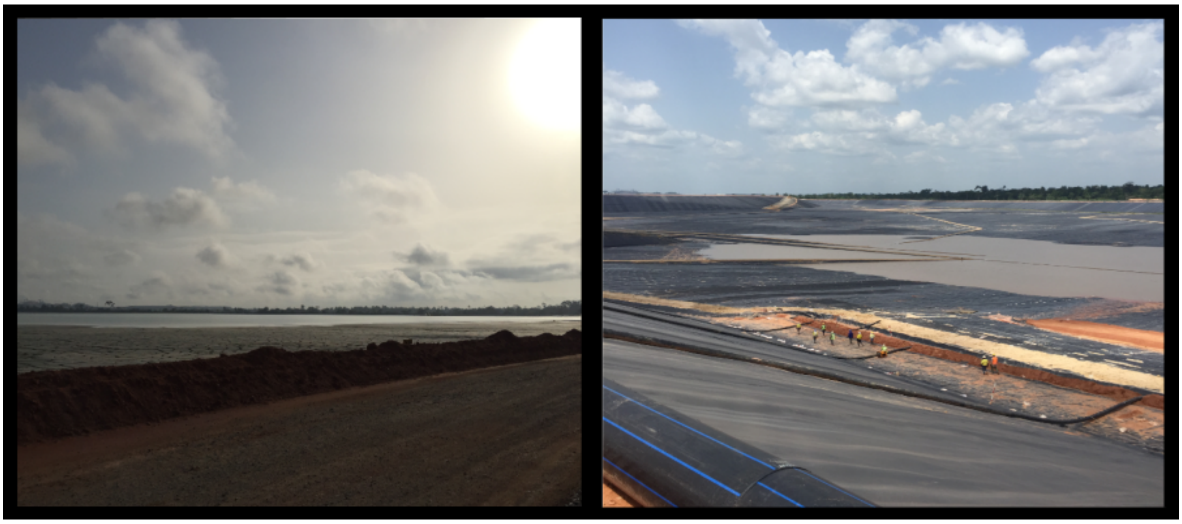


Figure 2-8: Picture of Tailing Storage Facility Cells
Left: Filled Cell 1 with a cross distance of roughly 1.3 km. Right: Nearly completed construction of Cell 2 with a cross distance of roughly 1.7 km.

production which excludes the underground project. This implies an additional raise is needed to store the underground ore tailings regardless of the recommendations of this thesis.

Both cells are designed with a basin underdrainage system. The underdrainage system reduces the phreatic surface and seepage (pers.com: P. Marx; 2019). Gravity causes the drainage to flow to a collection sump where recovered solution is pumped to a supernatant pond made available for reuse.
Both cells are complemented with emergency spillways to account for excessive discharge of rainwater reaching the storm water storage capacity.

Part II

Theory

Chapter 3

Literature Review

3-1 Introduction

A solution must be found for the transition from open-pit to underground mining operations. Specifically, an answer must be provided whether a full transition or a combination of mining activities is considered as optimum. The determination of an incremental open-pit cut-off grade will be key to provide answers to these potential cohering mining activities.

This chapter provides an introduction on the importance of strategic mine planning. General information on the misconceptions of its optimization techniques are discussed, which are all evolved around the cut-off grade concept. Furthermore, information on the known scientific research studies, that have focused on individual aspects of the mining value chain or as a whole, will be given. Additionally, relevant studies on known transitional optimization methods will also be presented and shortcomings to be discussed.

3-2 Misconceptions

The task of determining the correct cut-off grade is typically a misunderstood concept and has challenged many mining operations throughout decades. The general misconception that is being made is that the cut-off is determined by merely taking the costs for extraction and metallurgical recovery against the price of the mined commodity (Hall (2014)).

Considering the dynamic nature of such operations with continuously changing operating parameters, the cut-off grade should be thought of as a more complex principle. Especially when taking account for all different constraints further downstream of operations, the basic cut-off grade estimations techniques will turn out to be inadequate, resulting in value destruction and high probabilities of financial failures of projects (Ataei and Osanloo (2003)). Assessment of historical data by Newman et al. (2010) has proven the sensitivity of project profitability towards decisions regarding mine planning.

An issue regarding current mine planning is that the widely used methodologies for cut-off grade determination all seem to be following the same procedure; assumptions are being made that all parameters except the cut-off are fixed. This cut-off is subsequently used as an input to produce a seemingly feasible mine plan. This is thought of being intrinsic to these operations and having to be accepted since the cut-off ought to be driven by external factors over which the company has no control (Hall (2014)).

However, there are actually only two parameters over which a company has no control, namely the resource and the environment in which the company is doing its business (Hall (2014)). All the other decision parameters, a company can control thus allowing a different approach towards strategic mine planning. One where the decision parameters are combined to form an output which is used as strategic mine plan. The main difference here is that; instead of using the cut-off grade as an input, is to use it as an combined output of decision parameters. The key is to identify the option which would lead to the highest NPV while considering different variable decision parameters.

3-3 Strategic Mine Planning

A well constructed strategic mine plan is vital to generate and optimize profit over the life of mine. The execution of such planning is unrestricted by a fixed procedure as long as the corporate goal is prioritized. One of the most widely used key performance indicators used to quantify the value of mining and other company activities is the Net Present Value (NPV). Therefore, it is essential that the planning is done in accordance with the right combination of decision parameters to generate the highest NPV (Runge (1998)). The basic principle of each different strategic mine plan optimization technique can be associated to mainly two types, namely stochastic and deterministic methods (Cetin and Dowd (2002)). The stochastic methods use some form of randomness when searching for feasible solutions, whereas the deterministic methods uses a definitive starting point from which a sequence of possible solutions can be generated (Hickman (2014)).

None of the available methods would inevitably result in the optimum solution for each different case, due to the highly dynamic nature of mining operations. Each method having their possibilities and limitations. For this reason scientists are continuously developing and applying different techniques to all the stages of the mining value chain. As a result, extensive calculations prone to human errors have been a part of work. Only until recently optimizing algorithms, have been able to be applied to large scale mining operations, due to the exponential increase in developments of mine planning software capable of solving complex algorithms (Dagdelen (2007)).

3-4 Open-pit Applications

As mentioned regarding the scope of this thesis; the obtained data will allow for an higher degree of evaluation of open-pit options relative to the fixed underground data. For this reason, a brief introduction to the applied optimization techniques for open-pit operations will be given beforehand. For further information on the technical aspects of the variety of techniques, further reading on the provided references is recommended.

The techniques can be divided into four areas: the cut-off grade determination, production scheduling, mining limits and the mining value chain as a whole Kwiri and Genc (2017).

3-4-1 Cut-off grades

A big variety of optimization methods are being applied to the cut-off grade determination Kwiri and Genc (2017). The goal is to quantify the minimum amount of grade in order for the material to be economically mined. Essentially, all optimization techniques consider the same principle to determine the optimum cut-off grade subject to a set of constraints related to the mining, processing and refining activities. However, the cohering algorithms are built up differently. Challenges remain regarding the forecasting of the changing parameters throughout the process. Therefore, The cut-off grade is likely to vary over time with respect to the dynamic nature of mining activities.

Methods that account for the economical changes throughout the operations have been applied by Cetin and Dowd (2002) with use of genetic algorithms Genetic Algorithm (GA). This method can also be used in combination with additional optimization methods. For instance Ataei and Osanloo (2003) combines a GA with Golden Section search method. Yingliang (1998) uses the Control Theory weathered against the changes, whereas Xi (1985) applies a marginal index system together with Lanes theory Lane (1964).

Furthermore heuristics have been applied for maximizing NPV's. The results may not be optimal but the methodology is practical at hand, hence the solution can be produced in a reasonable time frame. Improvements are made by optimizing the cut-off grade with equivalent grade factors (Ataei and Osanloo (2010)). Additional genetic algorithms may also be applied to such heuristic methods. Craig et al. (2014) and Abrand et al. (2014) have implemented additional optimization algorithms to allow for blending multiple ore sources with stockpile options. These studies are considered as relevant however, the transitional aspect with utilization of current equipment is not accounted for.

3-4-2 Production scheduling

The second area of optimization techniques applied in mine planning is the production scheduling. The extraction sequence of mineable blocks within the pit boundaries is determined with respect the physical constraints and resulting NPV's Caccetta and Hill (1987). The applied methods include several linear and mixed integer programming methods, dynamic programming, fundamental tree algorithms and Lagrangian parameterization Kwiri and Genc (2017).

Franco-Sepulveda et al. (2019) used metaheuristics and artificial neural networks to prevent the extraction programming from falling into local optimums, but instead to find global optimization for dense data applications. Despite the advancement in computational power these methods remain restricted by such high data quantities, therefore less computational techniques are often applied Niemann-Delius and Sattarvand (2013). These methods consist of Ant Colony Optimization (ACO) by Shishvan and Sattarvand (2015) and Niemann-Delius and Sattarvand (2013), GA by Denby and Schofield (1994) and Simulated Annealing by Kumral and Dowd (2005).

An important aspect which is not being considered within the mentioned studies, is the implementation of the throughput-recovery relationships. Current approaches assume a fixed recovery percentage related to a fixed throughput rate. This will be an essential aspect of this study; as many potential different throughput options for current plant configurations are to be evaluated. Yap et al. (2013) studies these relationships by examining the impact on the NPV through mathematical programming. Notable results indicate that variable throughput rates increase the generated values.

3-4-3 Pit limits

Production scheduling is determined by the geotechnical constraints, whereas the pit limits are constrained by the economic value of mineable blocks within those pit limits. Pit shells are generated by maximizing the value for a given volume Hall (2014). The basic principle for this can be considered as a simple break-even calculation where the economic value of a block determines whether the uneconomic block above it is worth moving Dagdelen (2007).

Mostly used method applied to the mining limit is the Lerchs-Grossmann method, where the ultimate pit limit is defined by dynamic programming. The mineable blocks are calculated based on the individual block's costs and mineral content with regard to the physical constraints Lerch and Grossmann (1965).

3-4-4 Mining value chain

All the previously mentioned optimization methods only apply to one specific part of the entire mining value chain. New methods are being developed that target the entire mining value chain and to subsequently find optimum combinations of decision variables. The enterprise optimization method is the mainly used method developed by Whittle (2010) and applied in Whittle and Burks (2010).

The Enterprise method allows for simultaneous optimization of production stages and minimizes the limitations other optimization techniques have Kwiri and Genc (2017). The downside of this method is the required vast amount of knowledge on the data. This is typically a challenging aspect for its application on large operating mines, where there is less coherence between the different compartments.

The method is most similar to the method which has been applied for this project. The main difference is that the enterprise method will find its optimum through linear programming and search algorithms but the trade-offs between different scenarios will not be made visible. Additionally, the relationship between throughput rates and recoveries are to be made visible.

3-5 Open-pit to Underground Mine Transitions

The application of optimization techniques to different aspects of open-pit mining operations have proven to require a good understanding of the fundamental basics of the operation. When considering the different aspects of underground operations, these optimization techniques are presumably irrelevant. However, following research studies show that; despite significant differences between surface and underground mines, strategic mine planning optimization can be translated to similar problem solving algorithms.

Many researches have been conducted on the transition from either direction of; I) open-pit to underground; II) underground to open-pit or III) both open-pit and underground concurrently.

MacNeil and Dimitrakopoulos (2017), Bakhtavar et al. (2009), Bakhtavar et al. (2012), Bakhtavar and Shahriar (2007), Whittle et al. (2015) and Newman et al. (2013) have focused on the optimum transitional depth regarding the geological grade uncertainties. Different stochastic and deterministic methods have been applied. These methodologies have failed to solve the problem in a three-dimensional space. Chung et al. (2015) tackles the three dimensional space issue by implementing Mixed Integer Programming with simulated realisations regarding the grade uncertainties.

These methodologies were all focused on the optimization of the transitional depth, thus related to options regarding potential utilization of present assets. More relevant work has been done by Ben-Awuah et al. (2016) and Roberts et al. (2013) which used several optimization techniques to evaluate the best options for the kind of transitions which would yield the most value.

Ben-Awuah et al. (2016) determines what operation will result in the highest NPV, when considering open-pit- underground- or both activities simultaneously through Mixed Integer Linear Programming (MILP). This method is associated to the determination of the mining method. The possibility of adding incremental open-pit mineralized waste to run a concurrent operation is not considered. Whereas, Roberts et al. (2013) conducts an optimization study based on an existing underground mine. Possibilities are looked into for open-pit transitions through dynamic programming methods.

MacNeil and Dimitrakopoulos (2017) optimizes several decision variables as mining, blending, processing for both open-pit and underground simultaneously through simulated annealing. However, these are applied for current running operations, thus the decision regarding the mining method is already made. Therefore, there is not necessarily a focus upon the trade-off for the adaptation of current on-site equipment.

3-6 Conclusion

It is evident that the whole concept of strategic mine planning optimization for open-pit mines is conducted in numerous ways and applied to different aspects of the mining value chain. The dynamic aspect of mining operations imply that none of the current method are robust for all situations. It is therefore essential to have good understanding of the objectives and how the obtainable data can be applied sufficiently to reach the final goal.

Similar applications have been applied to transitional mining operations from open-pit to underground, vice versa or both simultaneously. When confining to a fixed set of decision variables the findings of relevant research studies have proven to be sub-optimal. A dynamic strategic mine plan will have to be weathered against any possible financial changes throughout the production years. All possible combinations of decision parameters will have to be considered in order to present the optimum result.

The specific methodology for transitioning from open-pit to underground with respect to remaining ore reserves and the recovery-throughput relationship, has not yet been studied. The following chapters will present the methodology and its underlying principles.

Chapter 4

Cut-off Grade Determination Methods

4-1 Introduction

Rendu (2014) defines the cut-off grade as the minimum amount of valuable product or metal that one metric ton (i.e., 1,000 kg) of material must contain before this material is sent to the processing plant. This definition is used to distinguish the blocks within a block model of being characterized as either ore or waste. Additionally, the cut-off grade will determine whether the material is worth stockpiling or requires immediate processing. For this reason there is usually a range of cut-offs used for the designation of blocks to subgroups defined by the mineral grade content. These different ranges will allow for a comprehensible overview of mineral inventories and enhance blending of ore to achieve certain production targets.

In essence a cut-off grade will need to be determined for the potential incremental open-pit ore. Depending on the final goal, different methodologies may be utilized for the calculation of this cut-off grade. Figure 4-1 illustrates the incremental cut-off as the point of which material will be stockpiled or not, whereas the planning cut-off is a long-term estimation of material that would directly be assigned to processing plant.
Several widely used methodologies for the cut-off grade will be explained in order to obtain a general appreciation of cut-off grade concept.

4-2 Break-Even

Hall (2014) and Rendu (2014) will be followed very closely for the clarifications of the following cut-off grade theories.
Before the general equations will be given some of the widely used definitions of the utilized cost drivers will be given.

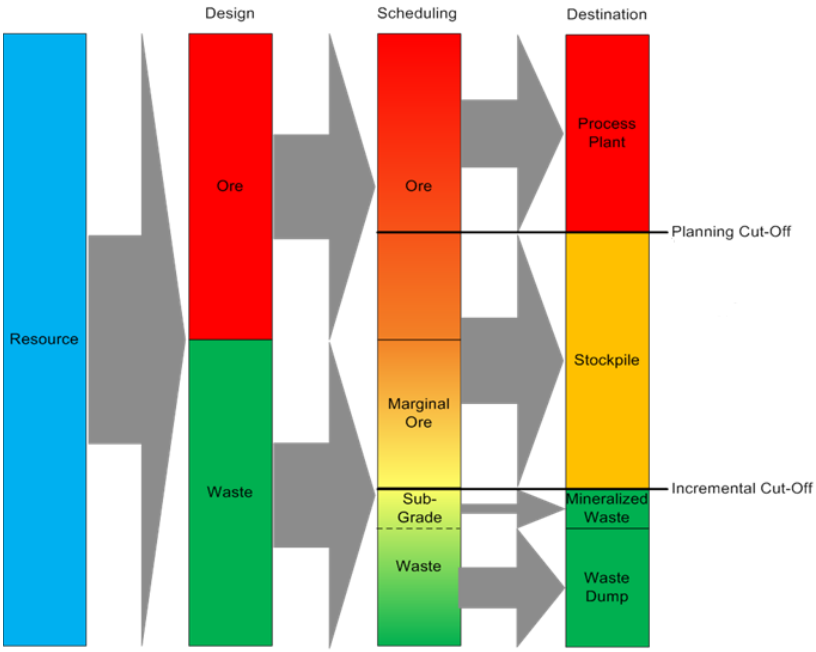


Figure 4-1: The Schematic Break-down of a Mineable Resource According to Cut-off Grade Designation
Modified from Akyem Cut-off Grade Sheet, 2018

Operating Costs

Variable Costs: costs which are directly influenced by the quantity of product being produced (e.g. mining costs referred to as dollars per tonne of ore or material mined).

Fixed Costs: costs which are not directly influenced by the quantity of product being produced. These costs will be the same within any specified time. Thus time would be the influencing factor for these costs (e.g. leasing costs referred to as dollars per year).

Capital Costs

Project Capital: capital which must be spent in order to purchase additional capability. The plant modification costs or TSF expansion costs are an example of such expenditures.

Sustaining Capital: capital which must be spent in order to maintain the current capabilities. These can be thought of as maintenance costs which are often related to the amount of product being produced thus often referred to as operating costs.

The basic concept of the break-even cut-off grade is that it is based only on financial parameters. It is the grade at which the obtained revenue is equal to the cost of production associated to that revenue. The revenue will be controlled by the recoverable amount of valuable material multiplied by the price of that valuable material.

The break-even cut-off grade can be expressed as followed:

$$Break\!-\!even\,grade\,[g/t] = \frac{Costs\,[\$/t]}{Product\,price\,[\$/g]\;\times\;Recovery\,[\%]}. \tag{4-1}$$

Generally, there is no fundamental agreement upon which costs are used for a specific break-even calculation. For this reason it must be clearly stated what costs will be included in the calculation. Four mainly used variants of such break-even calculations are:

- *Marginal break-even*: mining variable costs + milling variable costs
- *Mine operating break-even*: total mining costs + total milling costs
- *Site operating break-even*: total mining costs + total milling costs + total site administration and service costs
- *Full cost break-even*: total mining costs + total milling costs + total site administration and service costs + head office charges + allowance for capital

Disregarding what cost combination should be accounted for in the calculation, it is important to note that only the differential costs need to be considered. This can be explained through the following explanation.

The determination of the cut-off grade is based on the destination of the material in question. An answer must be given to whether the material should be processed, stockpiled or brought to the waste dump. Important to note is that the material will be mined out anyways. The destination with the highest obtainable value will be assigned to the blocks in question. This value is based on a set of parameters and can be implemented into the following equations for ore and waste, respectively:

$$Value\,of\,ore\,(x) = x\;\times\;r\,(V - R) - (M_o\;+\;P_o\;+\;O_o) \tag{4-2}$$

where,

x = average grade of valuable material

r = recovery of valuable processed material

V = value of one unit of produced valuable product

R = refining costs per unit of produced valuable product

M_o = mining costs per tonne of ore

P_o = processing costs per tonne of ore

O_o = overhead costs per tonne of ore

and

$$Value\,of\,waste\,(x) = -\,(M_w\;+\;P_w\;+\;O_w) \tag{4-3}$$

where,

M_w = mining costs per tonne of waste

P_w = processing costs per tonne of waste (only applicable in few cases; for example, to prevent additional contamination of groundwater Rendu (2014))

O_w = overhead costs per tonne of waste.

Thus, the break-even will be reached when setting equation 4-2 equal to equation 4-3. Which would result in a similar equation as equation 4-1.

$$x = \frac{(M_o + P_o + O_o) - (M_w + P_w + O_w)}{r(V - R)}. \tag{4-4}$$

Hence, equation 4-4 proves that only differential costs would need to be considered for the calculation of the break-even cut-off grade.

