Abstract

Feasibility study grants a specific technical, environmental and financial base for an investment decision process. This study presents long-term production planning of the Aynak Central Copper Deposit (ACCP), located in Logar, Afghanistan. This research was conducted with MineSight software and utilized the logging database provided by the Ministry of Mines and Petroleum (MoMP), Afghanistan. Moreover, the Inverse Distance Weighting (IDW) method was used for resource estimation and Lerchs Grossman (LG) technic was applied for pit optimization. The analysis shows that ACCP has huge quantity of sulphide and oxidized copper with a high copper grade. In addition, ACCP is technically feasible and economically viable and will be mined by open-pit mining method.

Declaration of Authorship

"I declare in lieu of oath that, the entire contents of this thesis is my own work except where otherwise indicated. All references and literal extracts have been quoted clearly. Information sources of figures, charts and tables have been acknowledged. This thesis has not been submitted to any other institution and has not been published".

Date: ______________________ Signature: _________________________

Acknowledgement

I would first like to show my greatest appreciation to Almighty Allah for providing me the golden opportunity and strength to accomplish this great work. Additionally, I would like to offer my special thanks to my supervisors, Prof. Dr. Carsten Drebenstedt and Prof. Dr. Peter Moser for their support, suggestions, remarks, and patience throughout the entire study program and throughout writing this thesis.

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Mohammadajan Baryal
Freiberg, Germany
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Chapter 1. Introduction

Considering the high demand for minerals, mining companies are desperate for having a feasible plan to extract minerals technically feasible and economically viable. In this case, feasibility studies are carried out which serve as the foundation report for the development of a mineral inventory. During conducting feasibility study, deposit modelling is the main and base element which makes a contribution to long-term production planning.

Long-term production planning is usually accomplished for feasibility or financial studies. It fortifies reserve estimation, pit design, and is a crucial factor in decision-making procedure.

Production planning mainly covers resource estimation, ultimate pit determination, production capacity, mine life, and sales revenues. Later on, it serves as a base line document for operational planning, which involves day-to-day, monthly and yearly planning to keep the operation of an active mine.

In this thesis, long-term planning was conducted for the Aynak Copper Deposit located in Afghanistan. It comprises the main features including reserve estimation, production capacity, mine service life and some other issues related to production planning. Furthermore, a 3D orebody model and a 3D block model was created for the deposit using MineSight software.

Aynak copper deposit is one of the biggest deposits in the world. Aynak copper deposit has been investigated and surveyed but has not been developed and exploited until yet. The detailed exploration of the central part and western part of Aynak was done by the Technical Group of former Soviet Union and Geological Exploration Department of Ministry of Mines and Industry of Afghanistan in 1978 and 1987 respectively [1].

This deposit is the biggest resources development project in Afghanistan, this is why the government has decided to choose the contractor based on international bidding. In August 2006, the MoMP sent the available exploration data to the world-famous mining companies. The interested companies were from different countries including United States of America, Russia, Canada, Australia, China, India and Kazakhstan. On November 20, 2007, the government announced the consortium of Metallurgical Corporation of China (MCC) and Jiangxi Copper Co. Ltd (JCC) as the owner of the bid amongst mentioned mining companies. The contract was officially granted on May 27, 2008, which allows the contractor to start the development at Aynak copper deposit [1].

The MCC has done the detailed feasibility study of Aynak copper deposit. In July 2014, this report was reviewed by Steffen, Robertson and Kirsten (SRK), a consulting enterprise Ltd, from the United States of America which was employed by MoMP of Afghanistan. SRK has only confirmed the open-pit mining method of MCC Feasibility Study Report and rejected the underground mining method because of the low level of rock mechanic studies [1] [2].

Chapter 2. Theoretical Basics

This chapter mainly comprises theoretical information related to deposit modelling, production planning, block modeling, waste management, mineral resources and reserves estimation, pit optimization and open-pit design.

2.1. Production Planning

Production planning is one of the most important tasks of mining. There are abundant determinants that affect the size and shape of an open pit including geology, grade, topography, bench height, and etc. During planning, make sure that the factors mentioned above are used properly because they are project specific and each of them will influence the pit. Production planning is classified into two types considering the time period; *long-term* and *short-term* production planning.

Short-term production planning which is also called operational planning is required for accomplishing the mining operations but long-term planning is carried out for feasibility studies and is a key factor in decision making process. During long-term production planning the following key points which are very helpful, must be considered [3] [4]:

1. The goals must be as much clear as possible because we deal with grade evaluation, geology prediction, and economics speculation.
2. The planning must be explained to those who accomplish plans and to those who make decisions to avoid confusion and abortion.
3. Geometry is also significant for a planner as is mathematics because we deal with quantity of earth.
4. Time must be used as much productive as possible because performance, productivity and cost adeptness depend on it.
5. The plan should be agreeable and convenient to turn into the company's objective and should not be the exhibitor of the planner's concepts.

2.1.1. Aims of Production Planning

The key points and the fundamental objectives of production planning are listed well in Open Pit Mine Design and Planning, 3rd Edition by Hustrulid and Kuchta 2013. In this study, the crucial goals are described below [4]:

1. To extract ore in such a manner that minimize the producing cost for a kilogram of metal.

2. To keep the operations ongoing by the plan with the combination of satisfactory equipment operating room and active bench, etc.
3. To create a reasonable and simple achievable start-up schedule considering the available resources such as equipment, infrastructure, and etc.
4. To decrease slopes uncertainty due to a reasonable geotechnical research, an accurate planning helps to increase pit slope angles.
5. To analyze the cutoff grade and economic benefit of ore production rate.
6. To accept the mining strategy, development plan and equipment selection of the expected mine.

2.1.2. Ultimate Pit

An ultimate pit which is also called final pit, is the pit which occurs at the very final stage of mining. There are several methods for designing the final pit and they are differentiated by the quality and quantity of the available data, size of the deposit, available aid of the computer and the expectation of the engineer. The following are the types of methods used for establishing the ultimate pit but in this thesis, the focus will be on computer methods [3] [4].

1. Manual methods
2. Computer methods
3. Computer-aided manual methods

Generally, the term “optimization” means getting a better result through a process by modifying the inputs, design, and approach. A mathematical accurate description of the term is to discover an admirable financial worth of an activity and this admirable financial worth addresses the littlest and highest values. The littlest value means that the minimal feasible worth to decrease the total expenses and the highest value means that the maximum feasible worth to increase the total profit [5].

A model for pit optimization covers the following aspects [5]:

1. **Goal:** to increase the financial worth of the pit.
2. **Input data:** a worth of currency ($) should be added to each block in a block model. The worth of each block is estimated allocated on the predicted cash flows if the block is dug and there can be positive worth (ore) and negative worth (waste) for blocks.
3. **Goal activity:** the addition of all values for all blocks included in the pit is the entire worth of the pit.
4. **Decision variables:** there is only and only one decision variable with two attainable settings to add or remove for each decision changeable in a block model.

5. **Restrictions:** a block can only be added into pit if the blocks overhead on it are added as well. Blocks overhead are added to be mined for pit slopes stability. The reliance of these blocks is described by arcs.
6. **Optimization approach:** Lerchs-Grossman (LG) approach is one of the most common approaches which is based on a graph theory and the steps are rerun until the optimum result. The result is the maximum positive worth of the blocks that conform all blocks and the objective is optimized.

It is strongly needed to determine the open pit limits during short-term and long-term production planning because they describe the quantity of extractable ore, the capacity of metal and the volume of waste that must be removed during the time of workings. After developing the pit limits and regulating the rules for categorizing the in-pit materials the ore reserves in term of tonnage and grade can be estimated. It should be mentioned that the planning of tailing areas, waste dumps, access roads, concentrating plants and all other surface accessories depend on shape, diameter, and position of the final pit [3] [4].

To design the final pit, for physical and economical parameters the financial worth will be accredited and the final pit will depict the utmost borderline fulfil the mentioned criteria. The existing material inside pit will fulfil the following two criterions [3]:

1. A block will only be dug down if it can cover all the expenses for its digging down, processing, marketing and removing the overburden.
2. Any block that fulfill the first criteria will be added to the pit to preserve the resources.

Computer methods

1. Floating Cone Method

This method is the most popular computerized method based on alike idea as Incremental Pit Expansion but manual mediation can be decreased or removed.

The foundation of the expansion is created by a group of blocks. If the base's grade is higher than the mining cut-off grade then, the expansion is outlined upwards to the highest position of the model. The cone is created by using the relevant pit slope angles [6].

A tabulation that includes all blocks in the cone is done for the costs of mining, processing, and for the revenues determined from ore. If the total earnings are higher than the total expenses for the blocks in the cone, then the cone is considered to be economic because it has the positive net value.

Each block has to be analyzed in proper sequence as a foundation of the cone and the final pit depends on the layout in which the next base block selected. This process is expensive for large models [6].

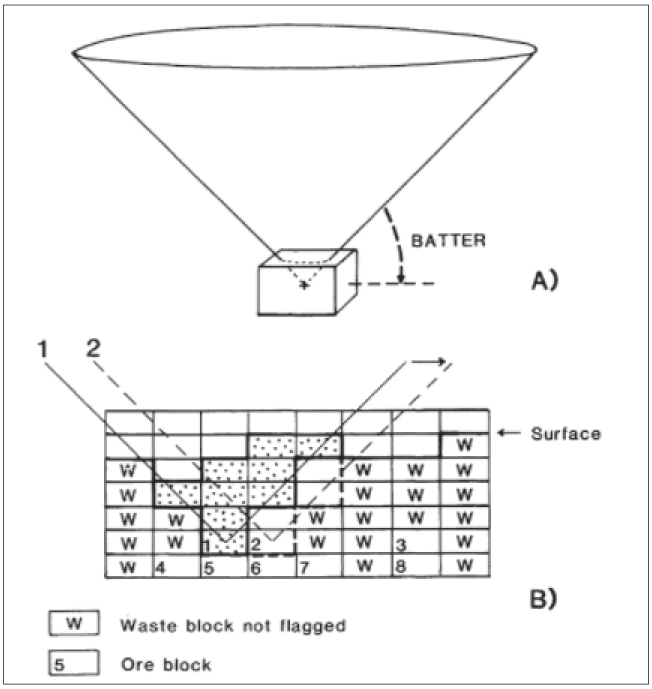

Figure 2.1-1: (a) Inverted cone with apex in the centre of a block. (b) Blocks flagged for the first and second positions of the floating cone [7]

2. Lerchs-Grossman Method

Lerchs-Grossman (LG) is the most common and well authorized technic for determining the ultimate pit. This accounts as an efficient procedure used for determining the ultimate pit for large mines in acceptable time. This procedure uses a direct graph theory G = (V, A) where a node in the graph shows a block in the ore body block model [8].