It is evident that the costs all seems to be driven by a certain quantity being mined, processed or produced; these costs are all operating costs. However, capital costs must also be incurred in cut-off grade calculation. This is no issue as long as the cost behaviours and other cost assumptions are identified and clearly stated and consistently applied within the calculations.

4-2-1 Internal Break-even

Similar to the break-even calculation, where a destination must be assigned to material that will be mined regardless, an *internal break-even* can be determined. The material must be processed when the grade is high enough to cover the processing costs. The mining costs will not have to be covered since the material will be excavated anyways.

One requirement for the utilization of the internal break-even calculation is that the mining costs for ore and waste are equal (i.e $M_o = M_w$). The calculation is based merely on processing costs and recoveries. The resulting equation for the cut-off grade is as followed:

$$x_{internal} = \frac{(P_o + O_o)}{r(V - R)}. \tag{4-5}$$

Both mentioned break-even determination methods only take account for costs and revenues related directly to mining and processing activities. The potential influence of the quantity of material mined, the production schedule and the grade-tonnage relationship are not being considered.

4-3 Mortimer's Analysis

The use of the break-even cut-off grade analysis clearly has its limitations. The fact that each grade of valuable pays for itself will not ensure a certain desired profitability. Also, the geologic structural features within a deposit are not being accounted for. Therefore, another widely used cut-off grade concept has been developed by Mortimer (1950).

Where the break-even analysis only accounts for the costs related to the amount of ore being mined, processed and refined; Mortimer's definition accounts for all costs, including the waste material needed to be mined. The basic difference here is that it is not only giving answer to whether a block should be considered as ore or waste (i.e. *boundary cut-off*), but also that a desired profit is achieved.

The desired profit will be obtained by adding a required profit margin to the initial break-even calculation. The definition for this cut-off grade is referred to as a *volume cut-off*. Hall (2014) explains this cut-off as the minimum average grade of all the material classified as ore for a planning time frame; not just as a single block, and is the minimum head grade required to deliver the specified minimum profit. In other words, not every tonne pays for itself but the average tonne does. Thus, the average grade is the head grade which will lead to the desired profit. This head grade accounts for the variations of the grade distribution throughout the deposit.

The cut-off grade related to the definition of Mortimer (1950) should be the greater of the two options. It should be greater or equal to the break-even with additional costs and the desired profit should be achieved with it. This profit will be obtained by reaching a certain head grade for which the relationship between cut-off and head grade must be made clear. One important tool for this analysis is the grade-tonnage curve.

It must be noted that the head grade to reach the desired profit will be greater than the break-even cut-off, however the cut-off related to the required head grade may be lower. This may be the case when fewer costs must be covered in order to obtain a specific profit; when mines are closing down or other external revenue flows are obtained.

4-3-1 Grade-Tonnage Curve

The Grade-Tonnage Curve (GTC) is based on the resource block model produced by computer software that implements data from obtained drill hole data.
The graph can be used to determine the amount of tonnage and average grade above a certain cut-off grade. Typically the estimated cut-off grade according to the break-even calculation is presented on the horizontal axis and the sequential tonnage and average grade can be consulted on the the vertical axis left and right respectively. Figure 4-2 is an example of such curves.
In order to determine the cut-off grade that will produce the desired profit, the head grade must be estimated and read from the right vertical axis. This may then be followed left to the "Average Grade Above Cut-off Grade" line and down to the horizontal axis for the cut-off grade.

4-4 Lane's Theory

It is evident now that the with the proposed cut-off grade theories, no losses will be made and that a certain profitability is achieved. However, these applications will not ensure that

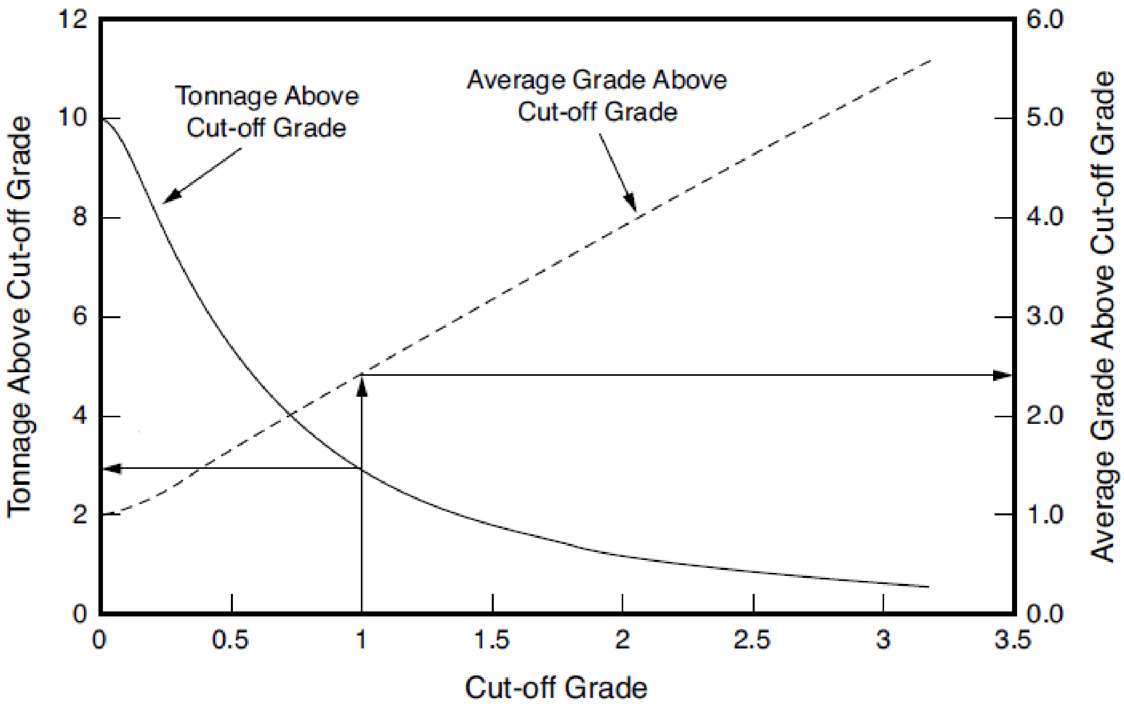


Figure 4-2: Example of Grade-Tonnage Curve

the resulting cut-off grade is actually optimized. In order to achieve such a cut-off grade Lane (1964) has developed another theory where the the capacity constraints and the time value of money are being considered. The basic principle of this theory will be briefly explained below. For further understanding, a consult in the work of Lane (1964) is recommended.

Lane's objective is to determine the cut-off grade which maximizes the NPV. In order to achieve this cut-off grade, Lane (1964) identifies the capacity constraints when treating the mined material (rock), ore (mineralized rock that will be processed) and product (that will be refined and sold).
For each of these three process stages limiting cut-offs will be identified which are equivalent to a break-even cut-off and constrain the maximum throughput of each stage. The cut-offs are each controlled by costs including opportunity costs, which account for the time-value-of-money.

Secondly, another set of three cut-offs will be identified that are based on the interrelation between the different processes stages. These cut-offs are referred to as balancing cut-offs. Balancing cut-offs consider the grade distribution and are not influenced by the changes in costs and prices.
The general concept that must be understood, is that it is important to identify the actual bottleneck within the process and consider the effect of certain debottlenecking actions regarding additional costs.

The optimum cut-off will be found when taking the middle values of each combination of the balancing cut-off with its two cohering limiting cut-offs for each process stage. Finally the middle value of those three observed middle values will be the cut-off. Hall (2014) can be consulted for a comprehensible illustration of this concept.

The effects of changes within the production activities are relatively simple to comprehend when there is a single flow of ore for a given mining sequence. Therefore this analytical technique can not be used to identify the optimum cut-off together with the optimization of the mining sequence for multiple ore sources. In order to account for these problems, additional number crunching techniques are needed to produce reliable solutions.

4-5 Conclusion

The methodologies show that different theories can be used for different goals. The actual break-even cut-off grade will only account for the costs and prices, for which a combination of different costs can be chosen. Mortimer's definition accounts for a similar break-even including the geological grade distribution to ensure a specific profit will be reached. Whereas Lane's method will optimize the cut-off considering capacity constraints and the time-value-of-money. When considering the objectives for this thesis, the present methodologies are currently not sufficient. For this reason additional methodologies will need to be applied in order to account for two different streams of ore.

Part III

Methodology

Chapter 5

The Applied Method

5-1 Introduction

The cut-off grade theories have provided a general appreciation of an important aspect of this thesis. Current planning cut-offs used by the mine are associated to the present open-pit configuration. Similar calculations have been made for the potential underground operation which are associated to the required plant modification. Hence, the two operations would operate according to their own specific cut-off.
For this thesis, it is essential to indicate whether the plant modification is actually necessary. Instead of this modification, incremental open-pit ore may be blended with the underground ore in order to fill-the-mill to potentially yield a higher NPV.
Therefore, an incremental cut-off grade will need to be determined which would ensure such results. Additional strategic mine plan optimization will be applied in order to ensure the optimum solution for the potential transition in mining operations.

The aim of this chapter is to present the necessary steps in order to give a solution regarding the goal of this thesis.

Key points will be to:

1. present the approach related to the objective;
2. present the metallurgical relationships;
3. present the blending optimization method;
4. provide the required equations.

5-2 The Approach

The decision whether a certain option will be realized is based on a specific measure of value. The option producing the highest value will have the highest likelihood of consideration. The

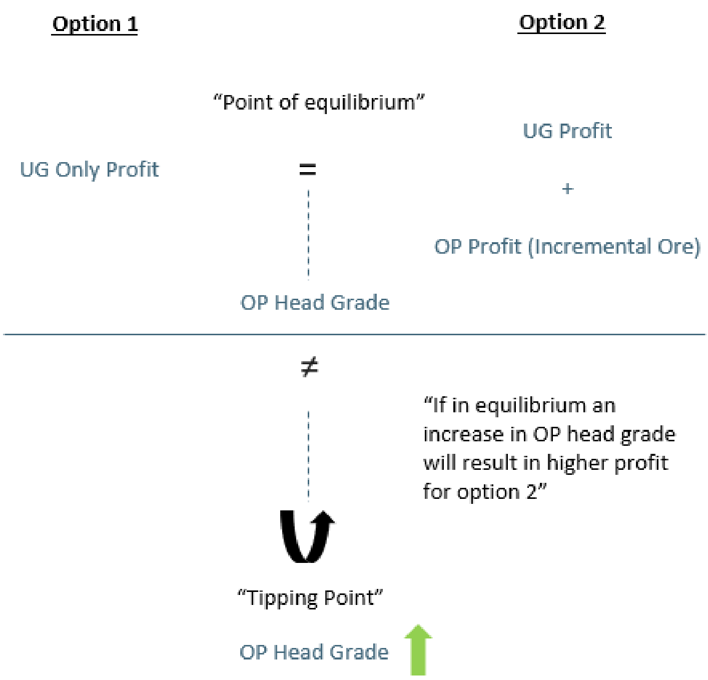


Figure 5-1: Schematic Illustration of Open-pit Head Grade Concept

acquisition of required permits will remain a critical aspect of such decision making. Despite the dependency on such permit acquisitions, the measure of value will be the determining factor for the results presented in this thesis.

Mirakovski et al. (2009) provides a comprehensible overview of the techniques used to evaluate mining projects. The use of the Discounted Cash Flow (DCF) or NPV has proven to be the most widely used measure of value within the mining industry. These measures of value will be used for overall comparisons of options.

The concept behind the calculation for the potential incremental open-pit head grade resulted from weekly discussions with Consultant Engineer - Underground Mining, Kristina Huss (pers.com: K. Huss; 2019). Kristina is a qualified Competent Person as defined under the SME Guide and currently part of the underground project under the supervision of Senior Study Director - Early Stage Projects, Steven Woods (pers.com: S. Woods; 2019).

The solution will prove whether a certain cut-off grade is applicable to ensure a specific open-pit head grade. This open-pit head grade will be blended with the planned underground ROM and processed through the current plant configuration with an optimized total yearly throughput. These results will have to generate more value compared to a modified plant where only underground ore is processed (i.e. the open-pit head grade will produce a revenue, together with underground ore, generating an equal or higher than underground only profit).

The minimum open-pit head grade will have to ensure that the two options are in financial equilibrium. Any increase in head grade at this point of equilibrium (*i.e. tipping point*) will result in a higher value, thus preference for the suggested option 2, with underground and incremental open-pit material, instead of option 1, with only underground material. See figure 5-3.

Figure 5-2: Approach Flow Diagram

Before the head grade calculation can commence it is essential to determine when the plant modification would occur. This will be determined by the blending of both open-pit and underground physicals. Figure 5-2 shows the flow diagram of the approach.

5-2-1 Head Grade

The determination of the required open-pit head grade is based on the following equation:

$$Open\!-\!pit\ head\ grade\ [g/t] = \frac{Incremental\ Net\ Revenue\ [\$]}{Incremental\ OP\ Throughput\ [t]\ /\ (\frac{Au\ price\ [\$/oz])}{31.1035\ [g/oz]})} \tag{5-1}$$

where the *Incremental OP Throughput* in the denominator is a decision variable, whereas the gold price is a given parameter.

The numerator from equation 5-1 is obtained through the following equation:

$$Incremental\ Net\ Revenue\ [\$] = (I.O.\ Costs\ [\$] + I.O.\ Profit\ [\$]) * (1 - Royalty\ [\%]) \tag{5-2}$$

where,

I.O. Costs: total costs associated to the extraction, processing and selling of the incremental open-pit ore.

I.O. Profit: the profit that will be obtained when extracting, processing and selling the incremental open-pit ore together with underground ROM instead of underground ore only.

Royalty: percentage of revenue that must be paid as royalty, based on the current gold price.

I.O. Costs

In order to obtain the values to be implemented in equation 5-2, a simplified cost breakdown has been developed. The costs have been assigned to three different scenarios. These scenarios including their cost centres can be consulted in table 5-1.

Table 5-1: Scenario Cost Centres

Cost Centre	(1) OP Only	(2) UG with OP configuration	(3) UG Only
Mining	✓	✓	✓
Processing	✓	✓	✓
Refining	✓	✓	✓
Mill Downgrade	✗	✗	✓
TSF Expansion	✓	✗	✗

The subdivision of the cost centres can be consulted in table 5-2.

Table 5-2: Cost Centre Divisions

Division	Mining	Processing	Refining	Mill/TSF
Opex	F & V (+ OD)	F + V	*	✗
Capex	S (+ CD)	S	✗	Project
G&A	F + Site V	F	✗	✗
Total	[$/t]	[$/t]	[$/oz]*	[$/t]

F = Fixed, V = Variable, OD = UG Operating Development, S = Sustaining, CD = UG Capital Development

These cost centres are presented providing a simplified overview of the cost structure related to the head grade calculation (equation 5-1). Each cost centre is influenced by several decision parameters which are linked within the financial model produced in Microsoft Excel. Assumptions that are made for each cost structure will be presented in chapter 6 followed by the produced results.

The total sum of the cost centres for all three scenarios is dependent on the yearly throughput. Hence the calculation for the costs in equation 5-2 is as followed:

$$I.O.\,Costs\,[\$] = \,Costs\,[\$/\,t]\,*\,Incremental\,Mill\,Feed\,[t] \tag{5-3}$$

The required project capital for the underground mine is not included in the head grade calculation. Reason for that is that these are considered as sunk costs. The underground development capital will be incurred before the potential plant modification.
However, these costs are included in the overall NPV calculation presented in chapter 6.

I.O. Profit

The profit from equation 5-2 is dependent on the generated profits from scenarios 2 and 3 (see table 5-1).

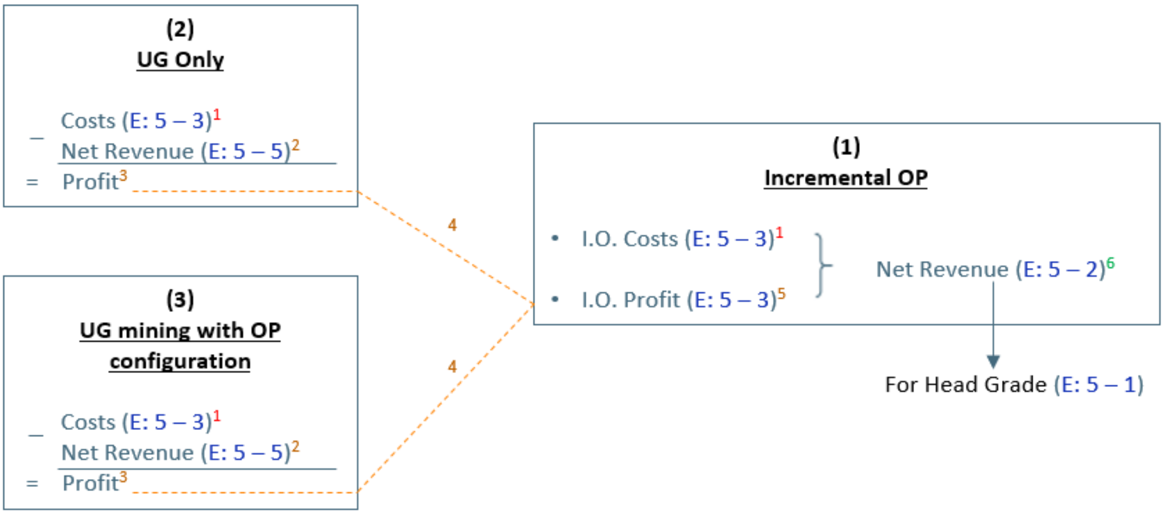


Figure 5-3: Illustrative Overview of Head Grade Calculation
Super-scripted numbers indicating the order calculations must be performed.

This can be explained through the point of equilibrium which must be reached, illustrated in figure 5-3. Therefore, the required profit will be obtained through the following equation:

$$I.O.\,Profit\,[\$] = Profit\,Option\,2\,[\$] \;-\; Profit\,Option\,3\,[\$] - Refining\,Option\,1[\$] \quad (5\text{-}4)$$

The refining costs will be deducted from the profit due to the different cost drivers when compared to the other cost centres.
The profit for the options 1 and 2 can be obtained when both costs and net revenue is known for both options individually.

The *costs* will be obtained through the same equations as 5-3 with the relevant cost parameters. The *profit* will be known through the deduction of the costs from the *Net Revenue* (5-5).

$$Net\,Revenue\,[\$] = Planned\,Au\,Production\,[oz] \,*\, Au\,Price\,[\$/oz] * (1 - Royalty\,[\%]) \quad (5\text{-}5)$$

The basic concept of the head grade calculation is presented schematically in figure 5-3.

One important notification is to consider the relationship between the throughput rate, recovery and head grade. Current methodology estimates the required average amount of gold in incremental open-pit material. The associated head grade is related to the throughput option together with the current gold price. This head grade and throughput relates to an amount of contained gold ounces, which will result in the required revenue. However, the contained ounces will not have a yield of 100%. Therefore, the recovery must be considered for the determination of the head grade. However, this recovery is dependent on the head grade in the first pace. So finally, this problem can be solved through a iterative procedure.