During assessment of final pit in the form of grade block model, the main goal is to maximize a chosen parameter such as profit, Metal content or marginal value based on discovering a group of blocks.

There are two principle geometries available for approximating an open-pit for a set of blocks [9, pp. 409-503].

1. **The 1-5 pattern:** where 5 blocks on level 1 are extracted to reach into the one block below on level 2 which is shown in Figure 2.1-2a.
2. **The 1-9 pattern:** where 9 blocks on level 1 are extracted to get access into the block below on level 2 which is illustrated in Figure 2.1-2b.

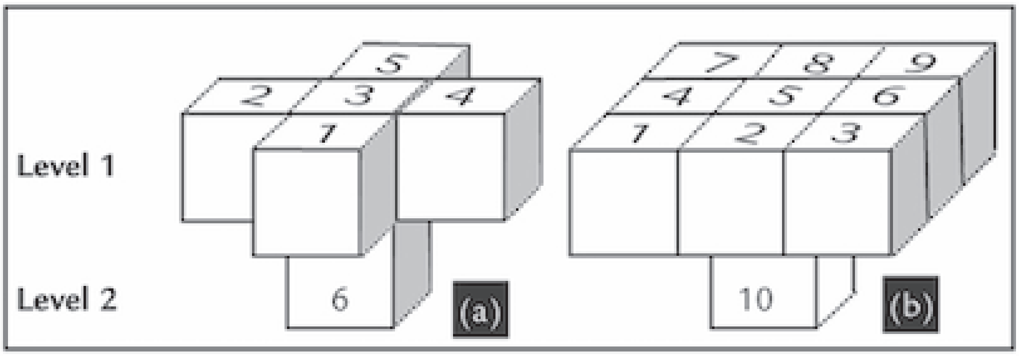

Figure 2.1-2: 1-5 and 1-9 blocks precedence relations [10]

<u>Dilution and Ore Recovery</u>

The waste material that cannot be isolated from ore all along mining operations and has to be mined with it is called dilution. The waste material is blended with ore and then delivered to processing plant. Dilution diminishes the ore grade but rises the quantity of the ore. Dilution is measured in percent and can be calculated using Equation (2.1-1) [11]:

$$Dilution = \frac{tonnage\ of\ waste}{tonnage\ of\ ore + tonagge\ of\ waste} * 100 \qquad (2.1\text{-}1)$$

There are two types of dilution occurs in a mining-block: internal and external dilution. Internal dilution refers to the dilution that in a mining block there is low grade ore pockets or waste inclusions that cannot be isolated and has to be unavoidably extracted with ore. The amount of internal dilution differs in various kind of deposits and depends on the lithology and grade distribution. It is not easy to prevent internal dilution but it is possible. External dilution refers to the dilution that during mining a mining block, the waste outside it is also mined. External dilution depends on the structure of the deposit, geology, equipment, extent of operations, and drilling and blasting methods [11]. Both kinds of dilution are clearly shown in Figure 2.1-3.

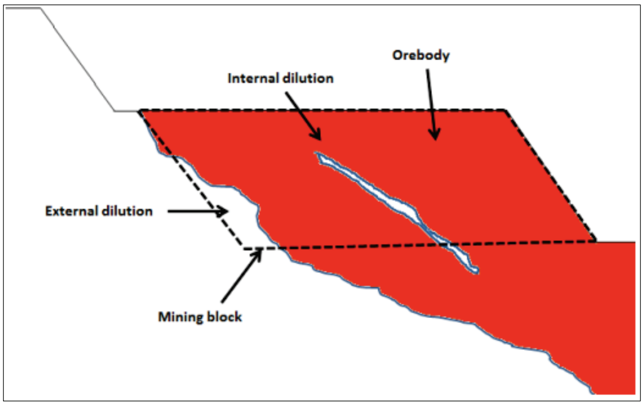

Figure 2.1-3: Different types of dilution in an open-pit mining-block [11]

The origin of this problem is linked to the deposit and mining workings. It is hard to solve this problem but could be minimized by measuring and checking. Even though measuring and assessing dilution is challenging but it assists to develop mine design [11].

2.1.3. Open-Pit Mine Design

Designing an open-pit is one of the challenging part of mine planning. There are several parts that have to be considered carefully. In this thesis, some of the most important components of open-pit will be explained.

Bench Height

The perpendicular distance between the parallel levels of the pit is called bench height. The height of all benches must be alike except that geologic circumstances govern. The height of a bench relies upon on production rate, physical features of the ore body, the size and type of equipment, the climatic circumstances and the selectiveness degree of the ore and waste separation [6].

The bench height must be fixed as high as possible considering the size and type of equipment used for production purpose and as well the safety of the bench. Typically, the bench height ranges from 15 m in large copper mines. Figure 2.1-4 shows the components of a bench [6].

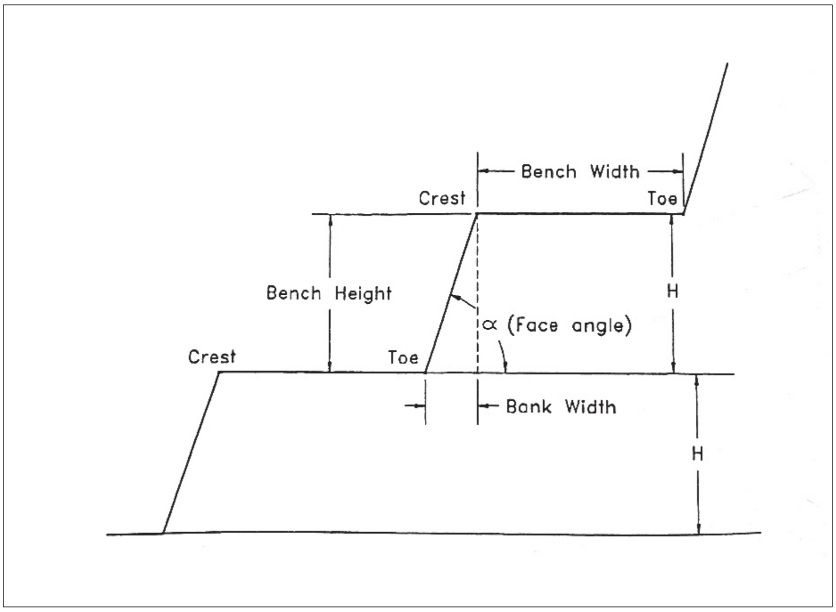

Figure 2.1-4 Components of a bench [9]

<u>Pit Slopes</u>

Amongst most important components of a pit, one is pit slope and is strongly required to be stable throughout the mining operations. Slopes affect the size and shape of the pit and help to ascertain the waste that must be removed in order to extract the ore. A careful assessment of slopes is required for stability. The pit slope is measured in degrees from the flat plane and stated clear. Slope angle depends on rock strength, faults, joints, and water. The pit walls must be fixed as sharp as possible to decrease the strip ratio [6].

The overall slope angle should be smooth to design a road. The angle from the toe of the lowest bench to the crest of the upper bench is called overall slope angle and depends on width, grade and placement of the road. The overall slope angle and bench slope angle is illustrated in Figure 2.1-5 [6].

Bench slope angle is the angle created with the horizontal of the line linking the toe to the crest and is kept being 60°-80° to the horizontal for active benches and 45°-60° for non-working benches. Overall slope angle is constructed with the horizontal of the line combining the lowest most toe to the upper most crest [9] [12].

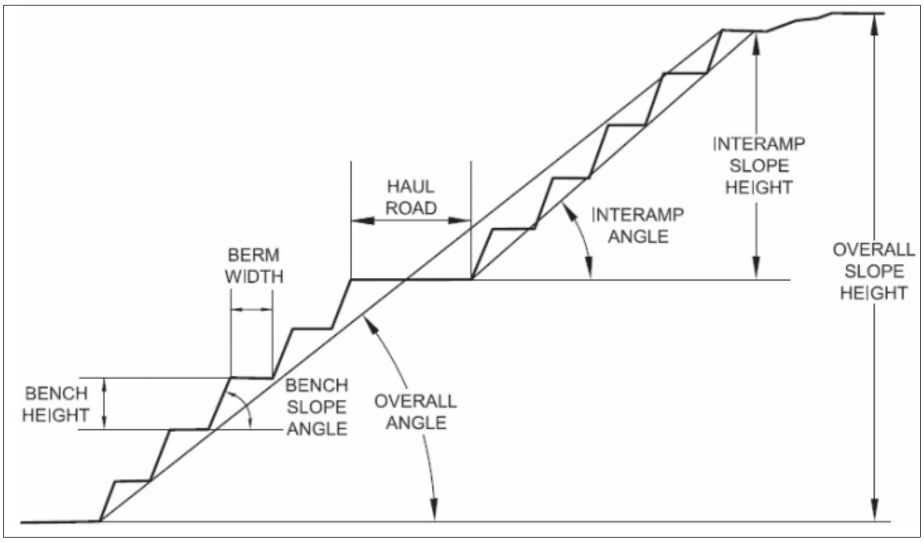

Figure 2.1-5 Main elements of an open-pit mine [13]

Both bench slope angle and overall slope angle have an influence on geotechnical issues, mining systems, and economics. In addition, overall slope angle has less impact or no impact on safety in contrast to bench slope angle if a sound pit slope management procedure is in place [13]

Cut-off Grade

A grade which is used to set apart any two manners of an activity for any particular reason. Usually, the reason used to set up cut-off grade includes the economic features of the project [6].

A mining operator has to use the grade of the block to decide whether a next block must be mined and processed; mined and stockpiled; mined and sent to the waste dump; or not mined at all [6].

Breakeven mining cut-off grade is the grade which pays for costs of mining, processing, and marketing of the block that has to be purposely mined [6].

Another cut-off grade is used for the blocks which are below the mining cut-off grade and cannot be mined for their own financial worth. But these blocks must be mined as waste by lower ore blocks and mining costs has to be paid by the lower ore blocks [6].

Strip Ratio

The number of tons of waste that needs to be removed to extract one ton of ore is called strip ratio. The average strip ratio for the pit is determined by the ratio of waste and ore and the amount of waste and ore included in a pit is determined by the pit design. This is different from the breakeven strip ratio. The last increment extracted along the pit wall is called breakeven strip ratio [6].

Breakeven strip ratio (BESR) is calculated based on Equation (2.1-2):

$$BESR = (A - B)/C \qquad (2.1\text{-}2)$$

Where:

A = ore revenue per ton

B = ore production cost per ton

C = waste stripping cost per ton

Sometimes a minimum profit per ton of ore is included and the formula in equation (2.1-2) for breakeven strip ratio changes as below:

$$BESR = [A - (B + D)]/C \qquad (2.1\text{-}3)$$

Where: D = minimum profit per ton of ore

2.1.4. Waste Disposal

Waste management is an essential and difficult part in surface mining. While designing an open-pit mine, it should be taken into account to design waste disposal sites regarding to their characteristics.