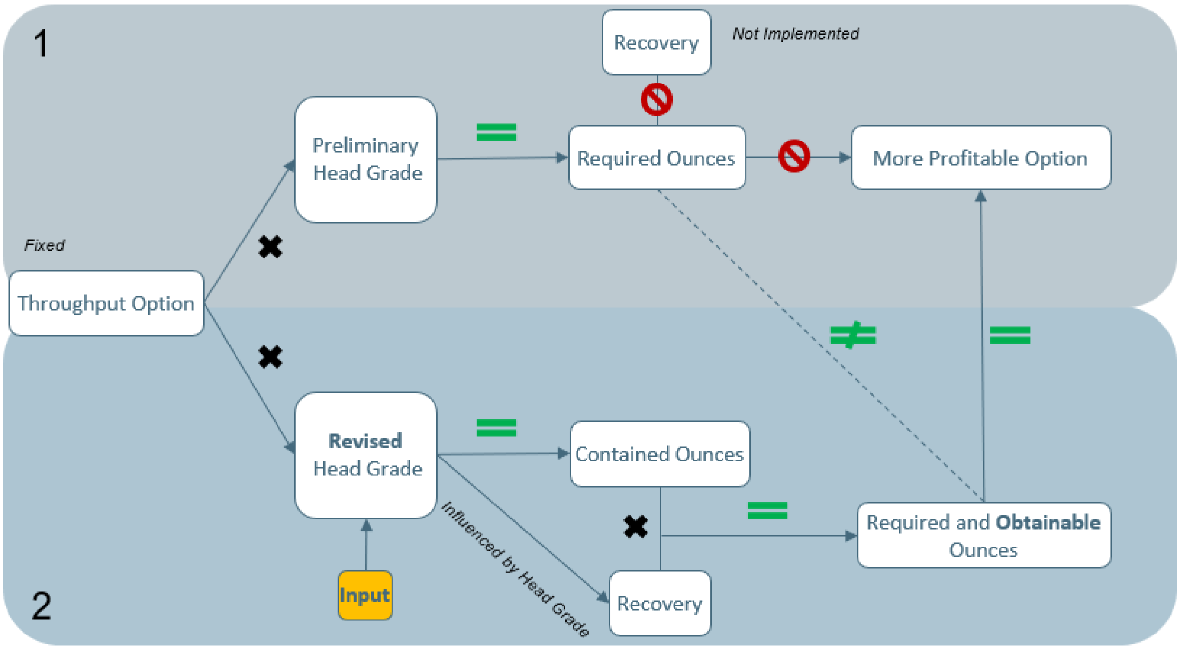


Figure 5-4: Illustration of Head Grade Revision Concept Accounting for Recovery Losses

Once the amount of required gold ounces is known, the iterative procedure can start. There will be three different input parameters, namely:

1. Throughput Rate
2. Recovery
3. Head Grade

This procedure implies that the initially calculated head grade will increase in order to comply with the recovery loss. Figure 5-4 illustrates this concept. The upper part, indicated with the number 1, illustrates the initial head grade calculation. The second part, indicated with number 2, illustrates the iterative procedure with the head grade as an input. The calculated required gold quantity from the first part is utilized for the second part.

5-3 Physicals

The head grade calculation starts with the cost determination (see figure 5-3). Costs are driven by the amount of ore being mined and processed, assuming that the stripping ratio remains constant for the incremental ore calculation.

The starting year of reporting for this thesis will be from 2020. Hence, obtained physicals from March 2019 regarding open-pit production and stockpile inventories were evaluated. Calculations were done in order to determine inventories from 2020 onward.
The physicals will be provided in chapter 6 and are influenced by the throughput and recovery models. These models are presented in the following subsection.

5-3-1 Throughput and Recovery

Open-pit

Previously conducted tests have provided formulas to estimate the total recovery for the processing activities. These tests have shown that with an increase in passing percentage of grinded material, the gold extraction would decrease. For this reason the plant is currently configured to to ensure an average grind of 75 μm with 80% passing (Metallurgy (2017)).
Due to the similar characteristics, the currently used equations are assigned for processing of both primary and transitional material.

The recovery is determined by the following equation:

$$Recovery\,[\%] = 1.0028 * (\,(0.0144\,*\,LN(HG)\,+\,(0.8712\,+\,0.015)) \tag{5-6}$$

where,

HG = head grade [g/t].

For the throughput the following equation is utilized and provided by the processing department:

$$Throughput\,rate\,[tph]\;=\;928.6\,*\,SAP^2\,+\,206\,*\,SAP\,+\,923 \tag{5-7}$$

where,

SAP = the percentage of saprolotic material, which is constrained to 30%. maximum.

Due to depletion of the saprolite, the total hourly throughput is set to 923 tonne per hour.

For this thesis, the maximum throughput rate is based on the budgeted forecasts for the BP20. These are estimated as a result of the probable availability and set to 7,723 kt of ore per year.

Underground

Tests have been carried out to predict the recovery percentages regarding the underground ROM. Results have shown similar recoveries as open-pit operations. The lack of actual processing experience of such material results in the utilization of the same recovery equation (5-6).
The maximum throughput rate is slightly altered down to 6.9 Mt of ore per year. Reason for this is that the underground material is characterized as more competent. Thus requiring a longer time in the SAG mill.

In case of the plant modification, the total throughput of ore is set to roughly 2.0 Mt per year.

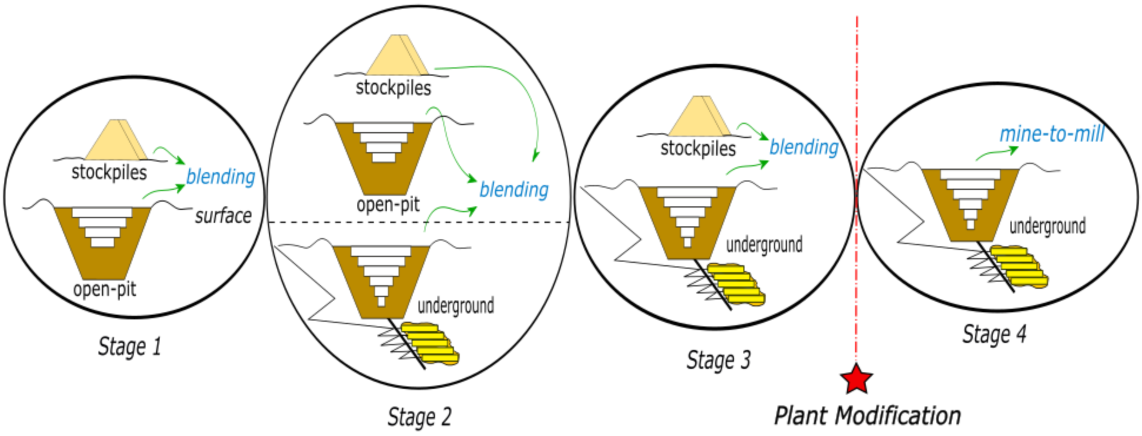


Figure 5-5: Schematic Illustration of Blending Stages

5-4 Blending

Blending of the different cut-off grade ranges is applied to meet specific production targets. This process will also be done to combine both open-pit and underground ROM to maximize the recoverable gold ounces.
All open-pit material above cut-off grade must be considered within the optimization prior to the modification. Reason for this is that the underground cut-off grade has been calculated for 3.0 g/t. Hence, the open-pit ROM will be considered as waste thus losing value when the mill has been modified.

The goal is therefore the following:

- Determine the yearly mill feed for the blend of different open-pit and underground ROM;
- Determine when the plant modification must occur.

Prior to the start of underground operations, the mill feed originates from two sources, namely direct open-pit mill feed and stockpile-to-mill feed (*Stage 1*).
Once the underground mine starts production, the blending consists of an additional underground ore mine-to-mill feed (*Stage 2*).
The open-pit mine-to-mill feed will have priority over the stockpiled material and will be depleted first. This results in a final blend of stockpiled material and underground ROM (*Stage 3*).
Finally, after the depletion of mine-to-mill and stockpile-to-mill feed, there will only be an underground mine-to-mill feed. This will indicate when the plant modification must take place (*Stage 4*). See figure 5-5 for illustration of the stages.

Blending Optimization

Depending on the stage illustrated in figure 5-5, there are three different input sources for the blending process:

1. Open-pit mine-to-mill feed
2. Stockpile-to-mill feed
3. Underground mine-to-mill feed.

The starting year for the evaluation is 2020 which implies that current stockpile inventory together with the planned open-pit physicals will be pushed up to December 2019. This will allow for a reasonable estimation of starting stockpile inventory.

The blending is optimized through a nonlinear algorithm using the generalized reduced gradient method (Lasdon et al. (1974)). The concept behind the optimization is based on mine planning experiences of MTP section of the mining department (pers.com: F. Kaba; 2019). The calculations were carried out by Microsoft Excel.

The algorithm is set to maximize the recoverable gold ounces related to potential throughput rates with cohering head grades. This is achieved by a set of decision variables. These decision variables are related to the three different input sources and can be consulted in table 5-3.

Table 5-3: Decision Variables per Input Source

Groups	OP Mine-to-Mill (1)	Stockpile-to-Mill (2)	UG Mine-to-Mill (3)
PHG	X_1, p	Y_1, p	✗
PMG1	X_2, p	Y_2, p	✗
PMG	X_3, p	Y_3, p	✗
PLG	X_4, p	Y_4, p	✗
PSG	X_5, p	Y_5, p	✗
UG	✗	✗	Z_1, p

where, p indicates the time period under consideration. UG = Underground ROM

The decision variables are assigned with an amount of recoverable gold ounces per time period. These recoverable ounces are dependent on the head grade and contained ounces presented in chapter 6, where the recoverable ounces will be determined through the recovery equation 5-6.

The objective function is set up to maximize the recoverable ounces within excel as followed:

$$Maximize\ Z = \sum_{n=1}^{5} X_{n,p} + \sum_{n=1}^{5} Y_{n,p} + Z_{1,p} \tag{5-8}$$

A set of constraints are set up to ensure that the decision variables would not exceed the inflow quantities. These set of constraints are as followed:

$$Z_{1,p} \leq UP_p \tag{5-9}$$

$$X_{1,p} \dots X_{n,p} \leq MP_{1,n} \dots MP_{n,p} \tag{5-10}$$

$$Y_{1,p} \dots Y_{n,p} \leq SI_{1,n} \dots SI_{n,p} \tag{5-11}$$

$$\sum_{n=1}^{5} X_{n,p} + \sum_{n=1}^{5} Y_{n,p} + Z_{1,p} = C_p \tag{5-12}$$

$$X_{1,p} \dots X_{n,p}\ ,\ Y_{1,p} \dots Y_{n,p}\ ,\ Z_{1,p} \geq 0 \tag{5-13}$$

where,

$_n$, indicates the the open-pit cut-off grade ranges (5-3);

UP_p, is the total amount of underground ore being mined within that specific time period;

$MP_{n,p}$, is the open-pit mine production per different cut-off grade group within that specific time period;

$SI_{n,p}$, is the stockpile inventory per different cut-off grade group available before the start of that specific time period;

C_p, is the maximum yearly throughput rate.

Note, that the 5-12 constraint is currently given as an equality to indicate the maximum capacity. However, after the plant modification the maximum capacity will be greater than the total yearly ore feed. Therefore the '=' is substituted for $\leq$.

5-5 Cut-off Grade and Determination Net Present Value

The combination of open-pit, underground and stockpiled material through the blending process will provide the physicals required for the head grade calculation (equation 5-1). An open-pit specific grade-tonnage curve will prove what cut-off grade is required to ensure the open-pit head grade.

The final evaluation will be based on the NPV as the measure of value. The calculation is done according to the following equation (5-14):

$$NPV = \sum_{t=0}^{n} \frac{R_t}{(1 + i)^t} \tag{5-14}$$

where,

i, is the discount rate (= 9 %);

t, is the year of evaluation;

n, is final year of evaluation;

R_t, is the total Net Cash Flow (5-15).

$$Net\,Cash\,Flow = Net\,Operating\,Cash\,Flow - Total\,Capital \tag{5-15}$$

Part IV

Results and Discussion

Chapter 6

Data Used

6-1 Introduction

This chapter will provide all the obtained data required for the approach explained in the previous chapter (5). The data is obtained from several departments each responsible for different aspect of operations.

Open-pit data has been provided by the departments on-site:

- Mining Physicals - Felix Adaania Kaba (MTP Mining Engineer)
- Metallurgical Models - Collins Adupoku (Chief Metallurgist) and Patrick Ansah (Metallurgist)
- Open-pit Block Model Numbers - Frank Peprah (Senior Geologist)
- Open-pit Stockpile Inventories - Theophilus Oduro-Bonsu (Senior Dispatch Engineer)

Finance Cost Summaries:

- BP19 related numbers - (Daniel Akinshola Famitumi)

The underground data has been obtained from NGC Headquarters in Denver:

- Mining Physicals including Costs - Kristina Huss (Underground Consultant Engineer)
- Metallurgical Data including Plant Modification Costs - Tom Logan (Principal Consulting Metallurgist)

TSF:

- Expansion Information and Costs - Peter Marx (EPCM Manager)

6-2 Physicals

6-2-1 Open-pit Production

The open-pit production is based on the mineable polygons obtained through Minesight planning software.

The current operational cut-off grade is estimated at 0.54 g/t. The ore has been divided into four groups that are currently being mined. See table 6-1.

Table 6-1: Ore Group Divisions

Groups		From [g/t]	To [g/t]
Primary High Grade	(**PHG**)	1.3	∞
Primary Medium Grade 1	(**PMG1**)	1.0	1.3
Primary Medium Grade	(**PMG**)	0.85	1.0
Primary Low Grade	(**PLG**)	0.54	0.85
Primary Sub-grade	(**PSG**)	0.42	0.54

The sub-grade material ranging from 0.42 to 0.54 is currently not being considered for processing within BP19. However, this material may prove to be profitable over the years due to changing economic conditions. Therefore, this material is currently being stockpiled with close access to the gyro crusher.

The total yearly productivity from 2020 onward can be consulted in figure 6-1. The figure shows a clear trend of a total 28 Mt of material being extracted in 2020 and 2021. This progresses towards a ramp-down period from 2022 to 2023 with respectively 16 Mt and 6.5 Mt extracted material. It is evident that with a decrease in stripping ratio the average grade increase significantly.

6-2-2 Stockpile Inventory

The stockpile inventory from April, 2019, has been obtained. These physicals were utilized within a blending procedure prior to the starting evaluation year of 2020. These numbers can be consulted in table 6-2.

Table 6-2: Stockpile Inventory April, 2019

Groups	Ore [kt]	Average Grade [g/t]
PMG	6,971	1.04
PLG	7,239	0.72
PSG	265	0.47

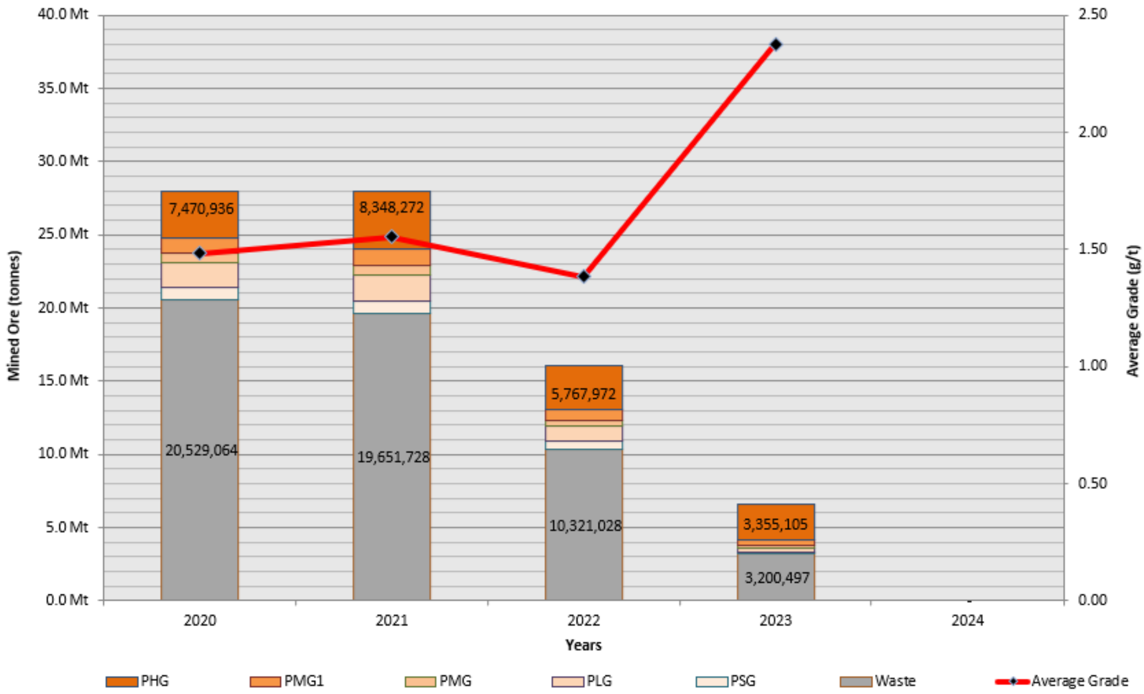


Figure 6-1: Open-pit Production

The higher ore grade divisions have been depleted prior to April, 2019. The blending process will give priority to the higher ore grade groups. For this reason the PMG will be processed first, followed by the PLG.

The open-pit production of May, 2019 until December, 2019, that is available for the blending process together with the stockpile inventory is presented in figure 6-2. This will indicate the amount of available stockpiled material to commence blending activities in 2020.

6-2-3 Underground Production

The underground mine physicals have been obtained in April, 2019; related to the BP20 physicals. Deswik planning software has been used to obtain the numbers presented in figure 6-3.

Operating development activities commences in 2020, whereas the actual ramp-up phase starts in 2022. Full production is estimated to start in 2024 with a total productivity of 1.6 Mt of ore. A ramp-down face is currently forecasted from 2029 until 2030 with a significant lower production of respectively, 1.3 Mt and 0.43 Mt of ore.

The total amount of ore being extracted through underground operations is currently estimated at roughly 10.9 Mt with an average grade of 4.12 g/t. These numbers amount to a total gold quantity of 1,443 kilo ounces.