Dump Design

A waste dump is a place where low grade or unprofitable material is placed which has to be removed for particular reason for instance for pit wall stabilization, for haul rod construction and to uncover the high-grade material in surface mining.

The first and most important task in outlining the dump is the choice of a site that must be able to manage the quantity of waste produced during the mine's life. The most significant aspects that have to be considered in the selection process of a site are given below [6]:

1. Capacity and position of the pit
2. Topography
3. Quantity and cause of waste
4. Outer limit of the land
5. Current seepage direction
6. Restoration prerequisites
7. Base circumstances
8. Material managing apparatus

The main goal of designing a dump is to decrease the distance between the origin and the disposition area. A better planned dump can play a vital part in the costs of total activities because handling material is the costly individual factor of the mining cost. To minimize transportation costs, it is recommended to transport waste material from the uppermost areas to the higher elevation and from the lower areas to the lower elevation [6].

Before starting the dump design, two supplementary and significant determinants have to be decided. These determinants are the material swell factor which is 10 to 45% for in situ material and 30 to 45% for hard rocks. The angle of repose which ranges from 34°-37° and decides the capacity of dump and the overall dump slopes [6].

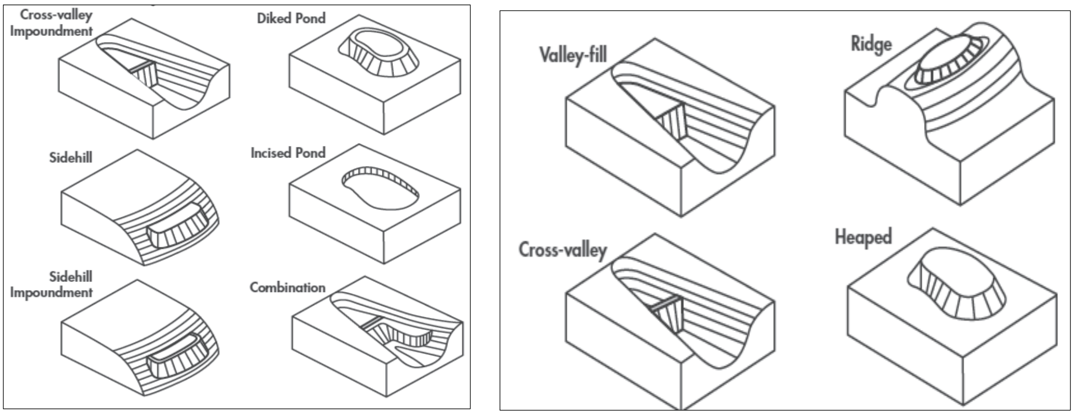

Figure 2.1-6 Mine Dump Configurations [14]

The transportation means also influences the dump arrangement and these mediums are trucks, conveyors and rails considering their utilization.

It is crucial to design a dump in an economic manner considering the transportation means, stability, drainage, restoration and land limits in order to minimize the costs and maximize the value of ore produced [6].

<u>Stability of Mine Waste Dumps</u>

Stability of waste dumps is a crucial part during designing a dump. The general stability of waste dumps relies on the following factors [6]:

1. Dump site topography
2. Manner of building
3. Geotechnical criteria of the waste material
4. Geotechnical criteria of the base material
5. Action of foreign forces on the dump
6. Ratio of improvement of the dump face

All the factors above mentioned play an essential role in the stability and instability of a mine waste dump. Considering the site selection of dump and their topography within an economic distance limitation, the topography is set to be permanent circumstance. The important factor of topography is the existent natural slope of the ground on which the dump has to be built. The safety factor decreases extremely where the surface gradient is higher than 20° disregarding the toughness criteria of the waste and base material [6].

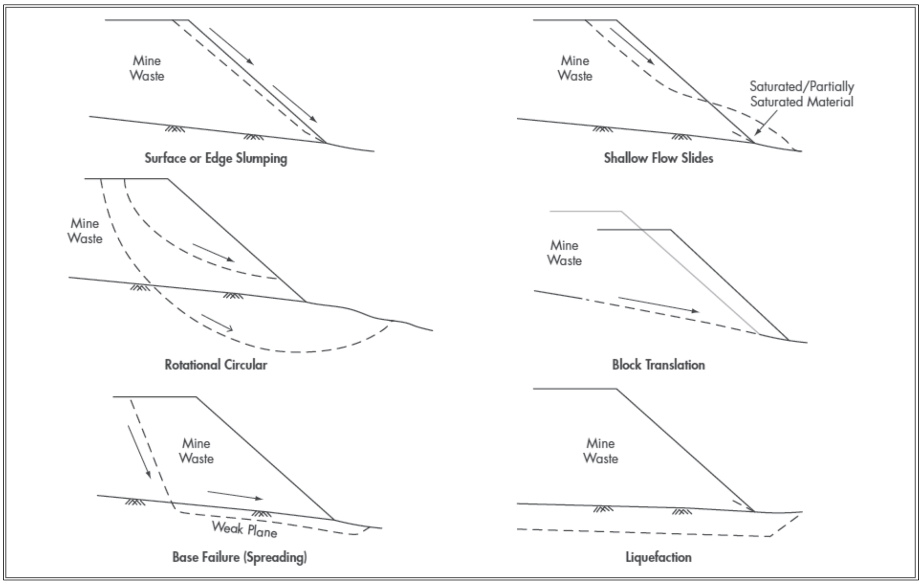

Figure 2.1-7 Waste dumps failure modes [14]

Generally, the two approaches; *in layer* and *end-dumping* technics are used to build up a mine waste dump. End-dumping (built from the top) is a fully disciplined breakdown method where the waste material is accumulated establishing a slope at or adjacent to its repose' angle and the safety factor is appropriately one. To predict and compromise with slope failures, it is strongly advised to watch carefully the working dump face. Layered dump (from the bottom top) technic can be disciplined, but needs an approximately light dipping topography. Additionally, it commonly involves lengthy transportation distance in the beginning years of the mine. This method is applicable where the base circumstances are weak [6].

Each mine waste dump has specific issues and have to be figured out in an individual and recognizable examination.

Two exterior issues such as water and earthquake play vital roles in the security of the mine waste dumps. As a result, these two factors have to be precisely examined during analysis. Earthquake or seismic forces affect the location of the mine waste dump. The consequence of water is quite difficult to assess but a specific attention has to be paid to prevent water from infiltrating or entering into the mine waste dump [6].

2.2. Reporting of Mineral Resources and Reserves

It is really important to report the terms mineral resource and mineral reserve precisely and accurately while describing and classifying mineral resources. Otherwise, it is possible that costly misrepresentations and misunderstanding will appear [9].

2.2.1. Mineral Resources

A mineral resource is the combination or existence of components or material of basic financial attraction in or on the Earth's coating. Furthermore, the existence should be in such a shape and abundance that for future economic exploitation, an acceptable anticipation is available. In addition, the grade, location, quantity and additional features of a mineral resource are recognized and calculated from a particular documentation and information containing sampling [15].

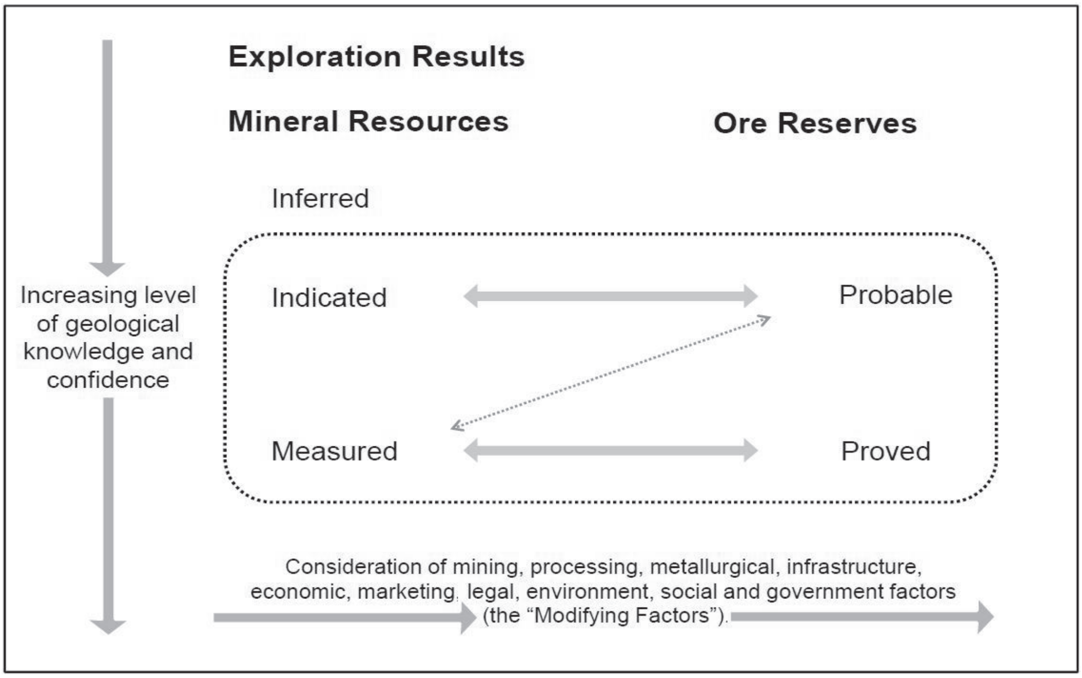

Figure 2.2-1: Classification of mineral resources and reserves [16]

Mineral resources are classified into three sub categories such as Inferred, Indicated and Measured resources considering the increase of geological knowledge and certainty [15].

An Inferred Mineral Resource is a share of mineral resource for which the grade and amount are calculated based on insufficient geological knowledge and information. Inferred mineral resources have lower certainty and reliability than indicated resources and could be improved to indicated resources with progressed exploration [15].

An Indicated Mineral Resource refers to that part of a mineral resource for which quantity, quality, density, shape and other physical features are calculated based on reliable geological knowledge. Furthermore, it authorizes the application of modifying factors in satisfactory analysis to back mine planning and economic assessment of the deposit. An indicated mineral resource has lower certainty than measured mineral resource and could be improved to a probable ore reserve [15].

A Measured Mineral Resource refers to that share of a mineral resource for which the quality, quantity, density, shape and physical attributes are calculated with high reliability.

Moreover, it lets the application of modifying factors to back mine planning and economic assessment of the deposit. A measured mineral resource has higher certainty than indicated and inferred mineral resources and could be transformed to a proved ore reserve or under confident situations to a probable ore reserve [15].

During reporting of mineral resources calculation, the terms ore or reserves should not be used because they refer to economic viability and technical feasibility. They are only applicable and compatible when all important modifying factors have been analyzed [9, pp. 670-715].

2.2.2. Mineral Reserves

A mineral reserve is that share of a mineral resource for which tonnage and grade is calculated and its extraction is cost-effective after the application of modifying factors. Mineral reserves are classified into two sub categories considering the modifying factors: *probable mineral reserves* and *proved mineral reserves* [15] [9, pp. 670-715].

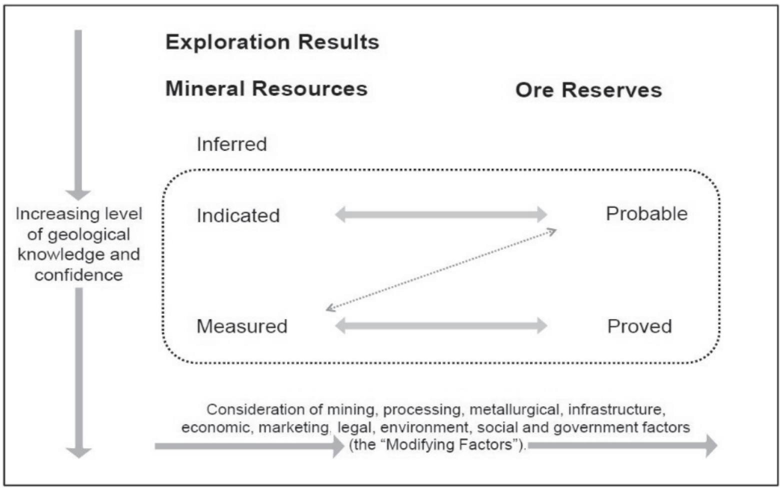

Figure 2.2-2: Classification of mineral resources and reserves [16]

Probable mineral reserves are that portion of indicated or measured resources which are economically mineable with a lower confidence of modifying factors than proved ore reserves. It comprises diluting materials and amount of money for losses which may arise when the material is extracted.

Proved mineral reserves are that part of measured mineral resources which is economically mineable. Proved mineral reserves show the highest level of confidence. [15] [9, pp. 670-715].

Public reports should indicate probable or proved or both of the classes. Furthermore, the reports must not include mixed probable and proved reserves except that the compatible numbers for each of the particular classes are also specified [9, pp. 670-715].

2.2.3. Resources Estimation

The big problem in mineral resources evaluation is the forecast of the tonnages and recoverable ore with a specific mine planning. A reasonable method to overcome this problem is to evaluate the grade for quantity appropriate to mine plan and the grade and other features should be evaluated in the un-sampled area. This estimation is complicated by the geological variability and there are various evaluation patterns planned for distinct objectives. The important objectives are to forecast grade and quantity of mineable material, certainty and reproducibility, and to declare huge geological trends [17].

There are various estimation methods available but the following three approaches are the common used methods for interpolation [17]:

1. Classic Polygonal Method
2. Inverse Distance Weighting
3. Kriging

Classic Polygonal Method

This method is established on appointing the impacted areas around drill holes interrupts. The polygons are drawn between a drill hole and its neighbor at right angles bisectors- is a line for which each point has the equal distance from either side of the line, see Figure 2.2-3. This idea can also be applied to three dimensions, while polygons are drawn in two dimensions [17]

Outer edges samples are too tricky because they are not enclosed by any other sample and the final result will be highly influenced if they are not restricted correctly [17]. This approach can be used to give an evaluation of the average grade [17].

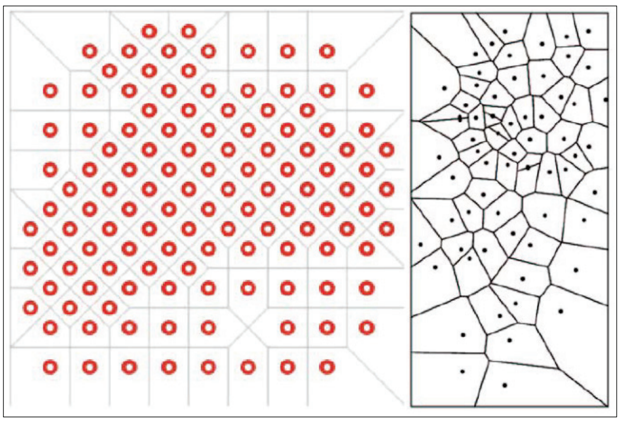

Figure 2.2-3: Two schematic examples of the polygons of influenced method; no distance units are given [17]

Inverse Distance Weighting (IDW)

This method is one of the first insertion approach established on experimental examination that the weight of each sample is equivalent to the opposite exponent of the length from the position of the approximate to the sample. The inverse distance weighting can be calculated using Equation (2.2-1) [18].

$$wi = weight\ for\ sample\ i = \frac{di^{-p}}{\sum di^{-p}} \qquad (2.2\text{-}1)$$

Where, d_i is the length between the sample i and the position being calculated and p is the weighting exponent [18].

The most usual powers used are 2 (Inverse Distance Squared, IDS) and 3 (Inverse Distance Cubed, IDC). Inverse distance cubed is used with flatly differing features, including topographic surfaces, thickness of geological units, strata bound deposits and in-situ insertion of largeness density. Inverse distance cubed is applied when closest samples are requested for big weights [17].

Kriging

Kriging method is used to compute the weights that decrease the variance of an anticipated error. On the other hand, kriging is used to calculate the rank of a point or block considering the medium weight of the neighboring points. This geo-statistical approach gives the best linear and unbiased calculation for grade established on a least-squares decrease of the calculation error and kriging error [17] [18] [19].

Kriging methods are over smoothed and the difference of calculation is less than the anticipated difference of SMU blocks. The grade distribution is discovered based on the difference of the calculated blocks if the calculation is impartial. Partial calculation or over smoothing drives an overestimate in quantity and decrease in grade.

There are various types of kriging approaches while in this chapter focus will be on Ordinary kriging (OK).

Ordinary Kriging is established on the equal lowest error difference discovered by linear calculation at a position where the valid or accurate worth is obscure. Kriging does not predict about the mean. As prediction, the obscure mean for the amount is calculated fixed as in equation (2.2-2) [5] [17] [18]:

$$z^*(u) = \sum_{i=1}^{n} \lambda_i z(u) + \left[1 - \sum_{i=1}^{n} \lambda_i\right].m \qquad (2.2\text{-}2)$$

$$[z^*(u) - m] = \sum_{i=1}^{n} \lambda_i\ .[z(\mathrm{u}_i) - m]$$

When the mean m is obscure, the $\sum_{i=1}^{n} \lambda_i = 1$ is the fair-mindedness case. Ordinary kriging is a mobile or movable method. By considering the various backing of the samples and blocks that are calculated, the source of the ordinary kriging is estimated as in equation (2.2-3) [17]:

By considering the weights and Lagrange multiplier as incomplete derivative,

$$\frac{\partial Q}{\partial \lambda_\alpha} = -2.\bar{C}(V, v_\alpha) + 2.\sum_{\beta=1}^{n} \lambda_\beta \, C(v_\alpha v_\beta) + 2.\mu = 0, \qquad \forall \alpha = 1, \dots, n$$

$$Q(\lambda_i, i = 1, \dots, n, \mu) = \sigma_E^2 + 2\mu \left[\sum_{i=1}^{n} \lambda_j - 1 \right] \rightarrow minimum \qquad \textit{(2.2-3)}$$

$$\frac{\partial Q}{\partial \mu} = \sum_{j=1}^{n} \lambda_j - 1 = 0$$

By adding Lagrange parameter μ due to the limits that the weights add up to 1, the derived ordinary kriging and the equivalent or matching ordinary kriging difference are calculated as in equation (2.2-4) [17]:

$$\begin{cases} \sum_{j=1}^{n} \lambda_j \, C(v_i, v_j) + \mu = \bar{C}(V, v_i), \forall \alpha = 1, \dots, n \\ \sum_{j=1}^{n} \lambda_j = 1 \end{cases}$$

$$\sigma_K^2 = \bar{C}(V, V) - \mu - \sum_{\alpha=1}^{n} \lambda_\alpha \, \bar{C}(V, v_\alpha) \qquad \textit{(2.2-4)}$$

2.2.4. Block Modelling

A block model (BM) is one of the most important element in production planning and is the foundation for all computer aided pit designs. A block model is a rectangular in shape, quite big to cover the deposit and the huge block is then detached into smaller rectangular blocks and as a group, all these blocks depict the area of attraction [5] [20].

A block model must contain all mineralized body in the area of attraction, topography, and the encircled area that the most admirable pit is included in the model.

A group of features such as tonnages, grades, and other information related to the materials is accredited to each block during the modelling action of the resource. These features affect the following key parameters [5] [20]:

- Quantity of product
- Price of product
- Expenses for mining, processing and transportation

- Time consumed during the process of bottleneck
- Flow of ore or waste
- Pit slopes

The accreditation of characteristics to each block is influenced by the ways of different interpolation methods and there are three specific methods including [20]:

1. Polygons method
2. Geo-statistics using Kriging
3. Inverse distance weighting methods

Data collection and modelling can be influenced by time, money, and technology so, practically the data added to each block will be less than ideal [5].

In practice, among the different kinds of block model, the 3-D fixed block model is the most common used model. The elevation of the block model is equal to the height of the bench and horizontal format is rectangular or square. The particularity of 3D fixed block model is the equality of dimensions in blocks [20].

Blocks may be behaved as discriminating part during planning and as well they depict the tiny part consumed to model pit slopes. Furthermore, accuracy of the pit slopes modelling depends on the size of the blocks. The smaller the blocks the accurate the pit slopes modelling. In order to do a reasonable block size planning following instructions should be considered [5]:

- Adjust the height of the block equivalent to the height for mining.
- Adjust the width and length of the block equivalent to block height due to the likely pit slopes.

Rule that it is wise to follow, the lowest size of block must not be smaller than ¼ of the moderate drill-hole break, for example 15 meters blocks for 60 meters drilling grid and 60 meters for 240 meters drilling grid [21].

Pit slopes with an angle of 45° to 55° must have block heights and widths completely equivalent to the block height. To find out the optimal width or length of a block in pit slopes with different angles than the above-mentioned extent, the following equation (2.2-5) is used [5]:

$$W = \frac{1.2H}{tan\theta} \qquad (2.2\text{-}5)$$

Where, H is the height of block and θ is the average pit slope.

To pursue geological boundaries carefully during modelling, sub-blocks or parcels are created. Sub-blocks are appointed with its own grade and tonnage features with a descripted position and volume in the ancestor block and each depicts a part of the block while a parcel does not have a determined position within the block. A group of parcels are created for each block to depict the distribution of grade [5].