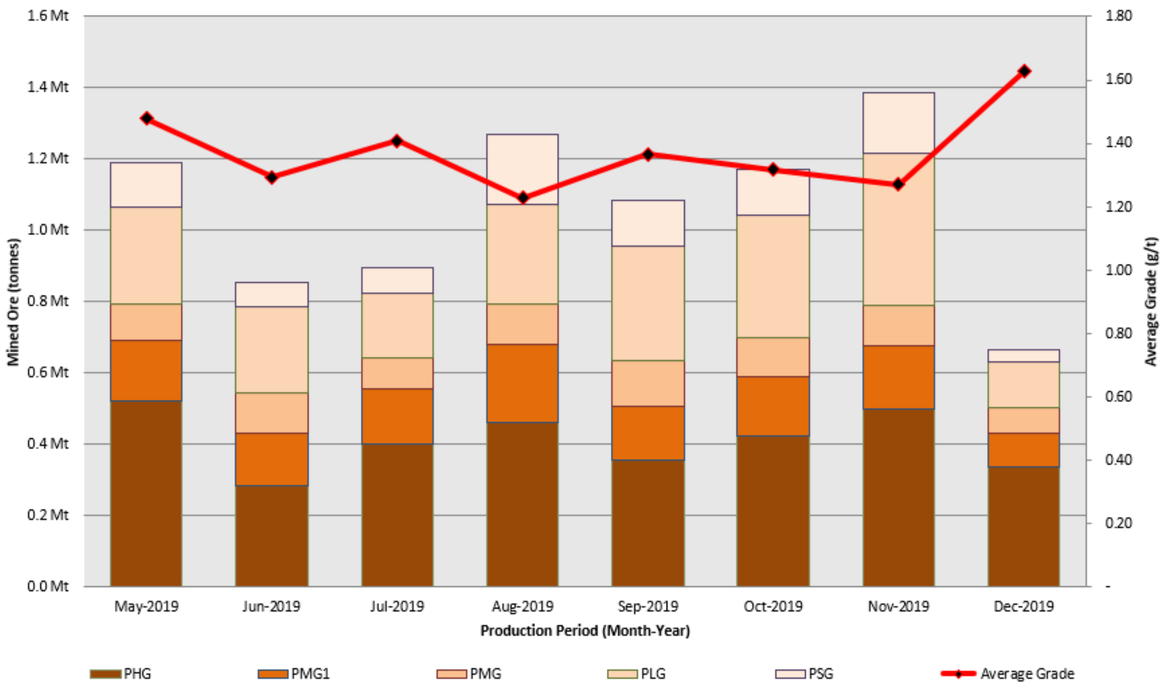


Figure 6-2: Monthly Residual 2019 Open-pit Production

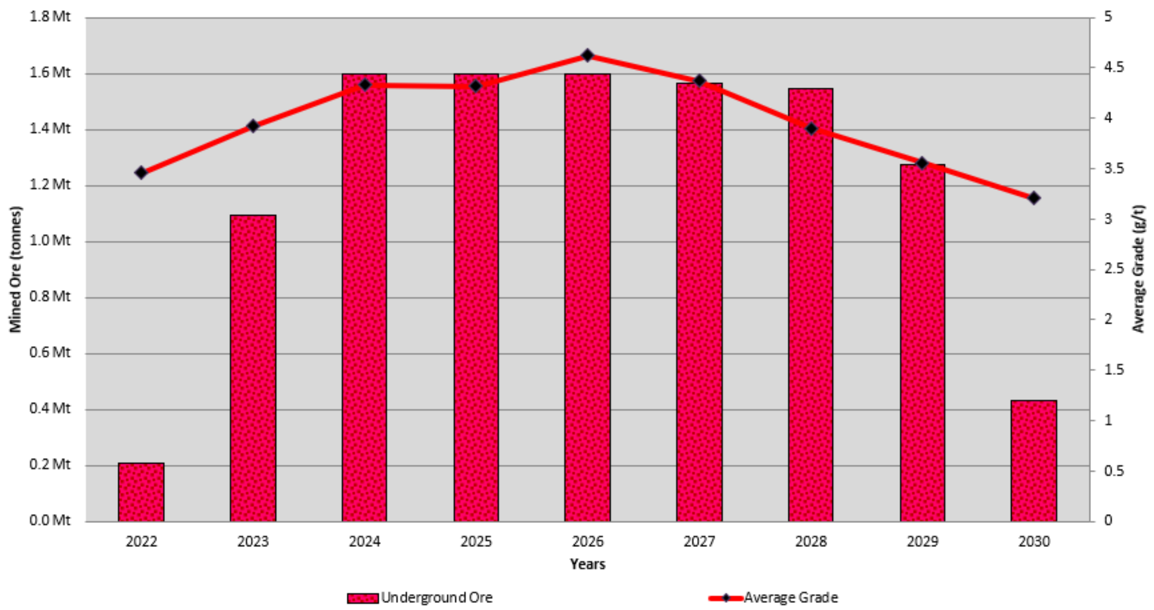


Figure 6-3: Estimated Underground Production as of April, 2019

Table 6-4: Sub-grade Ore Quantities

	PSG1		PSG2		PSG3		PSG4		PSG5		PSG6		PSG7	
	[kt]	[Coz]	[kt]	[Coz]	[kt]	[Coz]	[kt]	[Coz]	[kt]	[Coz]	[kt]	[Coz]	[kt]	[Coz]
2019	487	7,980	191	2,865	283	3,908	113	1,454	124	1,510	81	930	115	1,258
2020	395	6,485	227	3,398	271	3,749	112	1,449	109	1,334	149	1,720	122	1,337
2021	422	6,937	181	2,707	254	3,518	145	1,866	130	1,591	117	1,362	154	1,681
2022	227	3,712	117	1,742	191	2,647	95	1,220	92	1,114	77	895	115	1,257
2023	54	895	35	514	13	176	34	441	38	466	21	238	19	208
Total	1,586	26,008	752	11,227	1,012	13,999	500	6,430	494	6,014	445	5,146	525	5,740

Where, Coz = Amount of gold ounces available within the ore

6-2-4 Open-pit grade-tonnage curve

A grade-tonnage curve is produced according to the BP19 block model, as of April, 2019. The curve is presented in figure 6-4.
The graph provides information regarding the total ore tonnes (left horizontal axis) and average grade (right horizontal axis) above cut-off grade.
It is evident that the current pit contains 30.7 Mt of ore with an average grade of 1.71 g/t.

The sub-grades will be considered for further analyses and are associated to reasonable ranges slightly below current cut-off grade of 0.54 g/t.

This further division was made in order to quantify the amount of mineable polygons related to the potential sub-grades. The division for the sub-grades is provided in table 6-3. Note that the mineable polygons are associated to the current business plans, with a LOM until 2023.

Table 6-3: Sub-grade Group Divisions

Groups		From [g/t]	To [g/t]
Primary Sub-grade 1	(**PSG1**)	0.48	0.54
Primary Sub-grade 2	(**PSG2**)	0.45	0.48
Primary Sub-grade 3	(**PSG3**)	0.42	0.45
Primary Sub-grade 4	(**PSG4**)	0.39	0.42
Primary Sub-grade 5	(**PSG5**)	0.37	0.39
Primary Sub-grade 6	(**PSG6**)	0.35	0.37
Primary Sub-grade 7	(**PSG7**)	0.33	0.35

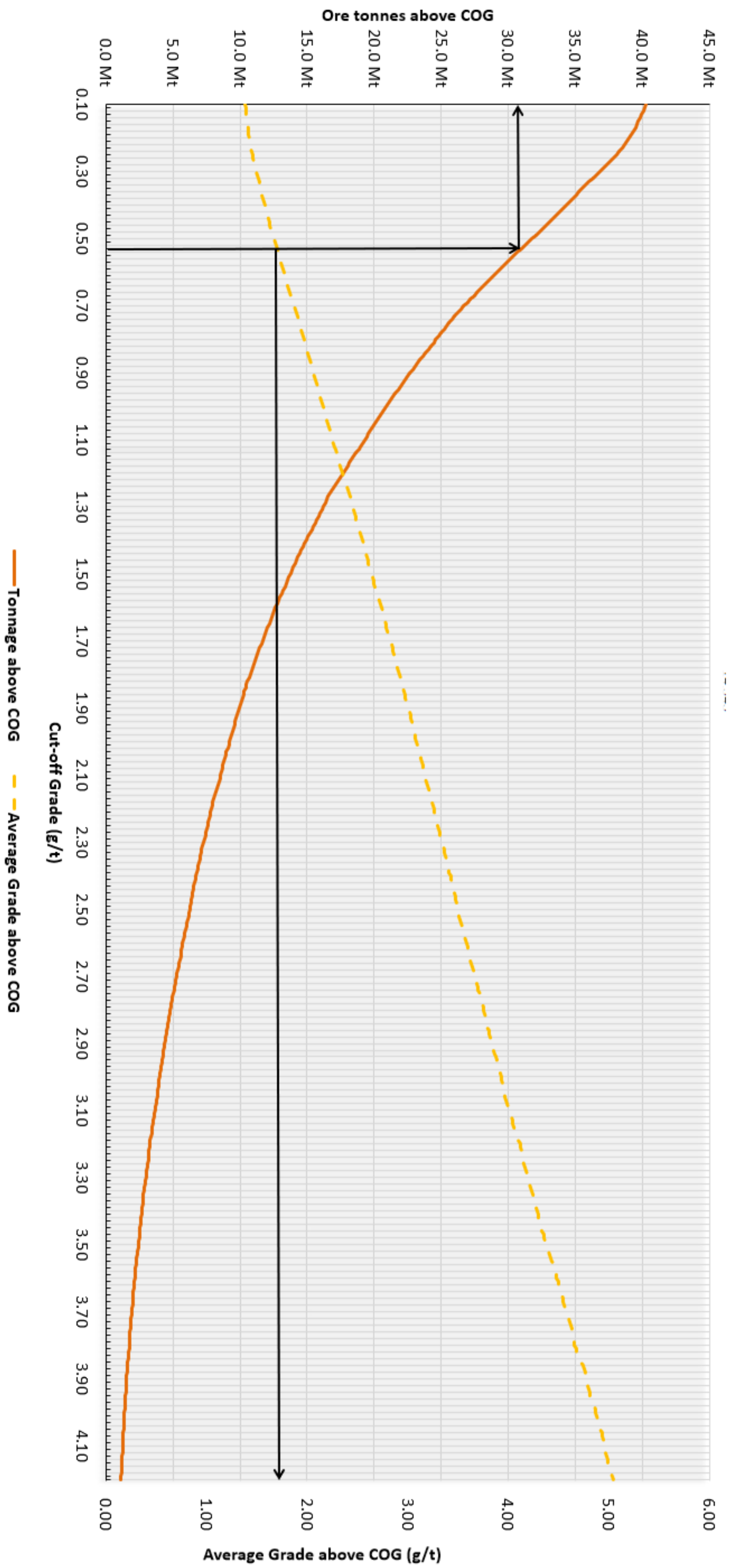


Figure 6-4: Grade-Tonnage Curve Based on April, 2019 Open-pit Block Model

6-3 Finance

The cost data is provided on behalf of the different departments as part of the site operations. The following section will comprehend these costs according to structure presented in table 5-2.

Open-pit operations will be considered as variables, whereas underground related activity costs will be presented as fixed physicals.

6-3-1 Operating Costs

Open-pit

Utilized cost factors can be consulted in 6-5.

Table 6-5: Open-pit Operating Cost Structure

Open-pit	Variable [$/t ore]	Fixed [M$/year]	Product Related [$/oz Au]
Mining	5.35	25.9	✗
Processing	7.85	23.0	✗
Refining	✗	✗	2.51

Mining related comments and assumptions,

- Based on 2.39 $/t material mined (mining costs, Akyem Cut-off Grade sheet, 2018)
- 40 % Fixed Costs (Full Potential cost translation)
- 60 % Variable Costs (Full Potential cost translation)
- Yearly maximum mineable ore tonnes = 7,745,000

Processing related comments and assumptions,

- Assumptions are based on BP19 Current Circuit from Logan (2019). Where, total variable costs + stockpile rehandling costs are considered as variable costs.
- Refining costs are based on the Akyem Cut-off Grade Sheet, 2018.

Underground

The total amount of operating costs related to underground mining operations is roughly $710 M. This amounts to an estimated 65 $/t of ore. The data is presented in figure 6-5. The figure shows the required operational costs prior to planned full production, excluding stoping and backfilling. Actual production activities are visible through implementation of *stoping* and *backfilling* activities from 2023 onward.

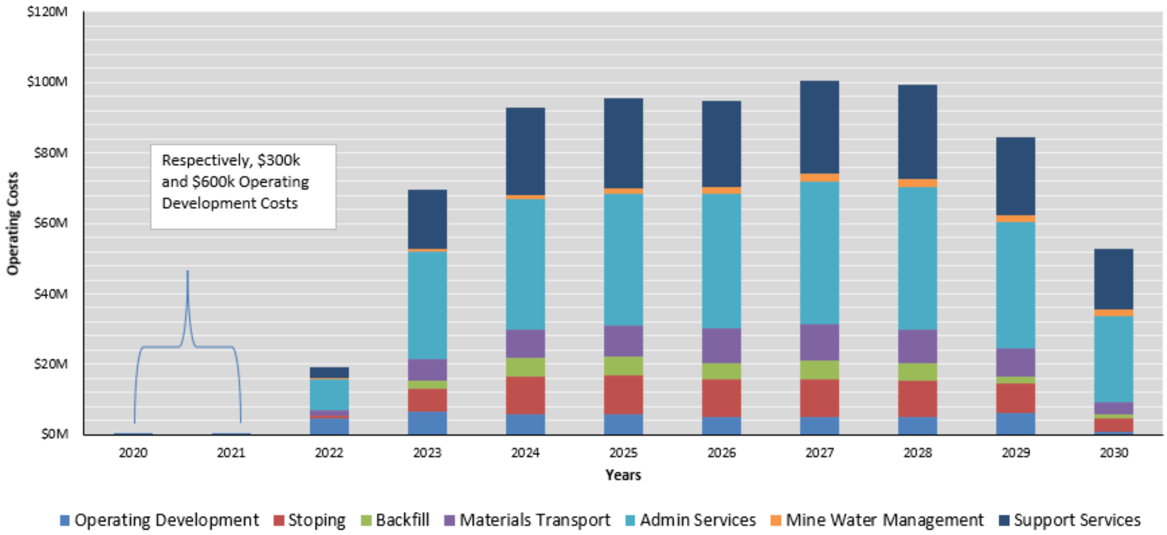


Figure 6-5: BP20 Underground Mine Operating Costs over LOM

The *Processing* and *Refining* related costs are dependent on the plant configuration. These modifications will be presented in figure 6-7 and are restricted to the processing of only underground material. The associated operating costs for the configuration is presented in table 6-6.

Table 6-6: Underground Processing and Refining Operating Cost Structure

Underground	Variable [$/t ore]	Fixed [M$/year]	Product Related [$/oz Au]
Processing	7.36	12.9	✗
Refining	✗	✗	2.51

The costs presented in table 6-6 are based on the current assumptions related to the option 1b from Logan (2019).

6-3-2 Capital Costs

Open-pit

Table 6-7: Open-pit Capital Costs

Open-pit	Sustaining capital		Initial/Upgrade capital
	Variable [$/t ore]	Fixed [M $/year]	[M $]
Mining	3.07	✗	(*Incremental OP ore*) – Table 6-9
Processing	1.57	4.6	(*Incremental OP ore*) – Table 6-9

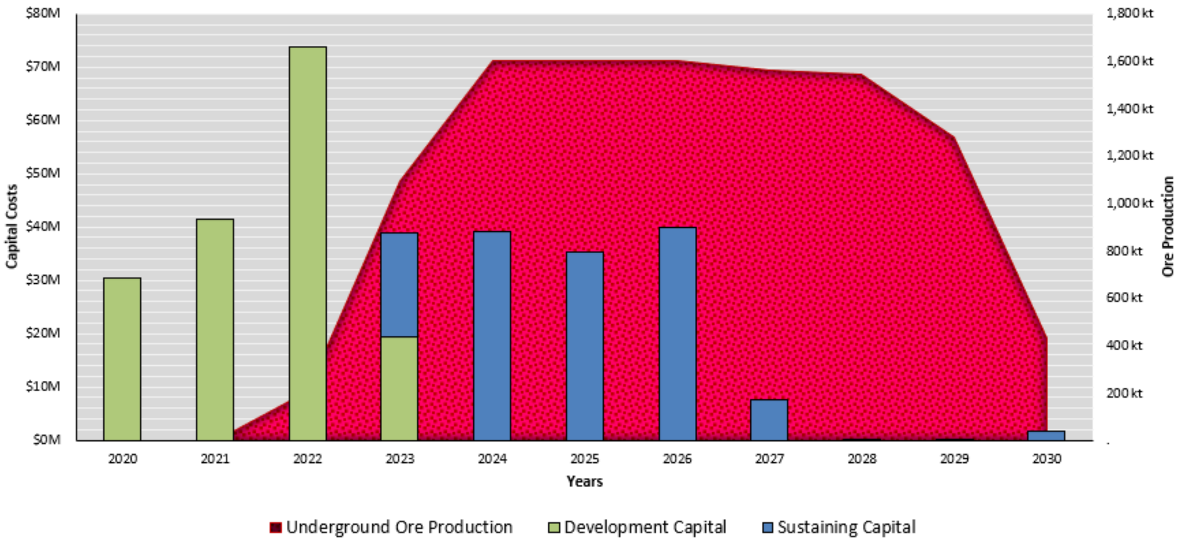


Figure 6-6: BP20 Underground Mine Capital Costs over LOM from April, 2019

Mining related comments and assumptions,

- mining related sustaining capital is based on 0.85 $/t material mined (sustaining capital, Akyem Cut-off Grade sheet, 2018).
- Mining related sustaining capital is provided as a variable costs with no translation factor for fixed costs. Whereas, processing related capital costs are provided both in variable and fixed.
- Initial/upgrade capital expenditures are not required for the current open-pit mining- or processing-related operations; these are considered as sunk costs.

Processing related comments and assumptions,

- Costs are based on 20% of *operating costs.*

Underground

Figure 6-6 presents the capital expenditures associated to the underground mining operations. The figure shows similar trends as figure 6-5, where the ramp-up phases requires significant capital expenditures ($165.2 M) from 2020 until 2023. After full production, the amount of sustaining capital ($143.3 M) can be related to the amount of total yearly production together with the year of closing under consideration.

Capital expenditures related to processing activities of underground *processing* ore, through a modified plant, are presented in the table 6-6.

Table 6-8: Underground Processing Related Sustaining Capital Costs

Underground	Sustaining Variable [$/t ore]	capital Fixed [M $/year]	Initial/Upgrade capital [M $]
Mining	(*Figure 6-6*)	(*Figure 6-6*)	Table 6-9
Processing	2.60	1.47	Table 6-9

The costs presented in table 6-8 are based on the application of a multiplication factor of 20 % to the *operating costs*.

Capital Expenditure Trade-off

The potential transition from open-pit to underground operations will comprehend major capital expenditures. These expenditures will be incurred for either a full transition or a concurrent open-pit and underground operation. Comprehensibly, an expenditure is associated to the plant modification for processing only underground ore. On the other side, the potential addition of incremental open-pit ore will comprehend a major expenditure related to the required expansion of the TSF. These scenarios are presented in the table 6-9.

Table 6-9: Capital Expenditure Trade-off

Trade-off	Plant Modification [M$]	TSF Expansion [M$]	Incremental TSF Expansion [$/t]*
OP + UG (BP20)	✓	✓	✗
Incremental OP	✗	✗	✓
OP + UG (BP20) + Incremental OP	✗	✓	✓
Costs	12	30	3.5*

The data presented in table 6-9 is based on information provided within the works of Logan (2019) and the TSF Option Analysis from July, 2019. The costs associated to the modification of the processing plant are predominately related to the modifications illustrated in figure 6-7. Where, the red-shaded boxes indicate the removal of facilities:

1. Oxide material feed system
2. Ball mill
3. Five of eleven CIL tanks

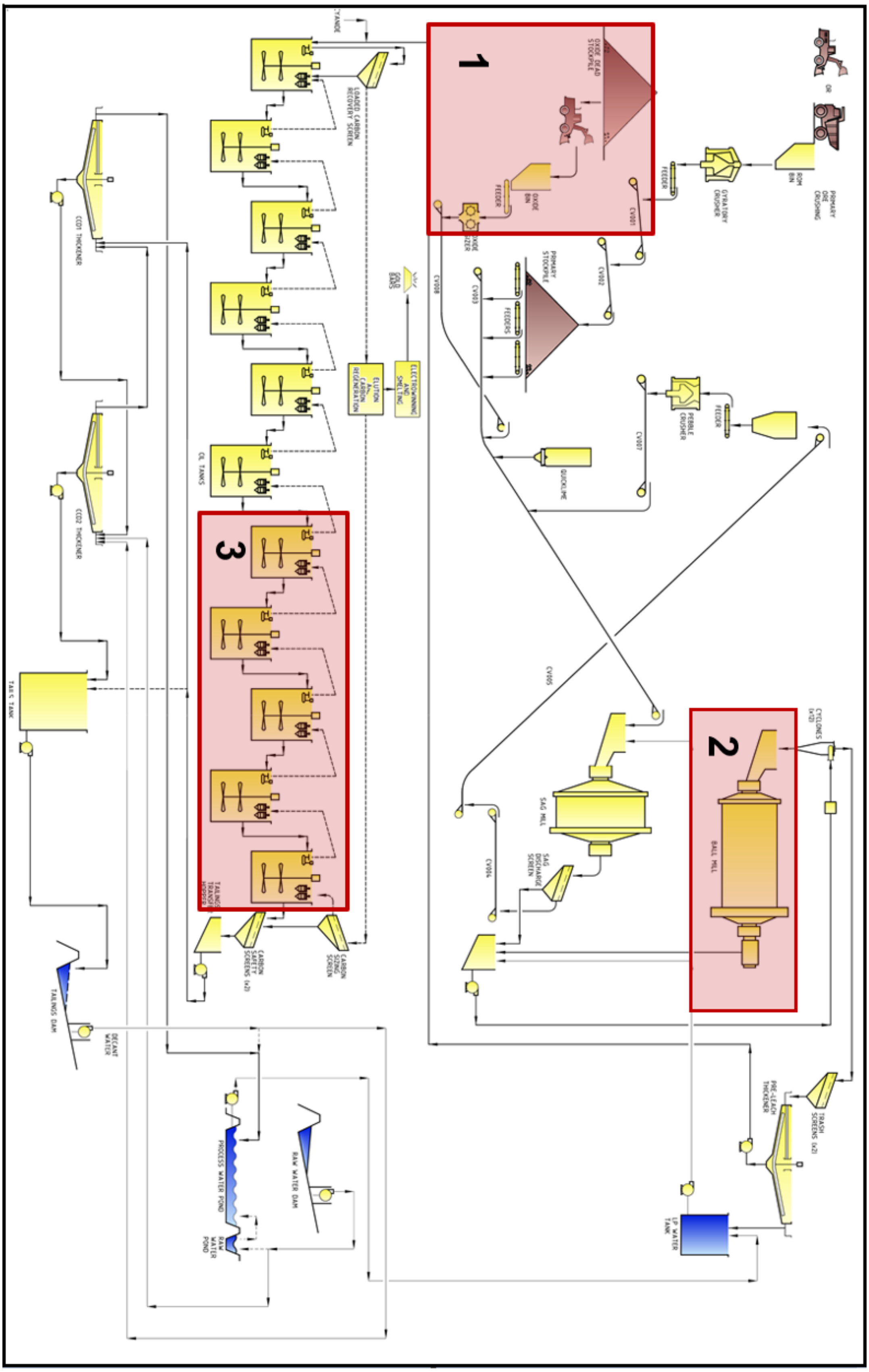


Figure 6-7: Potential Plant Modification Plans

Several options have been evaluated regarding additional TSF capacity. This thesis considers the costs related to the underground expansion according to BP20. Hence, the current OP + UG BP20 scenario is associated to an additional 10 Mt TSF capacity. Where, the 10 Mt is originated from the BP20 underground ore.