2.3. Price Prediction

There are two methods used for price anticipation; *trend analysis* and *econometric models,* but in this study an economic model is briefly explained below [9].

Econometric Models

A product model is a measureable depiction of a product market or corporation. The behavioral friendships contained reverse supply and demand features of price establishment, also other linked economic, political and social phenomena. There are various types of approaches used for modelling mineral markets and corporations. The methods selected for a model rely upon the specific economic behavior of importance. Each method focuses on various features of describing history, evaluating policy and anticipation. It could be price establishment, reserve and inventory impacts or other impacts [9].

The market model comprises the following aspects:

- Product demand, supply and prices
- Prices of replaced products
- Price lags
- Product stocks
- Revenue or action degree
- Technical aspects
- Geological aspects
- Policy aspects affecting the supply

Market models, which adjust supply and demand to generate equilibrium price are usually applied in the mineral enterprise to forecast historical description, to decide for policy investigation and anticipation [9].

2.4. Estimating Mining Costs

The costs incurred in a mining project differ and can be documented in various ways. Costs are grouped in three main classes such as capital cost, operating cost and general and administrative cost [9].

The capital costs are those costs that are needed for the mine and mill plant. The costs which are needed per ton basis such as drilling, blasting, etc. costs are called operating costs. The general and administrative costs refer to yearly expenses and may involve head office expenditures, mine office expenditures, labourer benefits, area supervision, etc. [9].

The operating cost can be documented by various system activities such as drilling, blasting, loading, etc.

There are definite costs which are noticed as fixed costs which do not depend on production level. Variable costs rely upon production level. Costs can be incurred against the ore, against the waste, or against both.

For machinery, the ownership costs can be divided into depreciation and annual average investment cost.

During explaining, estimating, and presenting costs a care attention should be paid to describe what is meant and involved [9].

Chapter 3. Background

In this chapter, general copper market analysis including mineral price and demand and supply were explored. Furthermore, location, infrastructures, climate, access, exploration status and geology of the study area have been discussed.

3.1. Copper Market Analysis

Copper was one of the first metals exploited and used by mankind. Copper was firstly used in coins and ornaments starting at 8000 B.C and 5500 B.C and is a metal of choice for different types of domestic, industrial and high-technology utilizations today [22].
The symbol of copper is Cu, and its atomic number is 29. Copper is the best pure metal amongst others, purple in color, anti- abrasive, slightly hard, ductile, anti-corrosion, stable in air and its density is 8.92 g/cm^3 [2].

Copper is broadly used for electricity, construction, light industries, industrial machinery, transportation vehicles, and some other industries. Mainly over 50% of copper is used in electric and electronic industries [22] [2].

3.1.1. Resource Condition

Copper resources are broadly spread in the world. The largest copper reserves are located in Chile, Peru, USA, Mexico, Indonesia, China, Poland, Australia, Zambia, Russia, Canada and Kazakhstan.

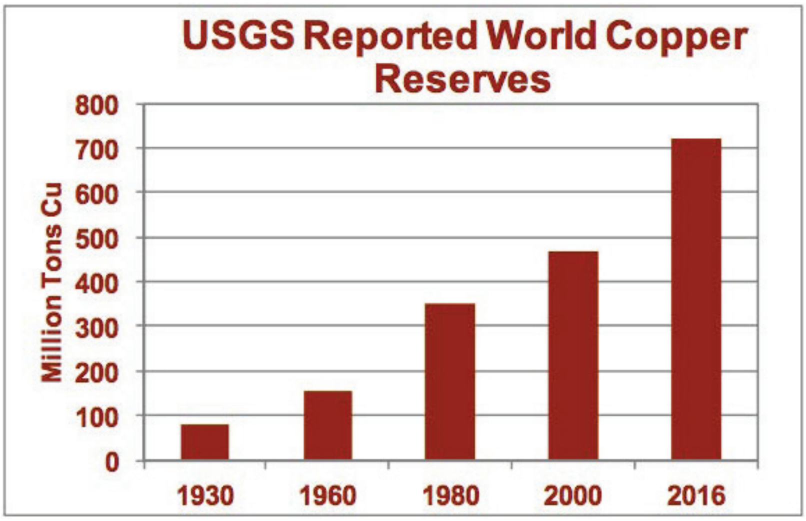

Figure 3.1-1: World copper reserves [23]

Generally, Figure 3.1-1 and Figure 3.1-2 illustrate that the world copper reserves are nearly rich and the estimated reserves were around 720 million tons in 2016. 33% of which were accounted for measured reserves. Moreover, the world copper mines production per year is 19.4 million tons and the life of existing reserves is estimated about 37 years. Chile and Peru represent above 29.1% and 11.25% respectively.

	Mine production		Reserves[7]
	2015	2016[e]	
United States	1,380	1,410	35,000
Australia	971	970	[8]89,000
Canada	697	720	11,000
Chile	5,760	5,500	210,000
China	1,710	1,740	28,000
Congo (Kinshasa)	1,020	910	20,000
Mexico	594	620	46,000
Peru	1,700	2,300	81,000
Russia	732	710	30,000
Zambia	712	740	20,000
Other countries	3,800	3,800	150,000
World total (rounded)	19,100	19,400	720,000

Figure 3.1-2: World copper mine production and reserves [24]

Among the different types of copper deposits, Porphyry deposit represents 53.5%, sedimentary and sedimentary-metamorphic deposits account for 31%, volcanic rocks pyrite deposit accounts 9% and the remaining 6.5% are related to other types of the copper deposits [2] [24] [23].

[8] For Australia, Joint Ore Reserves Committee-complaint reserves were about 24 million tons [24].

3.1.2. Copper Supply and Demand

The production and sales of copper have been increased since 1950s and showed an upturn. The increase rate has been decreased in 1970s but during the last 40 years (1960-2000), the production of the world copper mines and smelters grew constantly and the average annual increase rate was 2.89% and 2.73% respectively. The proportion of smelting was 10% higher than the mining capacity. The consumption of copper has been also increased and the annual growth rate was 2.93% respectively.

Table 3.1-1 World supply and demand balance, unit 10^4 t (rounded) [2]

Description	2000	2001	2002	2003	2004	2005	2006	2007
Output	1482	1567	1535	1522	1585	1661	1744	1798
Increase rate %	2.4	5.8	-2.1	-0.8	4.1	4.8	5	3.1
Consumption	1519	1468	1505	1532	1666	1676	1707	1804
Increase rate %	8.1	-3.3	2.5	1.8	8.7	0.7	1.8	5.7
Supply and demand balance	-38	99	30	-9.7	-81	-15	37	-6.4

Table 3.1-1 (continued)

Description	2008	2009	2010	2011	2012	2013	2014
Output	1848	1860	1918	1977	2044	2131	2298
Increase rate %	2.8	0.6	3.2	3.1	3.4	4.2	7.8
Consumption	1802	1835	1920	1950	2055	2104	2278
Increase rate %	-0.1	1.8	4.6	1.6	5.4	2.4	8.3
Supply and demand balance	46	25	-1.4	27	-11	27	20

After 1995, the demand for copper has been decreased because of the international economic environment which dropped the international market price as well. To cut down the economic loss, the world copper producers decreased the production. In 2003, the world economy has been restored and due to the high market demand, the supply was not quite enough therefore, the price increased. In 2004, the world copper producers tried hard to balance supply and demand but it did not work well therefore, in 2005 and 2006, there was a high increase of price in the world market [2].

Table 3.1-1 represents that in late years the supply and demand for refined copper did not have a stable balance. In 2007, the production growth rate of 3.1% was lower than the consumption growth rate of 5.7% resulting a deficiency in the supply of refined copper. In 2010, there was an increase in the production and consumption of refined copper and a basic equilibrium in supply and demand.

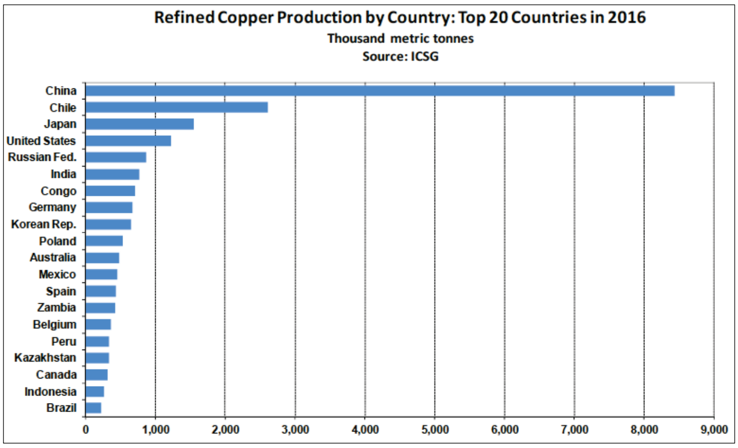

Figure 3.1-3 World top 20 refined copper production countries [23]

In 2005-2014, the refined copper consumption increase was as high as 3.5% in China and other developing countries. As illustrated in Figure 3.1-3 and Table 3.1-2, it can be said that china is the world largest producer and consumer of refined copper.

Table 3.1-2 World main refined copper consuming countries, unit 10^4 t (rounded) [2]

Country	2000	2001	2002	2003	2004	2005	2006
World total	1519	1469	1505	1532	1666	1676	1707
China	193	231	274	308	336	366	361
USA	302	262	236	229	241	227	213
Japan	135	114	116	120	128	123	128
Germany	131	112	107	101	110	111	139
South Korea	86	85	94	90	94	87	83
Italy	67	68	67	66	71	68	80
Taiwan	63	54	66	62	69	64	64
France	57	54	56	55	54	47	46
Mexico	47	43	38	35	39	44	40
Russia	18	22	36	42	53	79	79

Table 3.1-3 (continued)

Country	2007	2008	2009	2010	2011	2012	2013	2014
World total	1804	1802	1835	1920	1950	2055	2104	2278
China	486	513	714	742	791	884	983	1135
USA	218	191	170	178	175	180	183	184
Japan	125	118	88	106	101	99	100	108
Germany	137	138	112	132	125	109	113	117
South Korea	86	85	94	85	75	72	72	76
Italy	76	64	53	62	62	58	55	62
Taiwan	60	58	49	53	46	43	44	46
France	34	38	21	19	18	18	20	21
Mexico	30	31	34	32	23	40	33	32
Russia	69	73	46	42	68	68	48	57

3.1.3. Copper Price Analysis

While conducting feasibility study, it is quite important to do price analysis for a mineral. Generally, it gives an idea that what are the trends in prices. Moreover, it gives details of the historical price of the mineral and considering these aspects, the future price or price for the project can be approximated.

Current Mineral Price

There are various types of publications where current mineral prices can be found. There are various types of units in which the prices are expressed. Generally, the prices rely upon quality, quantity, source, form and packing.

There are three different types of tons that are used for abundant minerals which are described below [9]:

1. 1 short ton (st) = 907.2 kg

2. 1 long ton (lt) = 1016 kg
3. 1 metric ton (mt or tonne) = 1000 kg

For most metals including copper the unit of weight is Newton (N).
There are two types of compromises made by a seller and a buyer that has a high influence on the price. In Free-on-Board (FOB) agreement the buyer has to pay all the transportation costs and in CIF agreement the cost, insurance and freight are admitted in the price [9].

US$/tonne for 3 November 2017

LME COPPER OFFICIAL PRICES, US$ PER TONNE

CONTRACT	BID (US$ / TONNE)	OFFER (US$ / TONNE)
Cash	6914.00	6914.50
3-months	6940.00	6945.00
Dec 1	7000.00	7010.00
Dec 2	7020.00	7030.00
Dec 3	7025.00	7035.00

Figure 3.1-4: *LME copper currently price* [25]

Figure 3.1-4 and Figure 3.1-5 depict the current London Metal Exchange (LME) bid and offer prices for copper and the current copper offer price is 6914.50 US$/tonne [25].

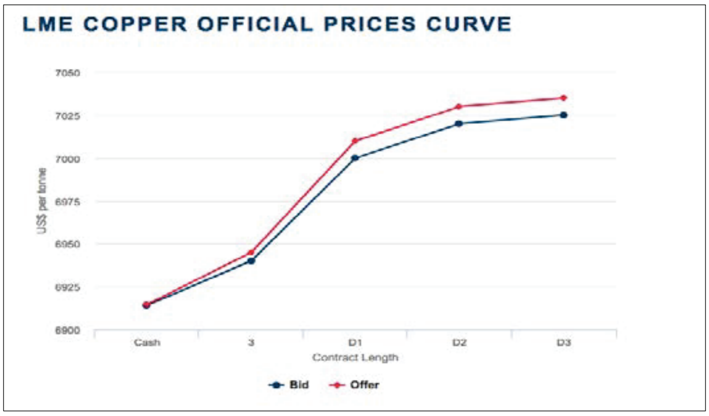

Figure 3.1-5: LME copper prices curve [25]

<u>Historical Price Data</u>

Considering the mineral prices that have been controlled over a time period of countless years, present a common skyward orientation. Nevertheless, this is not a stable growth with time but moderately, it is distinguished by recurrent variations. To specify the evidence of copper price uncertainty, the time span of 27 years has been considered and a line graph is plotted in Figure 3.1-6.

Figure 3.1-6: Copper historical prices for 27 years [26]

Line graph in Figure 3.1-6 shows that there have been too many fluctuations in the price of copper from 1990 to 2017. Generally, the price of copper has been raised to around 6900 US$/tonne comparing to 2200 US$/tonne in 1990.

Looking into line graph in Figure 3.1-6, using the end point value the average rate of price increase per year is 4.6% based on Equation (3.1-1) [9] [26].

$$r = (\frac{fv}{pv})^{1/n} - 1 \qquad (3.1\text{-}1)$$

Where,

r = average rate of price increase

fv = future price

pv = past price

n = duration

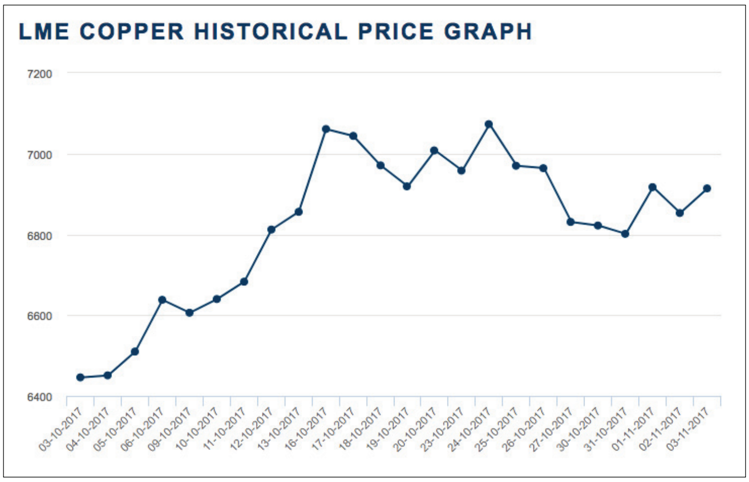

Figure 3.1-7: *LME copper historical prices for 22 months* [25]

Figure 3.1-7 also shows the LME historical prices plotted for 22 months. Furthermore, this graph represents the fluctuation in copper prices in as short period of 22 months.

As it is clear that mining project is a long-term project and may continue for some years to a few decades. A huge investment is needed to deliver a mine production and is reclaimed from the earnings achieved over a mine life time. The earnings are apparently and fully vulnerable on the mineral price. If the real price is fewer than that predicted, severe loss would be happening. Reclaiming investment would be risked declaring nothing of income.

In order to do the assessment estimation, two important problems occur. First issue is that which price must be used as base price and second is the prediction of future price history.

Using current price as base price is a poor decision and for further information reader is referred to Open Pit Mine Planning and Design, 3rd edition (Hustrulid and Kuchta, 2013). Another option could be to apply the current price history for the last two or five years [9].

Copper Price Determination for the Deposit

Considering the above specified situation of supply and demand in the copper metal market and copper market price attitude analysis, and historical price attitude, the copper metal price decided for the assessment of the deposit is 6000USD/t [2].

3.2. Location

The Aynak Copper Deposit is located in Logar province, 22° south east and 29 km far from Kabul. The north latitude and east longitude values are 34° 15′ 58″and 69° 18′ 12″ respectively. The average elevation is 2400 meters [27].

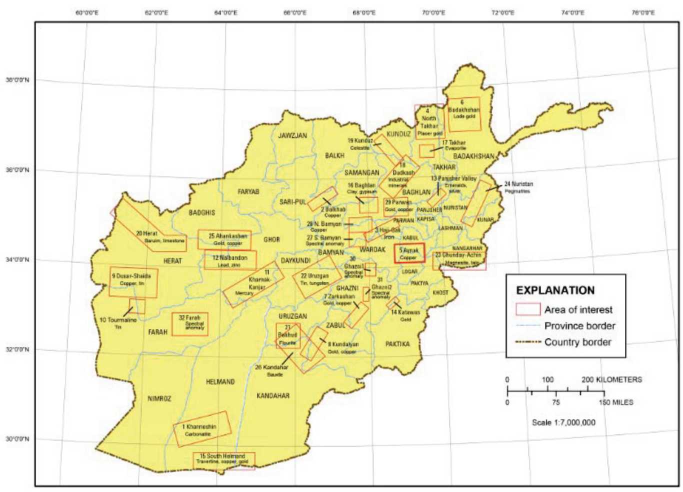

Figure 3.2-1: Location of Aynak Copper Deposit [28].

3.3. Accessibility

The accessibility to Aynak copper deposit is maintained by an asphalt highway from Kabul-Gardez road to the town Zaidabad which is around 30 km south of Kabul and then the remaining 16 km way is accessed by a gravel road to the camp site. These roads are available for the traffic throughout the whole year [27] [29] [30].

3.4. Climate

The climate of Aynak area is featured by warm summer and medium breezy winter. In the autumn and spring the precipitations form rain and in winter they form snow. The thickness of snow in winter is around 0.5 m and at depth of 1.6-1.7 m the earth temperature is nearly constant. Annually, the medium quantity of precipitations aggregates to 229-197.2 mm. In winter (from November to April), the precipitation amounts to 202-180 mm or it reaches to the 88-91% of the entire year precipitation. The highest quantity of precipitations reaches to 18.3 mm per day [30] [31].

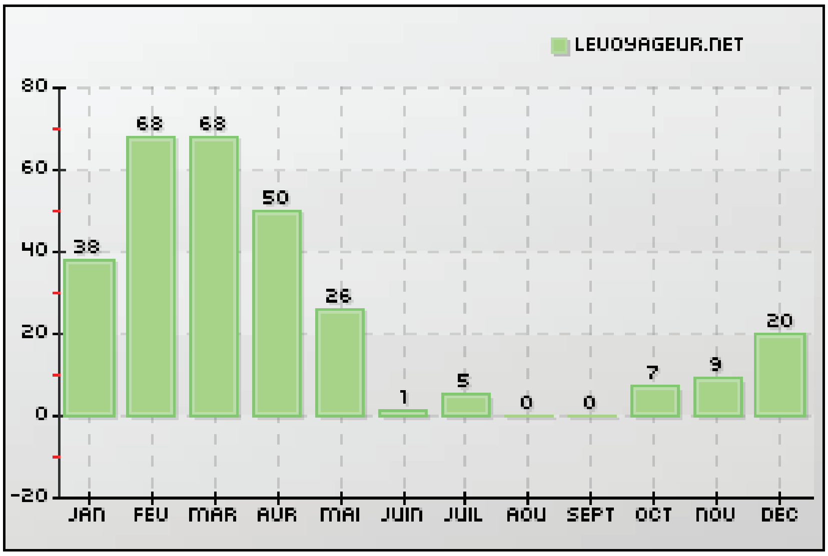

Figure 3.4-1: Average rainfall of the area in mm [32].

Medium yearly temperature is around +10 to +12°C. In winter (December to February) the monthly medium temperature reaches to -5 to -7°C while the minimum temperature can go down to its lowest amount around -42°C. In summer (June to August) it reaches to +23 to +24°C while the average high amount of temperature can amounts to +36°C [30] [31].

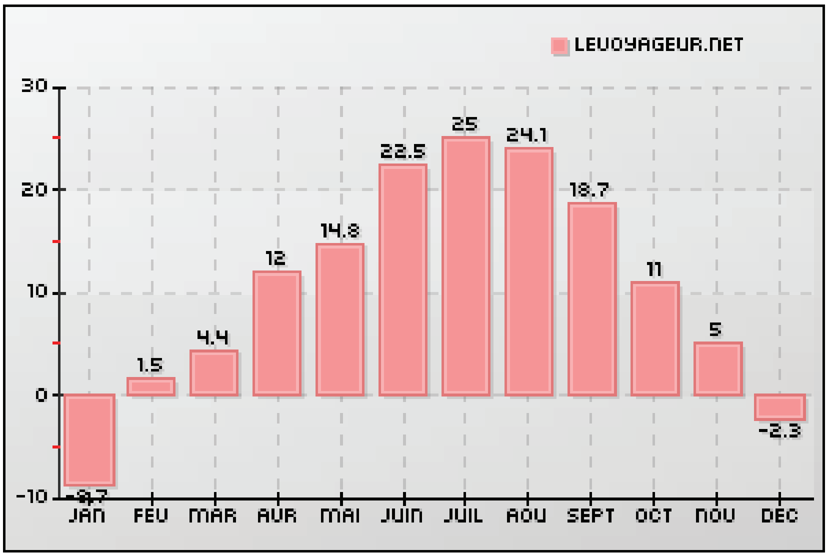

Figure 3.4-2: Average temperature of the area in C° [32].