Incremental OP ore was considered on top of current OP + UG (BP20) physicals. This scenario dismisses the plant modification costs and instead adds TSF related capital for the incremental OP ore. The incremental capital expenditure is dependent on the amount of open-pit ore associated to the throughput option.

6-3-3 G&A Costs

Obtained costs related to the G&A activities are presented in table 6-10. The open-pit variable costs are based on the 0.36 $/t material mined - from the Akyem Cut-off grade sheet, 2018. Open-pit fixed costs are based on calculated costs from regional guidance.

The translation to underground G&A costs is based on the following percentages:

- Mining Fixed = 21% of open-pit
- Processing Fixed = 39% of open-pit

These assumptions are based on the change of yearly production rate. The processing multiplication factor is modified with another 150%, assuming that the main layout remains similar.

Table 6-10: Open-pit G&A Cost Structure

Open-pit	Variable [$/t ore]	Fixed [M$/year]	**Underground**	Variable [$/t ore]	Fixed [M$/year]
Mining	1.30	5.50		1.89	1.14
Processing	✗	20.0		✗	7.75

Figure 6-8: Previous 30 Years Gold Price Overview

6-3-4 Gold Prices and Royalty Rates

The gold prices are the major driving factors for the results related to this thesis. Predicting these gold prices will comprehend a high risk margin due to the volatility throughout the past. An overview of the gold prices throughout the last 30 years is provided in figure 6-8.

The utilized royalty rates are dependent on the gold price and can be consulted in the table 6-11.

Table 6-11: Government Royalty Rates

Gold Price From [$/oz]	Gold Price To [$/oz]	Royalty [%]
0	1299	3.6
1300	1449	4.1
1450	2299	4.6

6-3-5 Results display

The results will be provided according to a set of predefined cost structure. The cost structure with their implemented costs can be consulted below:

Project Capital

- Underground Capital Development costs + Mill Modification costs + TSF Expansion costs + G&A costs

Total Sustaining Capital

- Mining + Processing Sustaining Capital, for all occurring operations

Total Operating Costs

- Mining + Processing + Refining Operating Costs, for all occurring operations

Net Revenue

- Gross Revenue - Royalty

Where, Gross Revenue = Produced Gold Ounces * Gold Price per Ounce

Chapter 7

Results

7-1 Introduction

This chapter will present the results related to the approach explained in the methodology providing answers to the research questions.

The incremental open-pit head grade calculation forms the core for the answers related to the optimization study. Additionally, the results of changes of different decision variables remain essential for the analyses and will be provided through a sensitivity analysis.

Prior to the open-pit head grade determination, the base case will be presented. This will allow for a NPV indication to serve as a reference point for the optimization.

To conclude on the structure for this chapter; results will be provided in the following order:

1. Base Case - according to current production plans with plant modification
2. Head Grade Case - according to open-pit head grade calculation

7-2 Base Case

7-2-1 Starting Stockpile Inventory

The first results consist of a blending product of the following two input streams:

1. The starting stockpile inventory of April, 2019, presented in table 6-2
2. Planned physicals for residual months of 2019, presented in figure 6-2.

The result shows the starting stockpile inventory as of December, 2019. These physicals are required for the actual major blending procedure in order to determine the mines productivity. The results can be consulted in table 7-1.

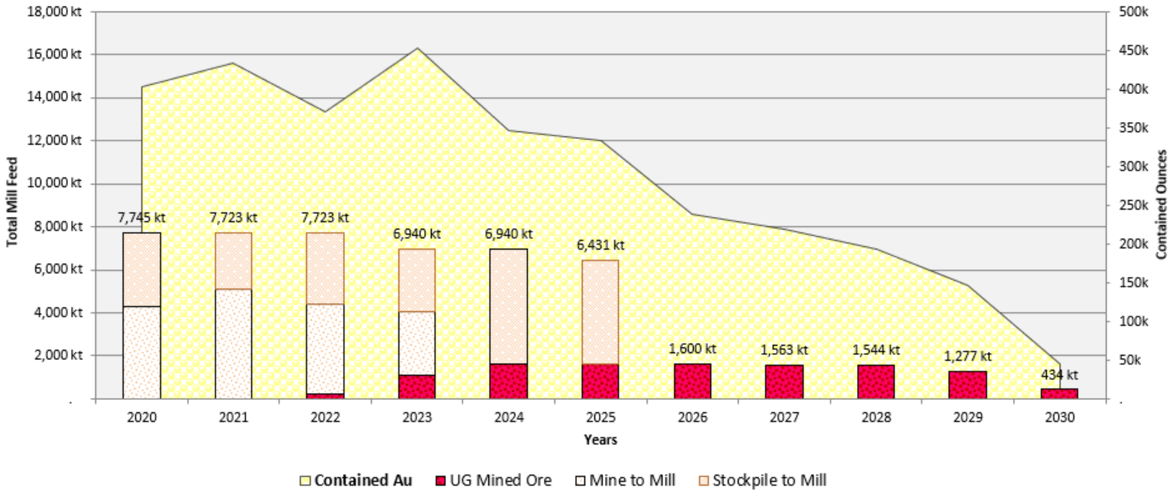


Figure 7-1: Base Case Mill Feed

Table 7-1: Starting Stockpile Inventory, December, 2019

Groups	Ore [kt]	Average Grade [g/t]	Contained Au Ounces [oz]
PMG	7,011	1.04	234,415
PLG	9,433	0.72	218,369
PSG	1,189	0.47	17,696

7-2-2 Base Case Production

The other two input flows for the blending process consist of the mine-to-mill streams. The streams originate from open-pit and underground mining activities and were given in figures 6-1 and 6-3, respectively. Hence, the combination of these physicals, together with the stockpile inventory of each preceding year, is utilized for the blending process. The resulting base case production numbers are presented in figure 7-1.

Figure 7-1 shows that that the open-pit mining activities will cease in 2023. Whereas, the stockpiles can still deliver ore until 2025. This implies that the plant modification must occur in 2026; once all valuable open-pit material is processed.

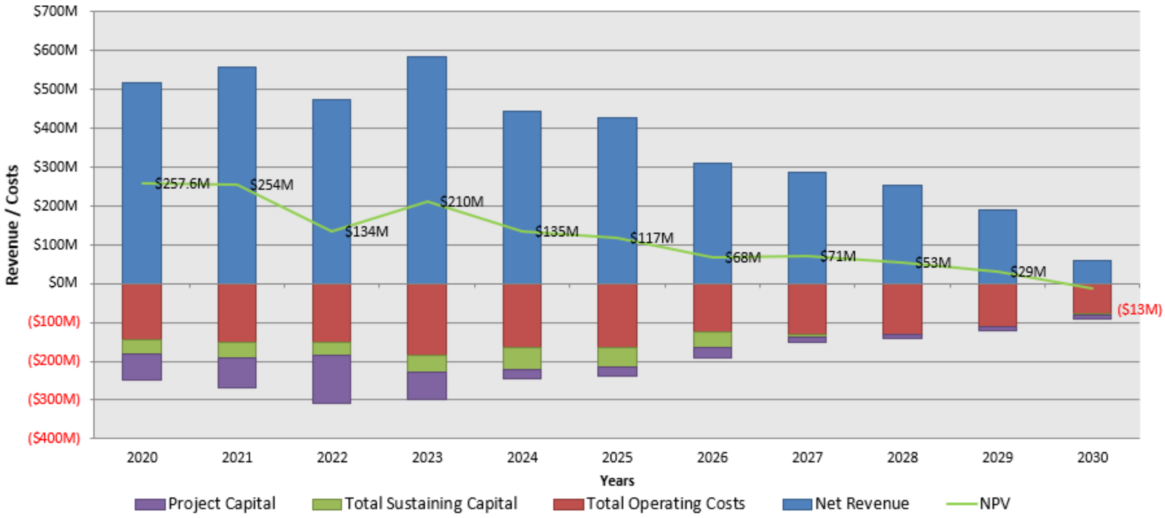


Figure 7-2: Base Case Cash Flow

7-2-3 Base Case Cash Flow

The physicals presented in figure 7-1 will determine the cash inflows and outflows. A set of key decision variables that are used for the base case calculation are presented below:

- Plant Modification in 2026
- Option 1b - Plant Modification
- TSF Expansion: BP20 UG - 10 Mt of ore (no incremental)
- Gold Price: 1500 $/oz
- Royalty Rate: 4.6 %
- Discount Rate: 9%

The implementation of these decision variables in combination with the physicals (figure 7-1) and costs configuration from chapter 6, form the base case cash flow. These can be consulted in figure 7-2.

The figure (7-2) shows the similar trends presented previously for the individual evaluation of underground operations. A slight increase in operating costs during the simultaneous open-pit and underground operations in 2022 and 2023. The increase in sustaining capital, after open-pit mining activities are completed, is related to the high underground productivity including the open-pit stockpile blending process.

The operating costs show a peculiar result; despite reaching the end of LOM with a major decrease in production, the operating costs remain significant. Stoping and development operating costs decrease, however the current calculated administrative costs remain very high. This explains the negative balance between cash inflow and outflow including the negative NPV.

	years	2026	2027	2028	2029	2030
UG Only	Costs	$189.4M	$162.2M	$153.4M	$135.9M	$96.5M
	Net Revenue	$310.1M	$286.2M	$251.6M	$189.5M	$58.0M
	Profit	$120.1M	$123.4M	$97.7M	$53.3M	($38.6M)
UG Mining with OP Configuration	Costs	$165.0M	$137.6M	$128.6M	$109.0M	$63.3M
	Net Revenue	$310.1M	$286.2M	$251.6M	$189.5M	$58.0M
	Profit	$144.5M	$148.1M	$122.5M	$80.2M	($5.5M)
Incremental OP Feed	Costs	$182.7M	$183.9M	$184.5M	$192.6M	$218.3M
	Net Revenue	$150.9M	$151.7M	$152.1M	$157.9M	$176.4M
	Profit	($24.6M)	($24.9M)	($25.0M)	($27.1M)	($33.4M)
	Head Grade	**0.586**	**0.585**	**0.584**	**0.578**	**0.562**

Figure 7-3: Example of Scenario Comparison for Head Grade Calculation with 6.9 Mt Throughput

7-3 Head Grade Case

The results for the head grade calculation account for the period after the plant modification has occurred. Previous base case results have proven this period to be in the year 2026. Despite the plant modification costs having to be accounted for prior to this period, these costs remain to be considered for the scenario comparison from years 2026 until 2030.
The major capital expenditure trade-off that is accounted for will be the incremental TSF capacity against the plant modification costs. These costs are complemented with the operating costs related to the scenarios explained in the methodology chapter (5).

In essence the results follow from the inputs which are implemented into formulas in the model. These calculations are based on the formulas explained in chapter 5 and visualized in figure 5-3. Dependent on the set of parameters, these results are provided as shown in figure 7-3. The figure serves as an example for the minimum required open-pit head grade with a total mill feed of 6.9 Mt per year.

The input for the results presented in figure 7-3 can be consulted in figure 7-4. The total costs per ore tonne is used to calculate the total costs for each scenario. Hence, these costs are also controlled by the given underground mine production (figure 6-3). The costs related to the *Incremental OP Feed* are subsequently dependent on this underground mine productivity, subtracted from the chosen total yearly throughput option.

The *Incremental OP Feed* scenario is calculated as a result of the upper two scenarios. The profit for this scenario is not equal to the sum of these other two scenarios due to an additional cost implementation. These costs are related to the refining costs.

The refining costs are driven by the amount of gold being produced. Therefore, this amount is subtracted from the total profit being made instead of applying it to the total cost per tonne of ore. This actual gold production is based on the recovery which is related to the head grade (equation 5-6). For this reason an assumption has been made for the head grade. The assumption is that the incremental open-pit head grade has a gold quantity of 0.54 g/t.

The trade-off regarding the capital expenditure associated to the plant modification and incremental TSF expansion are considered for each year. The *UG Only* scenario includes the cost structure related to the plant modification. Whereas, the *Incremental OP Feed* implements additional TSF costs driven by the ore tonnes. The remaining TSF costs associated to the BP20 underground can be neglected for the head grade calculation since they will be incurred for both scenarios regardless of the chosen option.

Throughput options and preliminary head grades

Six different throughput options were evaluated in order to determine the relationship between production rate, head grade and recoverable gold ounces. These throughput options are potential options to implement. They result from a 10 % difference of the current maximum throughput capacity. The plant will remain intact, hence the the fixed costs remain equal for all options. The only difference aspect related to these options are the variable costs. The throughput options are as followed:

1. Yearly Throughput of 6.9 Mt
2. Yearly Throughput of 6.3 Mt
3. Yearly Throughput of 5.4 Mt
4. Yearly Throughput of 4.9 Mt
5. Yearly Throughput of 4.2 Mt
6. Yearly Throughput of 3.5 Mt

The most relevant relationships between the throughput options and recoverable gold ounces are presented in figure 7-5. These graphs represent the results associated to the highest and lowest throughput option. Besides the throughput option, the other significant driving factor is the gold price which in these cases is set at $1,500 per ounce.

Similar trend have been observed of which both the highest and lowest throughput options are presented in figure 7-5. The figure shows these trends, where a decrease in throughput rate is associated with an expected increase in head grade. This can be seen on the right-vertical axis with changes in the head grade range.

Figure 7-6 shows the trend where for five of these potential throughput options. The production years are plotted for each throughput option with their cohering head grade on the plot axis. It is evident that the required open-pit head grade will be lower as a result of an increase in production rate for all options. The minimum required head grade is related to the highest potential throughput option. This head grade is currently 0.56 [g/t] which is 0.02 [g/t] above the current lowest operational cut-off grade, that was set at 0.54 [g/t].

Any increase in gold price should result in a lower required open-pit head grade. Results for a change in gold price from $1,500 to $1,600 per ounce, are provided in figure 7-7. The results show similar trends when compared with the $1,500 per ounce head grade evaluation. However, an improvement might be achievable when a head grade of roughly 0.53 [g/t] is obtainable with a throughput of 6.5 Mt in year 2030. This throughput is considerable difficult to realize. However, a reduction in throughput will result in a open-pit head grade greater than the current cut-off grade of 0.54 [g/t]. Hence, these options remain to be neglected.

	years	2026	2027	2028	2029	2030
	UG Only					
Mining	Opex	$94.9M	$100.6M	$99.4M	$84.5M	$52.7M
	Sustaining Capital	$39.9M	$7.5M	$0.1M	$0.2M	$1.6M
	G&A - Mining	$4.2M	$4.1M	$4.1M	$3.5M	$2.0M
	Total	$139.0M	$112.1M	$103.5M	$88.3M	$56.3M
	$/t ore	$86.87	$71.74	$67.03	$69.13	$129.76
Processing	Opex	$24.9M	$24.6M	$24.5M	$22.5M	$16.4M
	Sustaining Capital	$5.0M	$4.9M	$4.9M	$4.5M	$3.3M
	G&A - Processing	$7.7M	$7.7M	$7.7M	$7.7M	$7.7M
	Plant Modification	$12.8M	$12.8M	$12.8M	$12.8M	$12.8M
	Total	$50.4M	$50.1M	$49.9M	$47.6M	$40.2M
	$/t ore	$31.52	$32.06	$32.35	$37.28	$92.69
Refining	Total - Product Related (Au Oz)	$0.5M	$0.5M	$0.4M	$0.3M	$0.1M
	Total	$118.39	$103.80	$99.37	$106.41	$222.45

		2026	2027	2028	2029	2030
	UG Mining with OP Configuration					
Mining	Opex	$94.9M	$100.6M	$99.4M	$84.5M	$52.7M
	Sustaining Capital	$39.9M	$7.5M	$0.1M	$0.2M	$1.6M
	G&A - Mining	$4.2M	$4.1M	$4.1M	$3.5M	$2.0M
	Total	$139.0M	$112.1M	$103.5M	$88.3M	$56.3M
	$/t ore	$86.87	$71.74	$67.03	$69.13	$129.76
Processing	Opex	$77.5M	$77.5M	$77.5M	$77.5M	$77.5M
	Sustaining Capital	$15.5M	$15.5M	$15.5M	$15.5M	$15.5M
	G&A - Processing	$20.0M	$20.0M	$20.0M	$20.0M	$20.0M
	Total	$113.0M	$113.0M	$113.0M	$113.0M	$113.0M
	$/t	$16.28	$16.28	$16.28	$16.28	$16.28
Refining	Total - Product Related (Au Oz)	$0.5M	$0.5M	$0.4M	$0.3M	$0.1M
	Total	$103.15	$88.02	$83.31	$85.41	$146.04

		2026	2027	2028	2029	2030
	Incremental OP Feed					
Mining	Opex	$63.0M	$63.0M	$63.0M	$63.0M	$63.0M
	Sustaining Capital	$21.3M	$21.3M	$21.3M	$21.3M	$21.3M
	G&A - Mining	$14.5M	$14.5M	$14.5M	$14.5M	$14.5M
	Total	$98.8M	$98.8M	$98.8M	$98.8M	$98.8M
	$/t ore	$14.23	$14.23	$14.23	$14.23	$14.23
Processing	Opex	$77.5M	$77.5M	$77.5M	$77.5M	$77.5M
	Sustaining Capital	$15.5M	$15.5M	$15.5M	$15.5M	$15.5M
	G&A - Processing	$20.0M	$20.0M	$20.0M	$20.0M	$20.0M
	Total	$113.0M	$113.0M	$113.0M	$113.0M	$113.0M
	$/t ore	$16.28	$16.28	$16.28	$16.28	$16.28
Refining	Total - Product Related (Au Oz)	$0.2M	$0.2M	$0.2M	$0.2M	$0.3M
TSF	$/t ore	$3.71	$3.68	$3.67	$3.50	$3.04
	Total	$34.22	$34.20	$34.18	$34.01	$33.56