The average humidity in winter is around 64-52% and in summer it amounts to around 41-30% and the amount of yearly evaporation from open water surface in the Kabul area amounts to around 1,610 mm [31].

The principal direction of wind in winter is western and south-western but in summer this direction goes to northern. The velocity of wind hits its peak to around 25-30 m/sec [30].

3.5. Infrastructure

3.5.1. Water

Logar river is the main river in the area and is the main source for the site. It is situated in the west, around 15 km far from the site area. The width of the river is 15.5 m-22.5 m and the catchment area of the river is around 9,772 km^2. The river flows from south to north generally and revitalized in the east by a number of mountainous rivers and streams- related to seasonal stream. The lowest flow rate is 1.5 m^3/s from July to September and the highest flow rate is 149.4 m^3/s from May to October and the highest flood level altitude of 1,826.83 m is reported in 1951. Annual flow of the river is 315 million m^3. The water of river does not freeze in winter [2].

Logar River water is of carbonate-magnesium type, with a mineralization of 0.4 g/l – 0.6 g/l. Furthermore, the water hardness is 3.9 mg/l – 6.8 mg/l. and PH value is 8.4-8.65.

Aside from Logar River, there are other seasonal water sources and revitalized by snowmelt and rainwater. Usually the volume of water is not extensive but occasionally during abundant rainstorm mud-rock flow is formed [1] [2].

3.5.2. Ground water

According to MCC Hydrological Investigation report, ground water was found by drilling method in all water-bearing layers and rocks except Trias and Proterozoic era. Main feature of ground water in Alluvial-Pluvial Aquifer is that this layer is influenced by the changes in water level and flow of Logar river. In conformity with the distribution area of the Aquifer, it can be divided into various kinds: *close to river*, *composite*, and *terrace.*

Ground water in diluvium in entirely free from the impact of surface water. Changes in ground water level is influenced by atmospheric precipitation and permeability of subsurface runoff [33].

Ground water in Alluvial-Pluvial Aquifer is recharged by rainfall, percolation of irrigation, subsurface runoff, and river water and discharged by evaporation, plant transpiration and drainage via downstream riverbed.

Quaternary Alluvial aquifer located in Zaydabad village, 15 km far from mining area is the only unconfined aquifer with a good water yield and permeability. Its width is 3-4 km and average thickness is 30-50 m and permeability coefficient is 60-318 m^3/d. The recoverable reserve is estimated around 172,800 m^3/d which is equivalent to 2 m^3/s [1].

Water extraction in dry seasons affect the ground water level from 6.71-12.21 m and according to calculations based on 112,000 m^3/d, 80,000 m^3/d and 40,000 m^3/d, annual surface runoff will decrease by 14%, 9.7% and 4.8% respectively [1].

During the investigation from June 1976 to October 1976, change of flow rate differed between 6,048 m^3/d and 127,008 m^3/d. Ground water mineralization ranges from 0.3-1.2 g/L and is mostly carbonate-magnesium type. The hardness of groundwater is 4-10 mg/, but frequently 6-8 mg/ and PH value is 8.4-8.65 [33].

3.5.3. Traffic and Transportation

Afghanistan is a landlocked country. Import and export of all merchandize from offshore are done predominantly through the seaport of other countries and then transported via roads to Afghanistan. In the meantime, all goods are principally transported by roads, moreover, there are roadways open from the capital city Kabul to all other cities in the country [2].

Aynak copper deposit is around 31 km straight-line far from Kabul-the capital city that has been accessed through several highways- which are available through the entire year. Presently, leaving the mine for Kabul is available through a 15 km Gravel road to the Kabul-Gardez highway and then driving 34 km of distance toward the north on this highway. Driving on this highway toward south, one can reach the border between Afghanistan and Pakistan. This asphalted double-lanes highway is in good condition for traffic [2].

3.5.4. Power Supply

Currently, there is a shortage of internal power supply in Afghanistan. The available small hydraulic and thermal power stations do not meet the requirements of the country because they have a limited capacity of power production. As a consequence, the Aynak copper deposit is located in the vicinity of the capital City-Kabul, so there will be no supply of electricity to the mine area in the present and nor in the future expected years. Building a new power plant for the project is the only solution that will rectify the power supply problem [2].

The available energy sources for electricity production in Afghanistan are heavy oil, coal and hydraulic power. Building a hydroelectric power plant in a short-term will not be realized because it requires high cost and long period for construction and includes in land occupation environment and other factors. Coal-fired thermal power plant is also not the possibility for generating power for the project because it requires exploring coal and developing coal mine, in addition it also requires high costs and long-period of construction. Furthermore, the distance of Aynak copper deposit is quite far from the coal mine and is around 280 km, which needs enormous working and cannot be completed in short-period and will have difficulties in maintenance [2].

Considering the above-mentioned information, it can be concluded that the best option for the project is a heavy-oil power plant [2].

3.5.5. Supply of Raw Materials and Fuel

The main raw materials including rolled steel, cement, stone and sand etc. are the necessary materials for construction of the project. The main raw materials needed for production are the spare parts for machinery and reagents of different types used up during mining and mineral processing, etc. are drills, bits, drilling rods, shovel racks, steel wires, lime, limestone, etc. and the fuel mainly includes different types of fuels and coal [2].

Currently, in Afghanistan, there is no huge company, so the formation materials including steels and cement should be imported from abroad. For sand and gravel, there are some possibilities available inside the country such as buying in the market or processing locally [2].

Spare parts used for general purpose machinery can be bought inside the country or in neighboring countries but the spare parts used for special machinery must be ordered or bought from manufacturers or machinery acquired places [2].

General usable can be obtained regionally or adjacent countries but specific tangibles should be bought or ordered from special makers [2].

Limestone and quartzite can be supplied from regional market or self-run quarry [2].

3.6. Local Resources

The Aynak copper deposit is located near to Kabul, the capital city- with an extreme population, developed industries and comprehensive agriculture, which gives the possibility to hire people from there. Furthermore, there will be a possibility to hire unskilled labor from the close villages to Aynak deposit for the project. The main staff of the Aynak Copper Mine will be hired from Afghanistan because it is a new project but technicians, technical and management staff will be hired from abroad based on the requirements of the company [31] [2].

3.7. Regional Geology

Aynak copper deposit is predominantly surrounded by metamorphic, phyllites, slightly with garnets and amphibolites rocks of Cambrian age. These rocks are generally in contact with Precambrian metamorphic rocks on the north-east side of the area.
The Cambrian rocks are entirely in association with lower Permian limestone and dolomites. These lower Permian rocks are in exposure with ultra-basic and basic rocks, locally metamorphosed to serpentinites and to amphibolites on the south-western side of the area [27].

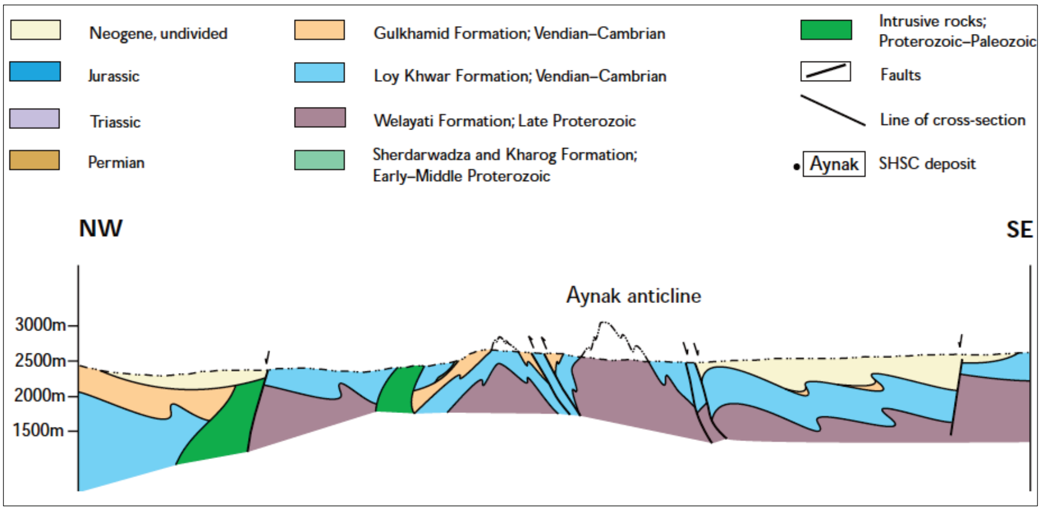

Figure 3.7-1: Cross section of Aynak area *[34]*

3.8. Local Geology

The central part of Aynak copper deposit is absolutely existed by moderate to steeply dipping, interbedded, carbonaceous shale, calcareous sandstone, sandy limestone and limestone. These rocks have been locally metamorphosed to argillite, micaceous schist, quartzite and marble. These metamorphosed sediments are governed by amphibolites which are likely amenable [27].

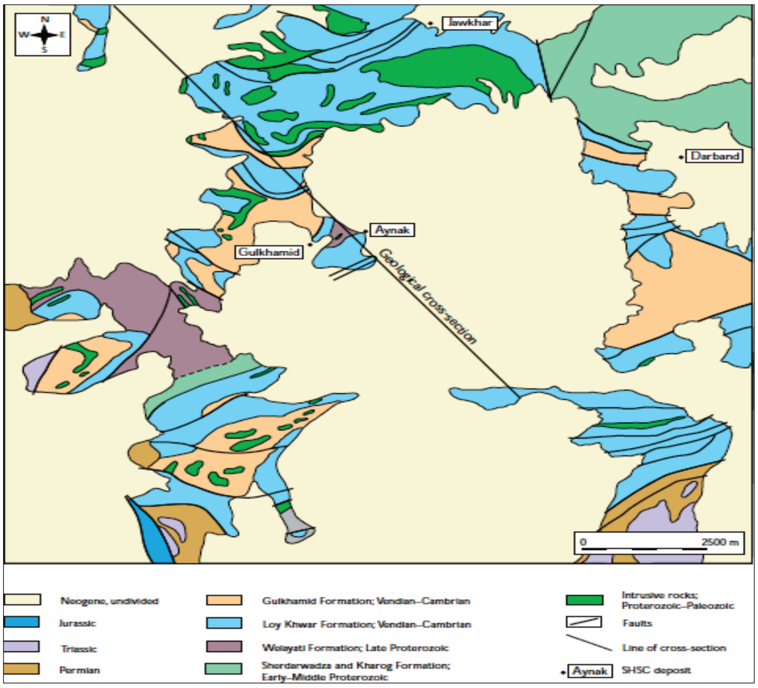

Figure 3.8-1: Simplified geological map of Aynak area [34].