Figure 7-4: Cost Structure Related to Scenario Comparison for Head Grade Calculation

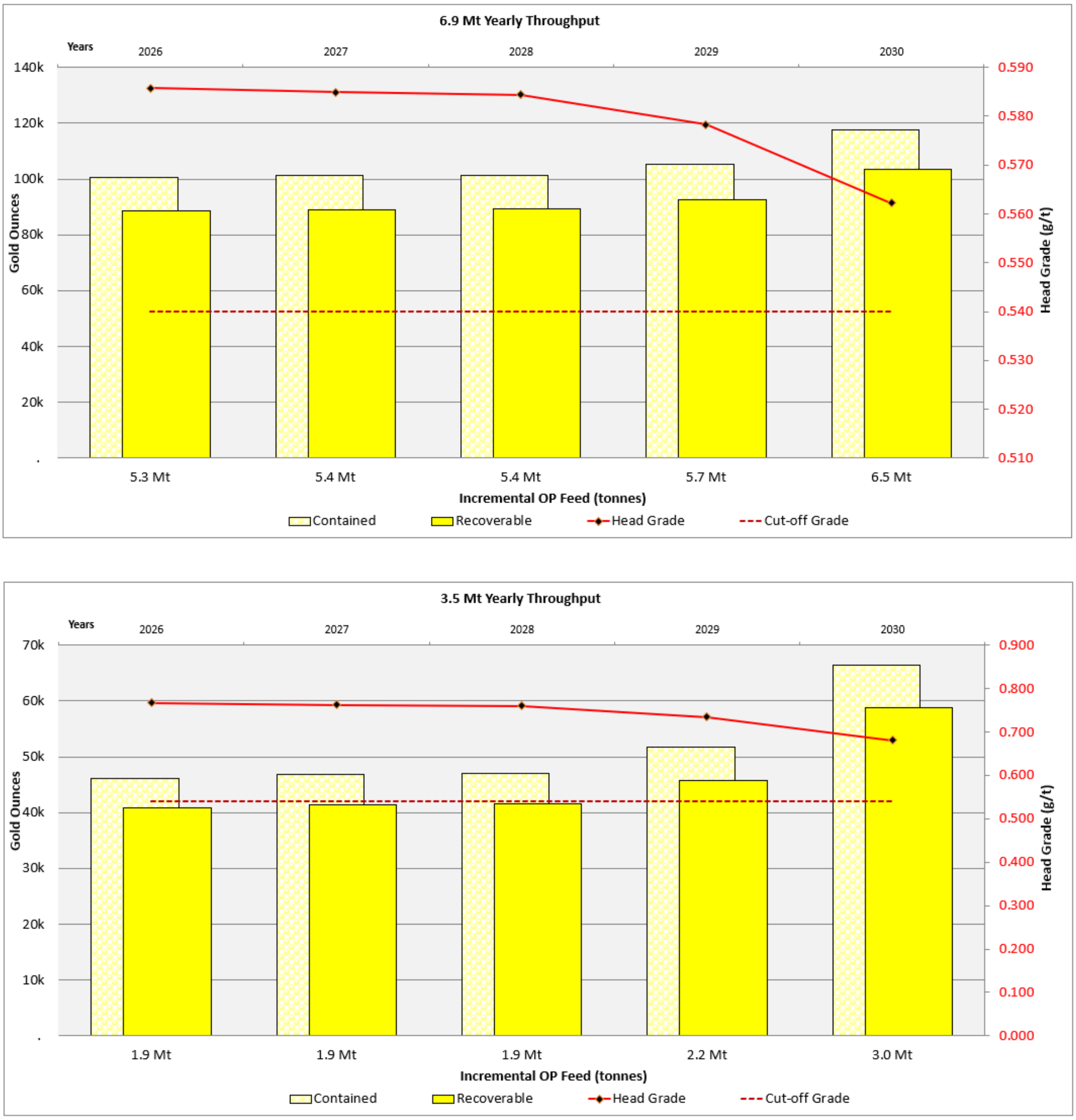


Figure 7-5: Head Grade Calculation Results Based on Throughput Options with a Gold Price of $1,500 per ounce

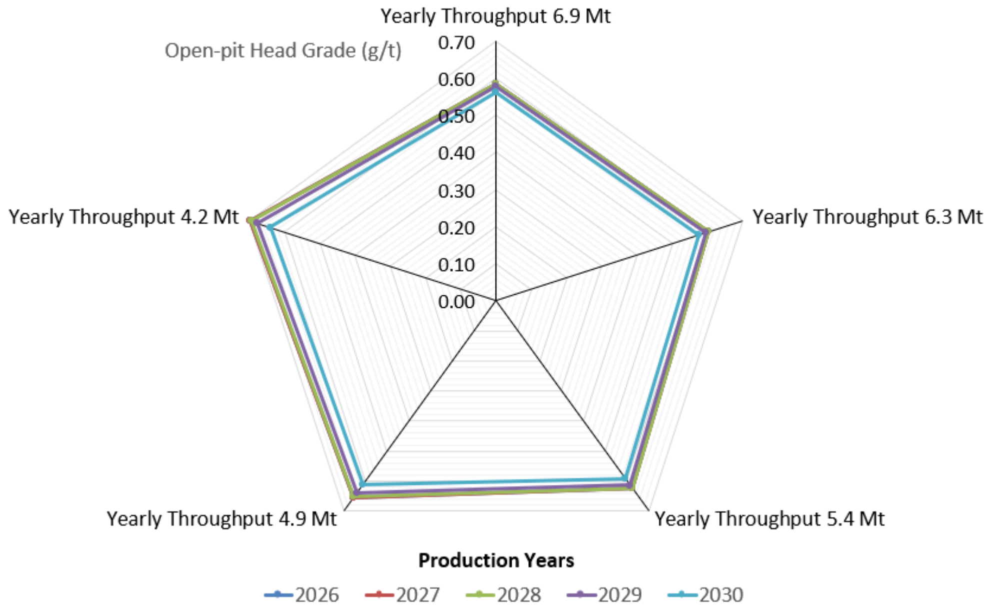


Figure 7-6: Illustration of Trend in Increasing Head Grade for Decreasing Throughput Option

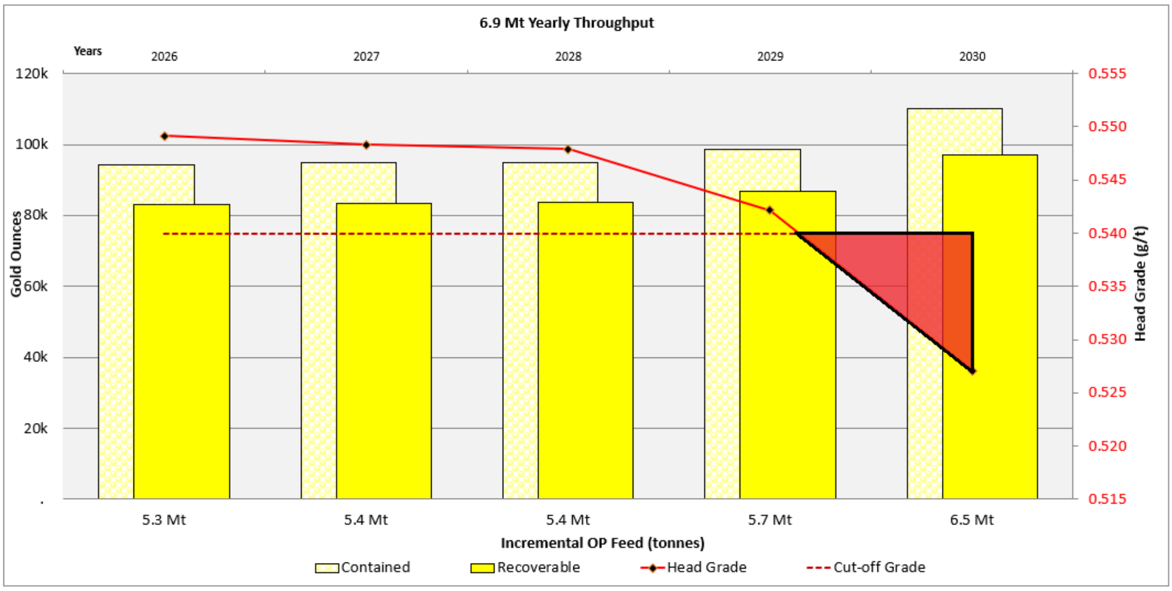
The results for the remaining throughput options are provided in a table format and can be consulted in figure 7-8. The figure includes results for more extreme gold price cases. The left table, including results for the $1,600 dollar per ounce gold price, shows the same results as shown in figure 7-7 for the 6.9 Mt and 3.5 Mt throughput options. The other two tables show results for gold prices of $1,700 and $,1800 per ounce respectively. The figure shows promising results for these cases without considering the probability of reaching such gold prices.

7-3-1 Considering the Time Value of Money

Previously provided results have shown possibilities regarding a set of different gold prices. However, the results neglect the time value of money. The effect of inflation must be accounted for and can be implemented by applying a discount rate to the net cash flow. Due to this principle of inflation, the required open-pit head grade is likely to increase. The results for the application of a discount rate of 9% can be consulted in figure 7-9.

The results show that under the current cost configuration, only an open-pit head grade might be achievable with a gold price of $1,800 per ounce. This accounts for all years under consideration for a throughput option of 6.9 Mt of ore per year. The 6.3 Mt throughput option is indicated as a transitional zone due to its coherence with the current cut-off grade. Hence, those years are considered as unlikely to add any value when discarding the plant modification. The only potential improvement for this throughput option can be seen for the year 2030 with a open-pit head grade of 0.53 [g/t].

The total incremental open-pit feed must also be realized in order to comply with the head grade calculation. Hence, the options highlighted with green in figure 7-9, must be checked



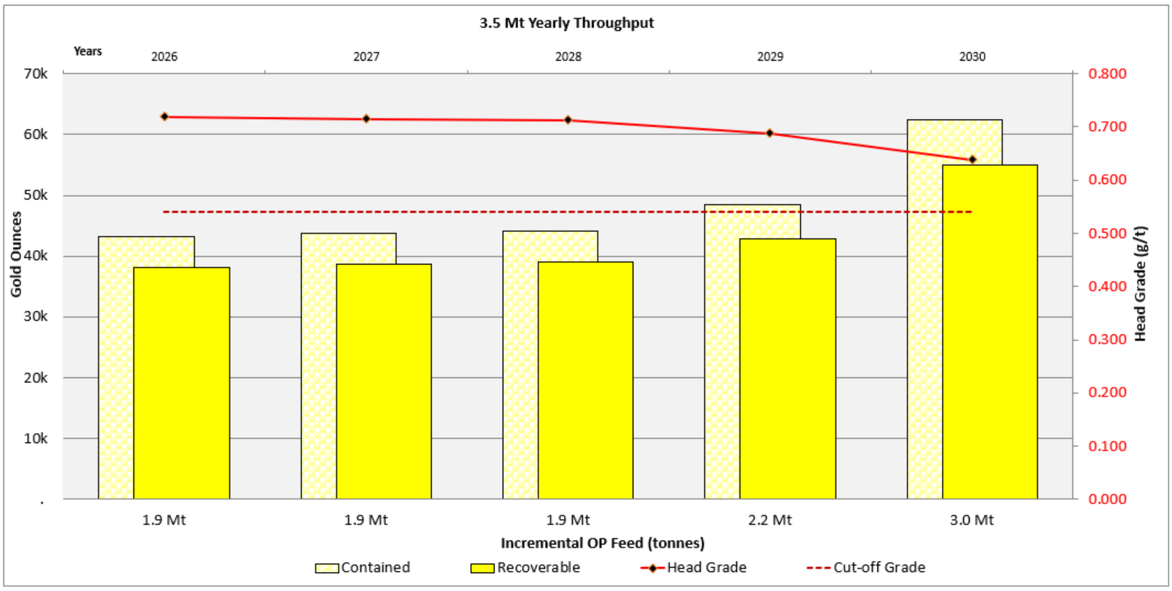


Figure 7-7: Head Grade Calculation Results Based on Throughput Options with a Gold Price of $,1600 per ounce

	Gold Price: $1,600 per Au Oz					Gold Price: $1,700 per Au Oz					Gold Price: $1,800 per Au Oz				
Throughput Options	**2026**	**2027**	**2028**	**2029**	**2030**	**2026**	**2027**	**2028**	**2029**	**2030**	**2026**	**2027**	**2028**	**2029**	**2030**
6.9 Mt per year	0.55	0.55	0.55	0.54	0.53	0.52	0.52	0.52	0.51	0.50	0.49	0.49	0.49	0.48	0.47
6.3 Mt per year	0.57	0.56	0.56	0.56	0.54	0.53	0.53	0.53	0.52	0.51	0.50	0.50	0.50	0.50	0.48
5.6 Mt per year	0.59	0.59	0.58	0.58	0.55	0.55	0.55	0.55	0.54	0.52	0.52	0.52	0.52	0.51	0.49
4.9 Mt per year	0.61	0.61	0.61	0.60	0.57	0.58	0.58	0.58	0.57	0.54	0.55	0.54	0.54	0.53	0.51
4.2 Mt per year	0.66	0.65	0.65	0.64	0.60	0.62	0.61	0.61	0.60	0.57	0.58	0.58	0.58	0.57	0.53
3.5 Mt per year	0.72	0.71	0.71	0.69	0.64	0.68	0.67	0.67	0.65	0.60	0.64	0.64	0.63	0.61	0.57

= Above current C.O.G = Transitional = Below current C.O.G

Figure 7-8: Open-pit Head Grades Based on Throughput Options and Potential Gold Prices for Non-discounted Cash Flows

Discounted Net Cash Flow

Gold Price: **$1,600** per Au Oz

Throughput Options	2026	2027	2028	2029	2030
6.9 Mt per year	0.59	0.59	0.59	0.59	0.58
6.3 Mt per year	0.61	0.61	0.61	0.61	0.60
5.6 Mt per year	0.63	0.63	0.64	0.64	0.62
4.9 Mt per year	0.66	0.67	0.67	0.67	0.65
4.2 Mt per year	0.71	0.71	0.72	0.71	0.69
3.5 Mt per year	0.78	0.78	0.78	0.77	0.75

Gold Price: **$1,700** per Au Oz

Throughput Options	2026	2027	2028	2029	2030
6.9 Mt per year	0.55	0.55	0.56	0.56	0.55
6.3 Mt per year	0.57	0.57	0.58	0.57	0.57
5.6 Mt per year	0.59	0.60	0.60	0.60	0.59
4.9 Mt per year	0.62	0.63	0.63	0.63	0.62
4.2 Mt per year	0.67	0.67	0.67	0.67	0.65
3.5 Mt per year	0.73	0.73	0.74	0.73	0.70

Gold Price: **$1,800** per Au Oz

Throughput Options	2026	2027	2028	2029	2030
6.9 Mt per year	0.52	0.52	0.53	0.53	0.52
6.3 Mt per year	0.54	0.54	0.54	0.54	0.53
5.6 Mt per year	0.56	0.56	0.57	0.56	0.56
4.9 Mt per year	0.59	0.59	0.60	0.59	0.58
4.2 Mt per year	0.63	0.63	0.64	0.63	0.62
3.5 Mt per year	0.69	0.69	0.70	0.69	0.67

= Above current C.O.G = Transitional = Below current C.O.G

Figure 7-9: Open-pit Head Grades Based on Throughput Options and Potential Gold Prices for Discounted Cash Flows

whether the productivity is reasonable. The cohering open-pit ore feed must account for the quantities provided in table 7-2.

Table 7-2: Required Incremental Ore Feed per Year related to gold price of 1800 $/oz

Years	2026	2027	2028	2029	2030
Head Grade [g/t]	0.52	0.52	0.53	0.53	0.52
Associated Feed [Mt]	5.34	5.38	5.40	5.66	6.51
Recoverable Ounces [Koz]	89.4	90.5	91.3	95.7	108.5

The incremental ore feed and its head grade relate to an amount of gold as product. This quantity of gold will be the driving factor behind the value generation regarding this option. Therefore, the option will prove profitable if the indicated recoverable ounces can be realized within those years. Any head grade above the required grade with a cohering throughput will lead to such increasing value.

The grade-tonnage curve provided in chapter 6, figure 6-4, can be consulted for an indication of the available ore in the pit. The required recoverable ounces provided in table 7-2 can be realized through implementation of a cut-off grade of 0.27 [g/t]. However, this amount of recoverable gold ounces is associated to total incremental throughput of 6.9 Mt of ore and equivalent to a head grade of 0.41 [g/t]. This amount surpasses the maximum throughput capacity by 1.6 Mt of ore.

When aiming for a permissible incremental throughput of 5.3 Mt, the associated recoverable ounces available in the pit is 75.1 kOz. This amount implies a deficiency of 14.3 kOz of recoverable ounces. Hence, the option is not considered as possible according to current reserve estimations. These results can be consulted in the figure 7-10. The results are presented with green and red words, indicating whether the result is possible (green) or vice versa (red).

7-3-2 Recovery, Throughput rate and Head Grades

Besides the time value of money, the preliminary minimum required head grade must account for the recovery deficiency from the processing plant. Provided required head grades increase

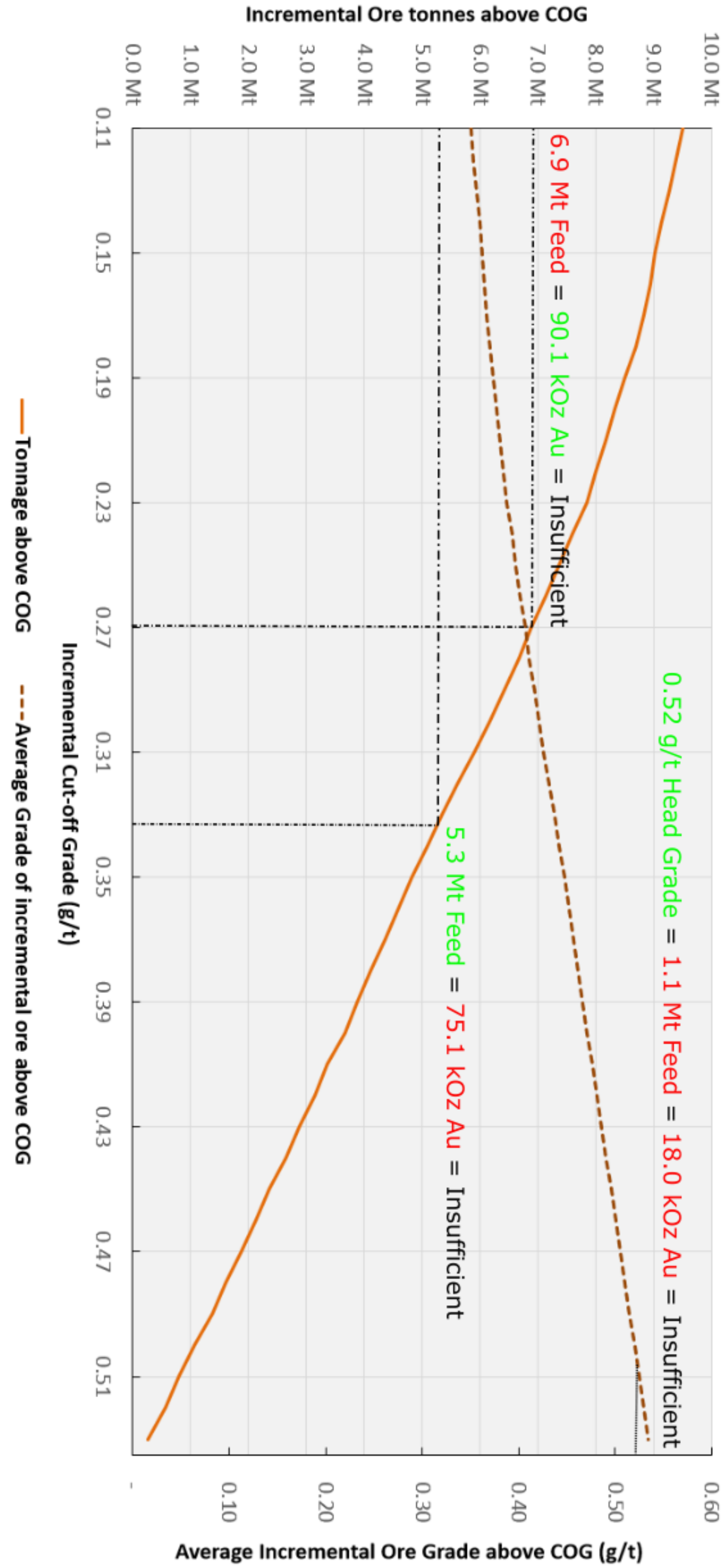


Figure 7-10: Grade-Tonnage Curve for Incremental Open-pit Head Grade Results Related to Gold Price of 1,800 $/oz

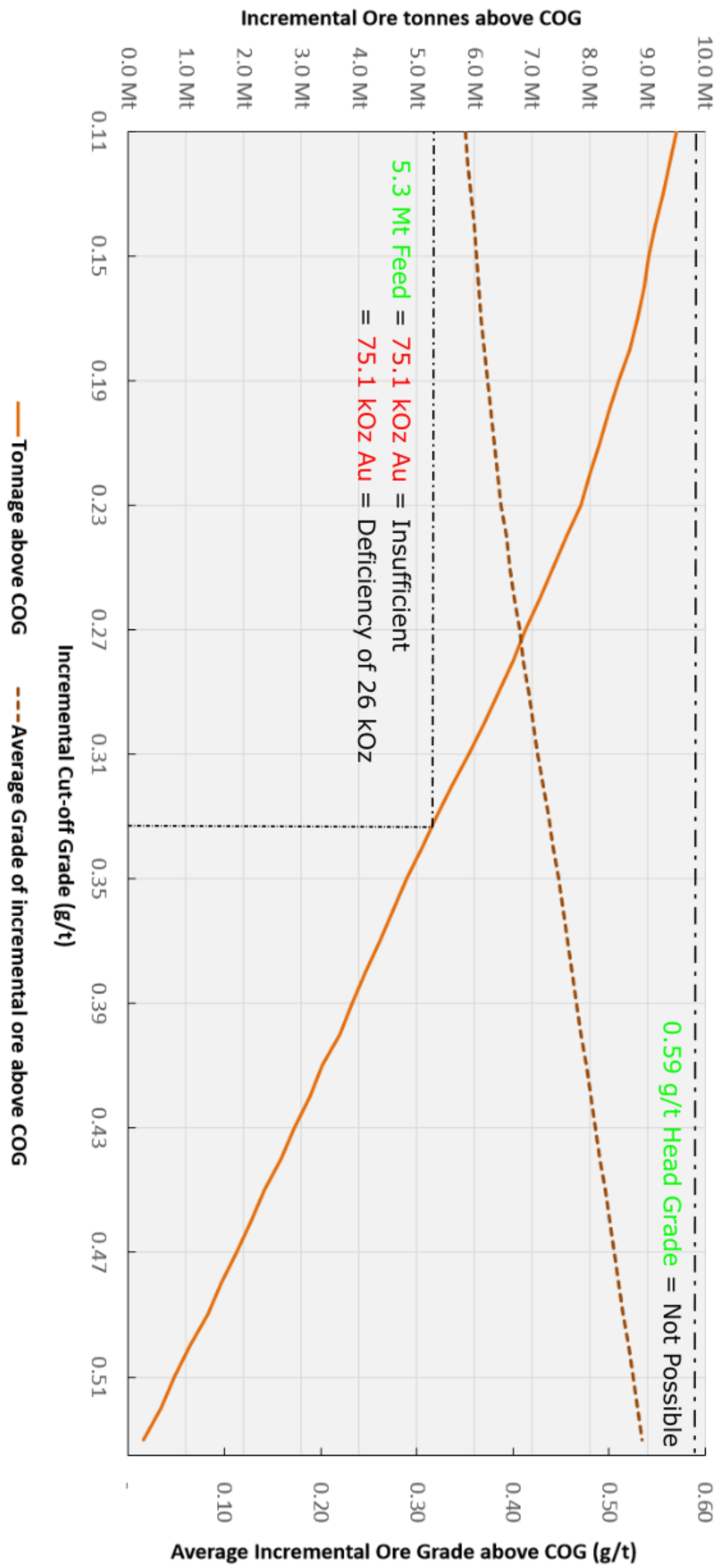


Figure 7-11: Grade-Tonnage Curve for Revised Incremental Open-pit Head Grade Results Related to Gold Price of 1,800 $/oz

as a result of this aspect. The iterative process responsible for these results implement the relationships between the throughput rates, head grade and recoveries.

Results will be given related to the lowest indicated head grade of 0.52 [g/t], associated to the scenario of a gold price of $1,800 per ounce. These results cohere with a total incremental throughput of 5.3 Mt of ore. The amount of required recoverable ounces has been calculated to be 89,000. These amount of recoverable ounces will be obtained through a head grade of 0.59 [g/t]. In turn, this head grade will effect the recovery, which is calculated to be 88.1 %. Thus, the combination of the recovery, throughput and head grade, the total amount of contained ounces must be 101,000.

An additional grade-tonnage curve has been provided to visualize the results related to the revised required head grade. These results can be consulted in figure 7-11. The figure shows the impossibility of obtaining such gold ounces from remaining open-pit gold reserves.

7-3-3 Sensitivity Analysis

The previously provided results were based on the assumptions stated in chapter 6. These assumptions have a probability of slightly deviating from the actual costs. Therefore, a sensitivity analysis has been conducted. The goal of this sensitivity analysis is to check whether a possible outcome is obtainable through a potential cost deviation. The gold price fluctuations have already been evaluated but remain to be considered for the changes in costs parameters. The results will be considering the time value of money thus the discounted net cash flow is implemented.

Throughput 6.9	-15%	-10%	-5%	0%	5%	10%	15%
$1,800	0.55	0.56	0.57	0.59	0.61	0.62	0.64
$1,700	0.6	0.6	0.61	0.63	0.64	0.66	0.67
$1,600	0.62	0.64	0.65	0.67	0.68	0.69	0.71
$1,500	0.66	0.67	0.69	0.71	0.73	0.74	0.76

Changing all chosen costs factors equally and simutaneously

Figure 7-12: Minimum Required Open-pit Head Grades for Simultaneous Equal Changes in Chosen Cost Parameters

Throughput 6.9	-15%	-10%	-5%	0%
$1,800	0.55	0.56	0.58	0.59
$1,700	0.58	0.6	0.61	0.63
$1,600	0.62	0.63	0.65	0.67
$1,500	0.66	0.67	0.7	0.71

Only Open-pit costs modifications

Figure 7-13: Minimum Required Open-pit Head Grades for Desired Simultaneous Changes in Chosen Cost Parameters

The analysis is done by applying a flex factor to costs that are considered as less reliable. These costs include the following:

Open-pit

- Mining: Operating Expenditure, Sustaining Capital and G&A costs
- Processing: Sustaining Capital and G&A costs

Underground

- Mining: G&A costs
- Processing: Sustaining and G&A costs

Flex factors ranging from -20% to +20% with intervals of 5% were applied. The results associated to these intervals are provided in figure 7-12. The intervals have been applied to all included costs deviations listed above with a throughput capacity of 6.9 Mt. This capacity is chosen in order to increase the possibility of improvement. The figure indicates that despite the changes in costs, the required gold ounces will not be obtainable.

Improvements could hypothetically be reached by altering the costs in a desired matter. This would imply a decrease in open-pit costs together with a fixed underground configuration cost perspective. The underground configuration is similar in both scenarios, except that the UG Only will include the plant modification costs. Therefore, these costs are not altered for this evaluation. The results are provided in the figure 7-13.

The results are either highlighted in red or orange. These both indicate that they are either slightly lower (orange) or equal to the minimum required head grades from figure 7-12. It is evident that despite these slightly favoured changes in cost deviations the desired results will not be reached. This goes for all cost deviations for both simultaneous and only open-pit costs modification scenarios. Another observation is that, besides the orange highlighted boxes, the head grade remains the same as a result of changing only the open-pit costs. This implies the minor influence these potential price changes will have on the overall results.

Chapter 8

Discussion

8-1 Introduction

As mentioned, the results are a product of the modelling done in Excel. The model's main objectives was to optimize the blending procedure and to produce a dynamic strategic mine plan. The final results are based on the financial modelling aspect which helped determine the potential value improvements. The methodology was set up in order to provide answers to the research questions which help test the hypothesis. Several different methodologies were produced prior to the current one used. Continuous revisions and countless adjustments to the methodology have allowed for a correct implementation of data. The produced results are deemed fit but this chapter will discuss the cohering trends and provide clarifications on its shortcomings.

8-2 Analysis of Blending Optimization

The blending optimization is based on the physicals from three different sources. The physicals related to the two open-pit sources were provided by the MTP engineers. These numbers were based on the latest BP20 physicals, as of July, 2019. The third source, originates from the underground consultant engineers and will undergo changes throughout the different stages of pre-/feasibility studies. This implies that current results will always differ from the actual production numbers. For this reason, it must be noted that the obtained results remain to be considered as a product of a high-level project evaluation. Changes in newly updated physicals, as a result of newly interpreted ore reserves, might therefore change the outcome of the overall study.

The objective function for the blending optimization is set to maximize the recoverable ounces due to the consideration of the time-value-of-money. The set of cohering constraints are based on rational interpretation that there will not be any negative amount of mill feed from either of the three sources. However, one notable comment must be assigned to the throughput constraint. The blending optimization has been set up to fill the mill. Hence, the relationship between residency time of ore in the mill and its recovery is not being considered due to the lack of actual testing experience. The metallurgical recovery between open-pit and underground material was currently to remain as a product of the ROM head grade. For now it deemed fit to use the current estimated maximum throughput rates according to the BP19 yearly feed numbers, together with current known metallurgical recovery models.

Generally the blending optimization has provided considerable results related to the forecasted production scheduling. The date for the plant modification is considered as reliable and can be referred to for actual future planning procedures. Prior to the plant modification the current forecasted open-pit and stockpiled material has proven to be slightly short of feed, in order to utilize the mill to full capacity. Therefore, slight changes could be considered regarding delays or further utilization of that available capacity.

8-3 Analysis of the Base Case Results

The results for the blending optimization have shown a clear role of the utilized objective function. The high ore grades quantities have priority of being processed due to their increased recoverable gold ounces. The results presented as the base case mill feed, clearly show that a higher ratio of mine-to-mill against stockpile-to-mill feed results in higher contained gold ounces.

The following years involve the underground material which gains priority over the remaining sources. Despite the minor additional inflow of high grade underground material in 2022, a different result has been observed. The lower quantity of high grade mine-to-mill feed, thus extra inflow of low grade stockpile-to-mill feed results in a drop in gold contained gold ounces relative to the previous year. These results prove the effect of the utilized objection function, which pushes the maximum recoverable gold ounces.

The year 2023 has a peak in gold ounces, which results from the significant additional high grade underground ore. The following year, indicates the first year after the open-pit mine closure with merely two sources of ore feed. The stockpile material still contains significant amounts of gold, which can be observed when analyzing the years from 2026 onward. These final years of only processing underground ore show a clear relationships between throughput rate and contained gold ounces. The relationships proves that the amount of gold ounces is directly proportional to the throughput rate. Whereas, the recovery is related to the total head grade which also is implemented into blending optimization. Due to the fact that the head grade has remained rather stable, the relationship between the throughput rate and recoverable gold ounces remain relatively constant.

The productivity is directly related to the obtained revenue which has been presented in figure 7-2. The trends are similar, however the revenue is dependent on the recovery factors that had not been visualized previously. As mentioned previously, the negative NPV at the end of LOM results from the unchanged administrative costs. The costs can be explained through the following interpretation. As the mining activities gradually reach deeper depths, it will become more feasible to conduct additional exploration drilling and infillings. Therefore, when future exploration activities prove additional ore reserves; additional material may be mined which would lead to an increase in NPV. On the other hand, high closure costs may be implemented for this final year which has not been disclosed due to the degree of high level analysis.

8-4 Analysis of the Head Grade Case

The quantity and quality of the data for the calculation of the head grade has been discussed in the previous chapters. An extensive elaboration was conducted regarding the results however, additional aspects will be discussed in this paragraph.

Generally, the results have not resulted in the expected outcomes. It was initially expected that the results would prove that a potential plant modification would deem unnecessary. The remaining gold situated within the current UPL which is below current operational cut-off grades, has proven to be insufficient. The quantity of this mineralized waste has proven not to meet the required incremental average gold ounces.

Together with the potential clearance of the plant modification, a major capital expenditure is needed for the storage of incremental open-pit tailings. These associated costs are significantly influenced by the strict space restrictions, together with further legislative and community issues related to the construction of raises of or additional TSF cells. This necessary trade-off is translated into the calculations.

The results associated to this trade-off have shown that the *UG Only* scenario would initially generate a specific profit which also must be obtained through the concurrent mining operations. The costs for *UG Mining with OP Configuration* result in a lower total cost when compared to the *UG Only* scenario. This is predominantly a result of the fact that the capital expenditure for the plant modifications are considered for each year individually. Reason for this, is that the possibility of neglecting the modification may occur each year. However, this would only account if the previous year was considered as possible. If the previous year would not be possible, then the following years will not require any further analyses. Once the modification has occurred the possibility of processing incremental open-pit ore will be discarded, due to the increased cut-off grade calculated for the scenario with a modified processing plant.

Since the amount of ore being produced for the *UG Only* scenario and the *UG Mining with OP Configuration* scenario are the same; the total costs for the *UG Only* is higher. It is therefore, correct to assume that the plant modification should be discarded regardless of the *Incremental OP Feed*. However, this *UG Mining with OP Configuration* scenario will only produce such result in combination with the *Incremental OP Feed*. Reason for this, is that the fixed costs will remain the same but the amount of ore to cover those costs will decrease. Therefore, the total cost per tonne of ore will increase when inducing a lower throughput rate.

Another remarkable observation was the negative profit that is produced by the *Incremental OP Feed*. This feature is a result of the fact that the *UG Mining with OP Configuration* scenario and the *Incremental OP Feed* are linked together. The first mentioned feed, will only be equal or more profitable when complemented with the *Incremental OP Feed*. This was considered as the only approach in order to determine the required incremental open-pit cut-off grade.

The amount of required gold ounces would be unknown, if not having determined the subsequent *Net Revenue*. Therefore, this *Net Revenue* is the driving factor for reaching a state of equilibrium between the scenario's. Subsequently, the *Net Revenue* determines the desired minimum head grade. It must be noted that the ounces related to that head grade will not have a recovery of 100%. Therefore, the head grade must be higher to account for this loss in recoverable ounces. However, the recovery is a function of the head grade, which must be used as input; this head grade is known as a result of the throughput and the gold price. Therefore, the iterative process was considered as fit in order to obtain the desired results.

The results have shown the relationship between the throughput options and cohering head grades. Increases in the throughput options have shown a decrease in minimum required open-pit head grades. However, with these increased throughput options are subsequently more difficult to achieve. This could be explained through the depletion of valuable material within the pit.
When considering the results associated to the discounted cash flow, similar trends were made visible. A slight increase in head grade is required to account for the applied discount rate of 9% which is consistently applied to projects in Ghana. Even with consideration of an hypothetical extreme gold price scenario; the remaining gold quantities will not be sufficient to produce a higher overall profit.

Another important point of discussion can be assigned to the obtained input cost parameters. The assumptions have been discussed however, they remain to be assessed critically. Despite a sensitivity analysis with minor changes in the cost parameters assigned with lower confidence, the produced results remain unchanged. This all can be explained through the major cost contribution factor of the additional TSF capacity. The amount assigned to this capital expenditure outweighs the plant modification costs significantly. Especially, when considering the low amount of mineralized waste left within the current pit. However, with actual costs may prove significantly different which can lead to major alterations in the results provided in this thesis. This also may account for the actual metallurgical recovery and throughput relationships. These relationships might actually be highly in favour of an increased recovery associated to a blend of open-pit and underground ore. Yet, the remaining mineralized waste in the material is likely to prove insufficient to cover the high amount of required capital expenditure.

Since the minimum required open-pit head grade is not obtainable, following scenario analyses could not be conducted. The associated results are not provided due to the lack of a cohering incremental operational open-pit cut-off grade. The amount of incremental open-pit ore which will be fed under current plant configurations can therefore not be determined. The resulting NPV calculations for a different set of potential options is therefore dismissed within the thesis. However, it can be stated that the excavation and processing of any remaining incremental open-pit material will result in a lower overall value. This can be interpreted from the methodology behind scenario comparison for the minimum open-pit head grade determination.

Part V

Conclusion and Recommendations

Chapter 9

Conclusion

This chapter will elaborate the conclusions that have been drawn from the heuristic analyses associated to the strategic mine planning presented in the previous chapters. The conclusion is presented with regard to the research questions stated in the introduction chapter.

9-1 Optimum Blend and Plant Modification

The strategic mine planning optimization has provided results on two main potential future options for potential transition from open-pit to underground operations. Results from the open-pit MTP and forecasted underground physicals have provided a reliable foundation for the three input sources, required for the blending optimization. Current blend is a product from the maximization of the recoverable gold ounces while aiming to reach full plant capacity. The principle of time-value-of-money substantiates this approach by granting priority to the underground ROM to the mill. The lack of actual metallurgical testings, regarding certain blends of open-pit to underground ore ratios, have implied this approach to be sufficient for further implementations. Results have shown that the optimum period of modification will be in 2026, once all open-pit material is processed. Processing of any open-pit material through a modified plant will lead to a substantial loss of value, as the operational cut-off grade will be significantly higher. Reason for this, resulted from the fact that open-pit material would be considered as waste when considering the plant modification.

9-2 Incremental Open-pit Head Grade

Subsequent mine planning was based on the analyses of two potential options with 2026 as starting year. Results have proven the utilization of current processing plant configurations to be infeasible, due to the lack in quantity of valuable material within the remaining pit reserves. A potential incremental open-pit cut-off grade can therefore not be realized in practice. The lack of material is related to the high demand of recoverable gold ounces to help cover additionally implemented costs. These additional costs result from the required additional TSF capacity, to help store the incremental tailings. Tight space restrictions and community demands are considered as the challenging factors associated to such aerial expansions of TSF cells. These challenges are translated into high cost estimations for the most likely options regarding such expansions.

9-3 Throughput rates, head grade and recoveries

Changes in throughput options to potentially find reasonable incremental open-pit feed quantities have also proven unsuccessful. As the throughput rate decreases the cohering head grade will have to increase to account for the losses of available gold containing capacity in the plant. Additionally, the relationship with the recovery has been made clear and reserve estimates have proven insufficient. Current estimated head grades will increase in order to account for the recovery losses. The same goes for the time-value-of-money which must be accounted for, especially for such future planning activities. Further analyses related to the potential favourable changes in gold prices together with an additional sensitivity analysis have not altered the final results. It may therefore be concluded that current planned activities regarding the transition from open-pit to underground mining operations is therefore the optimum choice. The relationship with desired changes in specific decision variables are visualized and shown to be insignificant for groundbreaking changes in the overall strategy.

Chapter 10

Recommendations

Based on the conclusions of the results associated to the conducted analyses, current set of input parameters and their cohering relationships have proven to produce results that can not be achieved. However, the analyses were conducted based on high-level evaluation parameters which might turnout different in reality. These potential alterations could hypothetically produce achievable results. For this reason, the produced model allows for easily adjustable parameters to indicate the possibility of the potential changes. Further evaluation on the input parameters are therefore recommended to conduct, in order to narrow down the uncertainty measures associated to these variables.

Furthermore, underground development operations could be analyzed and considered for further optimization. The material that will be excavated during the development works, in order to reach the stoping levels for underground operations, could potentially be used for blending prior to the plant modification. This approach would delay the modification and therefore increase the total NPV of the operation due to the time-value-of-money.

Additionally, results have shown that the full capacity would not be reached during the final year prior to the plant modification. This minor shortfall of throughput could potentially be filled with current sub-grade material, as indicated in chapter 6. This will require additional analyses of differential mining costs between ore and waste, with consideration of the rehandling costs for the stockpiled material. Similar to previous recommendation the additional material would hypothetically add value to the operation.

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Personal Communications

The following people have been cited as a personal reference within this thesis:

- Joep Coenen; General Manager of Akyem Gold Mine
- Steven Woods; Senior Director - Early Stage Projects
- Kristina Huss; Consultant Engineer - Underground Mining
- Peter Marx; EPCM Manager
- Eduard Manson; Senior Geotechnical and Hydro Engineer
- Felix Adiaana Kaba; Medium Term Planning Mining Engineer
- Rainer Juati; Short Term Planning Mining Engineer
- Patrick Ansah Aboagye; Metallurgist

Appendix A

Synopsis

MSc Thesis Synopsis

Optimization Study for the Transition from Open-pit to Underground Mining of Akyem Gold Mine, Ghana

By

T.L Goense

Graduate Student Master of Science
in Resource Engineering

at the Delft University of Technology, Aalto University and RWTH Aachen University

Supervisor(s):	Dr. M.W.N. Buxton, Dr. Masoud Soleymani Shishvan Ir. Kristina Huss	Delft University of Technology Delft University of Technology Newmont Golcorp Corporation
Thesis committee:	Dr. M.W.N. Buxton, Dr. Masoud Soleymani Shishvan Dr. Rodrigo Serna, Uni.-Prof. B.G. Lottermoser, Ir. Joep Coenen,	Delft University of Technology Delft University of Technology Aalto University RWTH Aachen Newmont Golcorp Corporation

TU Delft

October 21, 2019

Motivation

I am a MSc. student from the Delft University of Technology in the Netherlands with a BSc. in geology. In September 2017, I decided to follow a MSc. degree in mining engineering. This is a triple degree from three different leading technical universities: Aalto University, RWTH Aachen University and TU Delft. This cooperation between three international top-universities has taught me to broaden my horizon with new knowledge from different learning environments. Each university having their unique ways of teaching with all their own specific vision of proper education. After being introduced to a bright multi-cultural group of fellow students, that have been around throughout the entire study period, I have hoped to apply my newly gained knowledge elsewhere in the world.

My current studies at the university have nearly been successfully finalized. The remaining objective is to finish with a thesis starting in 2019. Newmont Mining has provided with a special opportunity to start an optimization study for the Akyem Gold mine. This will require me to acquire and analyze data from the mine in order for me to improve their operations. I believe that, with my expertise and eagerness to improve processes, I could be of great value. Mining in resource rich countries like Ghana must always remain to be done in both an ethically and technically approved manner. This is one of the key things that I have learned throughout my studies with different cultures. I am convinced that by improving these operations, not only the mining industry will improve but also the country as a whole.

My goal is not only to gain additional knowledge from the company but most important to get to learn a new culture from a different continent. Meeting the local people that work and live around the mine will provide me with special experiences and certain life lessons. A positive image that can be created between different individuals will remain forever and can be passed on to others. Altogether I am very fascinated by this wonderful opportunity for the possibility of visiting Ghana and wish to be able to give something in return.

Goal

Finding the optimum solution for the transition from open-pit to exclusively underground mining operations regarding the ore feed for the processing plant at Akyem mine, Ghana.

Now that open-pit operations are nearly coming to an end and underground operations will commence, the amount of throughput of ore to the processing plant will change. In order to maximize value it is necessary to keep the processing plant running utilizing its capacity with a stable feed of ore. The question is to what extent the ore from open-pit stockpiles should be blended with ore from the underground mine.
This thesis will require me to first visit Denver, Colorado to gain some knowledge from the processing group together with additional training for approaching this task. After this I will be ready to visit the mine in Ghana.
By first getting an overall proper understanding of the operations at Akyem mine by speaking and following local employees of the mine with an up close insight, I will then start to acquire and analyze the received data. Once all relevant information is received I will be able to analyze different scenarios for the mine which will lead to cost reductions for further operations. This will be finalized at Delft University of Technology, Netherlands where I will be writing my thesis and providing Akyem mine with my results.

TU Delft

Hypothesis

Open-pit mining at Akyem mine has nearly reached its end, whereas their underground mine will be opening in the coming years, if the project goes successfully through feasibility studies. Due to the fact that the processing plant has a greater capacity than what the underground mine will be able to provide, there will be opportunities for blending ore from both different operations. I believe that once a desired mixture is made the processing plant can still be fully utilized which results in lowering costs of operations.

Objectives in Ghana

1. Achieving proper understanding of current mining activities at Akyem Gold mine
2. Analyzing both open-pit and underground ore by estimating optimum cut-off grades
3. Obtaining cost image of open-pit and underground mining operations
4. Obtaining general costs at the site

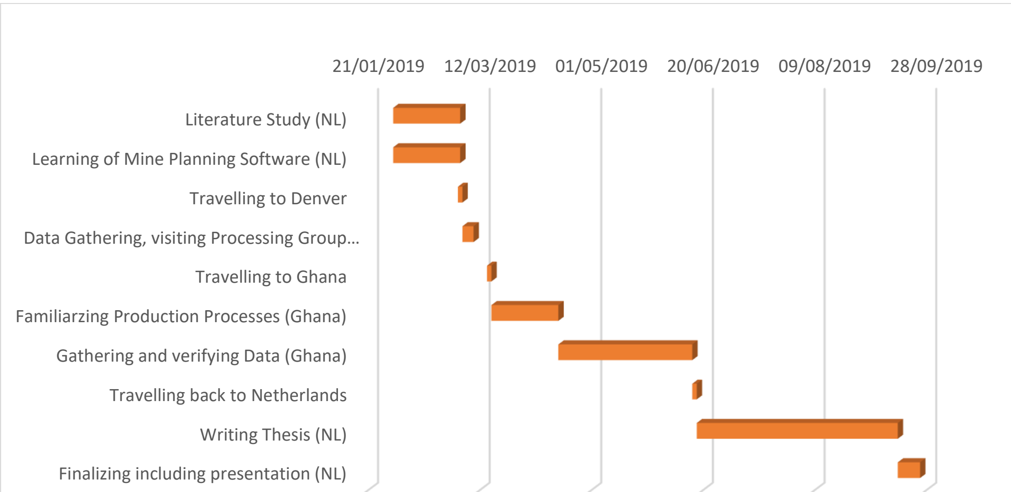
Scope

The research will be conducted under supervision of the Underground Mining Consultant Engineer, Kristina Huss. Data acquisition that will take place at Akyem Gold mine in Ghana, should be sufficient to produce a proper optimization of operations. Once all relevant technical and financial data has been obtained, the completion of the research together with writing of MSc Thesis shall be done at the university in the Netherlands.

The study in Ghana will mainly focus on obtaining a cost structure and all operating parameters. Mine planning will be a necessity for the optimization of operations by optimum utilization of the processing plant. The combination of data from Ghana and Denver will allow for the statement of conclusions regarding the subject.

Time Plan

Table 1: Time plan with duration given in days



Appendix B

Maps

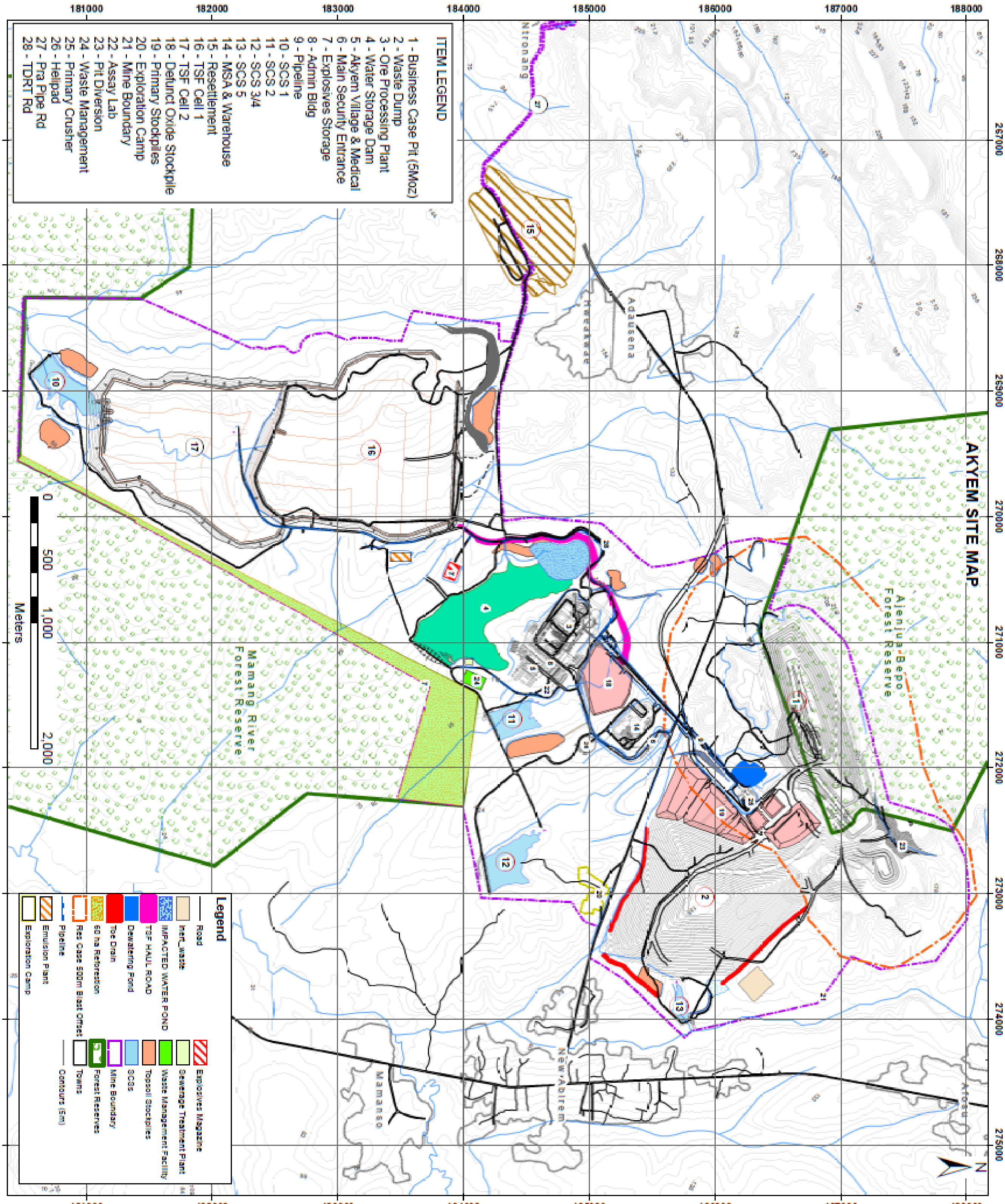


Figure B-1: Akyem Site Map: illustrating main site features including facilities

Modified from Survey Department

Appendix C

Lithologies

CODE	NAME	Mineralogy	Grade (gpt)	Modeled Domains	DESCRIPTION
Alteration					
SA0	Silica /Albite (0)	Chlorite-Biotite	>0.2		Unaltered rock with mylonitic fabric but insignificant or no alteration. "Primary" textures like grain size gradation and "bedding" may be noticed. Color is grey to dark grey
SA1	Silica /Albite (1)	Calcite-Chlorite	0.2-0.6	QSP1	Zones of Metasediments with mylonitic fabric and <50% quartz-sericite-Fe carbonate-pyrite(QSP) alteration by volume. Color is green to yellowish.
SA2	Silica /Albite (2)	Iron-Carbonate Chlorite Sericite Pyrite	0.6-2.0 (avg. 2.0)	QSP2	Zones of Metasediments where mylonitic fabric is well developed and moderate to intense quartz-sericite-Fe carbonate-pyrite (QSP) alteration is observed over a minimum of 2 meters. A minimum of 50% of rock volume has to be altered. Color is yellowish to tan (autumn colors)
SA3	Silica /Albite (3)	Quartz Iron-Carbonate Sericite Pyrite	>2.0	BX	Pervasively silicified rock with strong sericite often developed over mylonite or breccia units. Preserves original rock texture which is commonly fragmental. Breccia may be re-mylonitized and show a pervasive mylonite fabric. Color is tan to dark grey depending on sericitization, silicification and Fe-carbonate alteration. Sulfide content varies, and higher sulfide varieties may correlate with gold grade. No chlorite
Structure					
BX	Breccia	Quartz Iron-Carbonate Sericite Pyrite	>3.0	BX	Discrete breccia units with >50% by volume breccia, minimum thickness of 2 meters and moderate to intense quartz-sericite-Fe carbonate-pyrite alteration. Defined by clear and pervasive sub-angular to rounded fragments in a matrix of hydrothermal minerals with clear evidence of shearing or fragment abrasion.
MY	Mylonite		>0.2		zone of ductile deformation during intense shearing encountered during folding and faulting
Formation					
GRAP	Graphite			GRAP Surface	Occurs as discrete units or graphitic zones in both Metasediments and Metavolcanics, sheared and/or brecciated. The most continuously developed unit separates footwall Metavolcanics from hanging wall Metasediments. It is the Akyem Carbon Fault (ACF). GRAPR for Graphitic rubble, GMY for Graphitic mylonite
GRAPR					Graphite broken into rubble with fragments covered with striated slicks and mixed with gouge or finely crushed rock.
Lithology					
QV	Quartz Vein			QV	Quartz veining or silica replacement is defined where a minimum of 75% quartz veining or silica replacement occurs. A minimum of 0.5m was logged.
MV	Mafic Volcanics			MV Contact	Chlorite dominated greenschist facies meta-mafic volcanic unit with minor shearing. Relatively undeformed; laminated to semi-massive. Minor quartz-eye/qtz veinlets included. This is located beneath the graphitic fault zone. Pale greenish andesitic, largely massive but can have a shear fabric close to graphitic units and splays within it.
MSED	Metasediment			MSED Contact	Generic term for silty, laminated-to-massive commonly chloritic sediments of turbiditic origin. Other primary sedimetary structures (ripples, etc.) are rare. At Akyem, these sediments are alternating cm to dcm-scale alternating lithic sandstone, graywacke, siltstone and shale. Graded beds are locally common. Gold is associated with moderate to intense alteration assemblages within this rock unit
RLAC	Red Lateritic Clay				The RLAC unit is comprised of a 0 to 1m true thickness layer of red hematite-rich kaolin-dominant highly plastic clay with quartz fragments. It commonly includes transported materials and always occurs as the upper most unit, typically thickest in drainages. This unit is not always developed across the project.
SAP	Saprolite			Bottom Of Oxide	Deep residual weathering of primary rock types produces Saprolite. This unit is autochthonous unlike RLAC and DUR that are allochthonous. Akyem saprolite is tan to grayish-white silty clay and clay formed from intensely weathered/oxidized bedrock. The unit ranges from 0 to 20m thick, with an average thickness of 15m, and primary textures are locally preserved in lower portions of the unit. SAP was defined as weathered material that could be broken between a geologist's fingers
POX	Partially Oxidized			Top of Fresh rock	POX was defined as weathered material harder than SAP. The bottom of POX was defined as the last depth of oxidation, excluding weathering associated with structural features like joints and shears. Quite often POX has islands of fresh rock in it
Veining					
VEINING DENSITY	 5%-10% > 15%	Quartz Quartz Carbonate Pyrite			Sheeted Quartz Veins, <50% of interval occurs as parallel to sub-parallel, distinct-margin, white quartz veins @<1m spacing. Single generation of banded, planar QV + sulfides. Rare VG

Figure C-1: Description of Akyem Deposit Lithologies

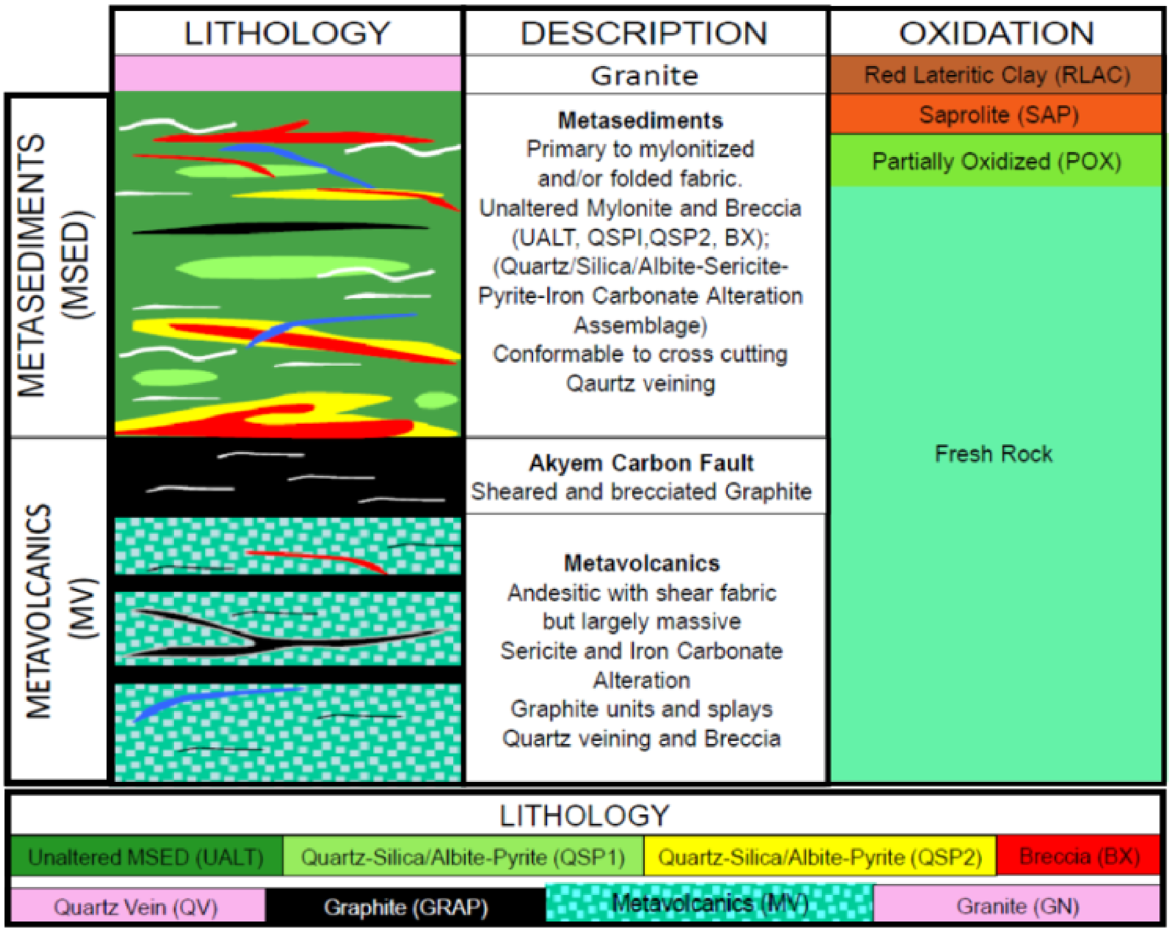


Figure C-2: Simplified Lithostratigraphic Column Akyem Gold Mine Deposit