In a nutshell, Aynak copper deposit within an area of around 40 km^2, exist of sedimentary and volcanic-sedimentary metamorphosed rocks of Precambrian to Cambrian age [27].

3.9. Mineralization

The copper mineralization at Aynak deposit is stratiform and hydrothermal-metasomatic and is mineralized as epigenetic and likely mesothermal. This mineralization is featured by chalcopyrite and bornite distributed in dolomite marble and schist of Loy Khwar Formation. At central Aynak, Bornite mostly features the main ore body. However, chalcopyrite characterizes trivially. On the other hand, at western Aynak, 80% of the mineralization is featured by chalcopyrite while bornite represents the remaining 20% of the mineralization [27] [35].

3.9.1. Zones of Mineralization

There are three zones of mineralization at Aynak copper deposit [27]:

1) The western
2) The central
3) The southern

But the most crucial part for this study is the central region and will be discussed [27].

Central Area

The Aynak copper deposit is mineralized in two fragments in the central area [27]:

1) The south-western fragment
2) The north-western fragment

1. The south-western fragment

This fragment of mineralization is outcropped with a strike of 800 meters and situated between portions 23 and 33. There is an interval of waste which disassociates the mineralization of two ore bodies with 0.5% of cut-off grade. The north-western-most ore body No. 1 which dips between 60° and 70° south-east is fundamentally perpendicular to the strike [27].

The ore body No. 1 which is 71 meters in width is disjointed by a moderate width of 44 meters from ore body No. 2, whose medium width is 212 meters [27].

2. The North-Western Fragment

This fragment is a Neogene covered, with a strike length of 700 meters and situated between portions 33 and 40. The depth of this portion along strike is around 10 meters in section 33 and 280 meters in section 40. At the maximum width of 700 meters the depth of the overburden is around 300 meters [27].

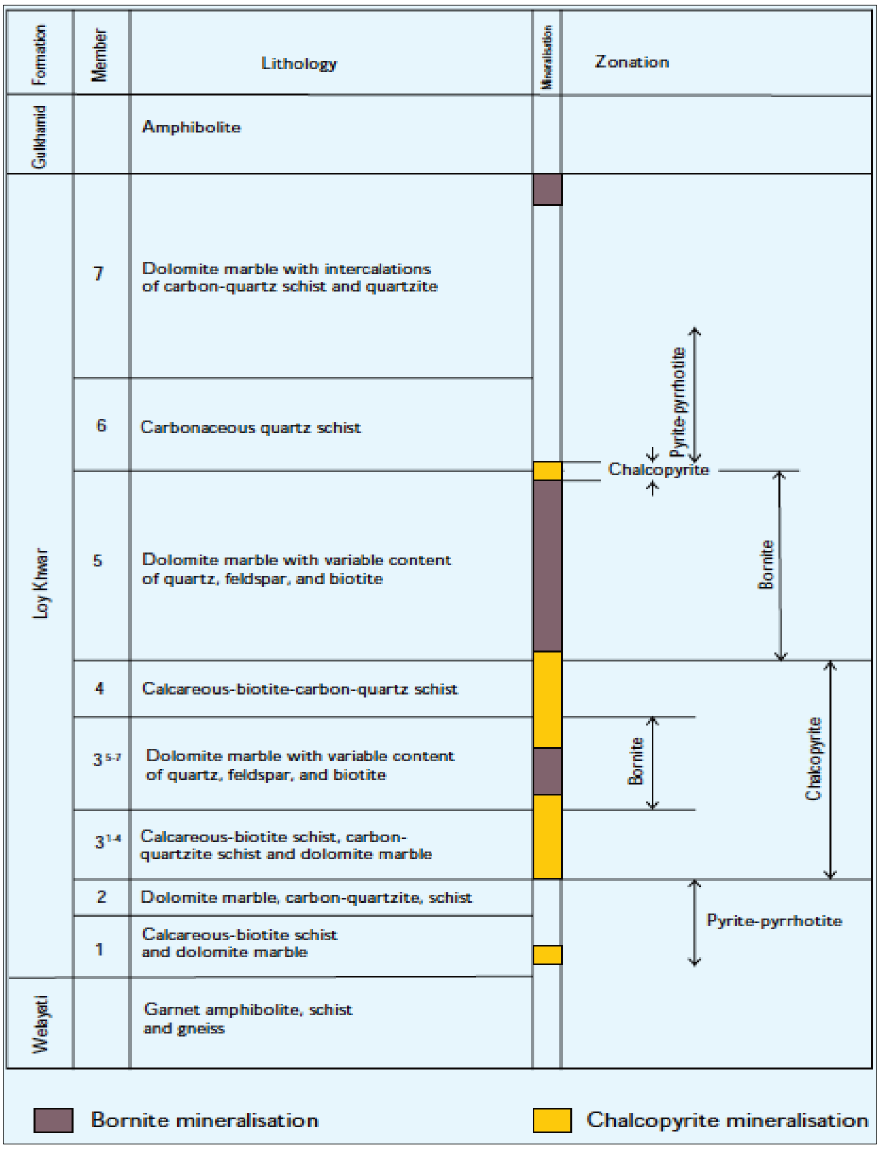

Figure 3.9-1: Simplified stratigraphic column shows major rock types and mineralization zones [35].

3.10. Structure

The deposit is located on the north-eastern part of a hill called Aynak, which is related to the one of the structures of the folded Baykal basement in the central part of Kabul tectonic block. [36].

The metamorphic bedrock of the area is comprised of three structural-formational complexes such as: *the lower*-uniting the Upper Proterozoic gneiss and schist incised by gabbro-amphibolite and keratophyre intrusions; *the middle*-comprised of Upper Proterozoic metavolcanics of differing combination, chipped by small intrusions of hornblende-gabbro and gabbro-diabase; *the upper*-depicted by Vendian-Cambrian copper-carrying rocks of the Carbonate-Schist Formation, building the cover of the epi-Baykal terrace [36].

To sum up, the structure is fairly complicated but may be elucidated as drag-folding of the sediments on a common monoclinal structure. It strikes north 20° east, dipping 50°-80° east and plunging North-Northeast at a low angle under the Neogene, where the structure arises to be flatten out. In brief, the structure of Aynak copper deposit is asymmetrical anticline, which length is 4 km and width is 2.5 km [27] [37].

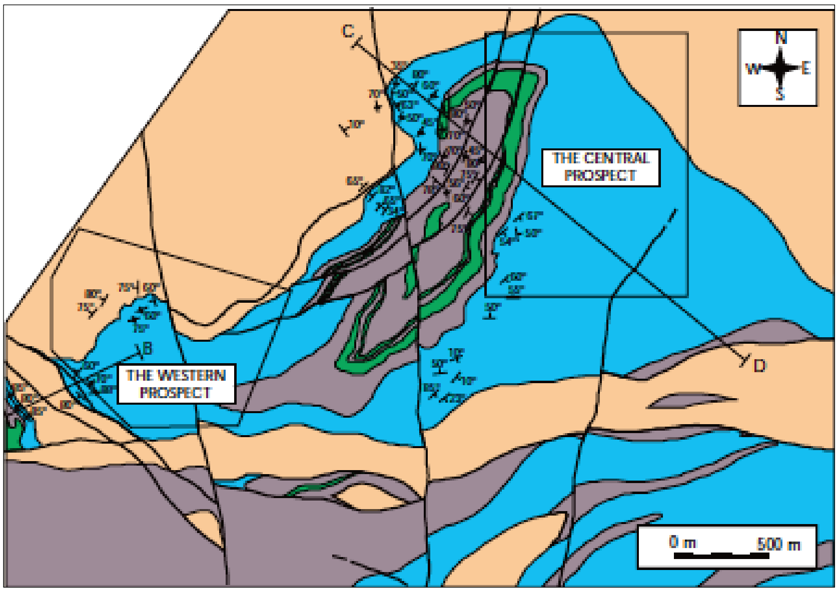

Figure 3.10-1: Simplified geological map the Aynak area [37].

3.11. Exploration Status

In the early 1960s, the Aynak copper deposit was found and then the investigation was accomplished in 1973. Field exploration was implemented in 1974-1987 and the Elementary Geological Exploration Report on Central Aynak and Elaborated Report on Central Aynak was finalized and complied in 1977 and 1978 respectively. Furthermore, the Elementary Exploration Report on Western Aynak was finalized and endured in 1987 [2].

According to the classification of the USSR State Mineral Reserves Committee, regarding the natural conditions of the deposit, the Aynak deposit belongs to the second most complicated group of deposits [31].

The deposit has been explored by core drilling, trenching and several underground exploration workings situated in parallel crosscuts and geological sections [31].

The assessment of exploration workings framework for different reserves categories is as below [31]:

B category: The distances between exploration sections should not go beyond 100 m and the distances between neighboring ore crossings should not surpass 75-150 m for sulphide ores [31].

C_1 category: Relying upon the complication of the ore bodies morphology, the distances between sections are accepted at 100 and 200 meters and between the neighboring ore crossings within sections are affiliated no longer than 150 m and alike at 75- 100 m only if the areas are morphologically complex [31].

C_2 category: Sulphide ore contours are constituted on the essential of single ore crossings and slightly on geologically approved predictions [31].

Until March1, 1977 the southern part of the main ore body has been entirely determined in plan and vertically between sections No XXIII and XXXIII and furthermore, the boreholes placed in southern part were investigated and delimited in detailed 4 tabular ore bodies and 4 extra deeply dipping ore bodies were slightly investigated [31].

Totally, there are 19 tiny lentil-shaped ore bodies of which 4 have been bisected by 3 to 8 contrary investigation workings and slightly determined in plan and the remaining ore bodies are bisected by isolated boreholes [31].

3.12. Drilling

Generally, by March 1, 1977 the central part of Aynak deposit has been investigated by 30066.7 meters of boreholes, 2785.3 meters of underground workings and 7880 m^3 trenches [31].

Until September 1, 1977 the central part was slightly determined from the west, south and east-south. Therefore, 24 further boreholes of 7560 m of length were drilled to get more information and upgrade reserves to categories C_1 and B and the assessment of boreholes consent to the USSR Mineral Reserves Committee [31].

The exploration which has been done by Russian, indicates that several hundred boreholes have been drilled. These boreholes also contain holes for hydrological and geotechnical analysis. The exact number of these boreholes is unascertained because of the defective mood of the AGS archive, but it is strongly accredited that the most of boreholes were drilled for mineral inspection and resource evaluation. In addition, there has been a possibility to find borehole logs for 150 of these [38].

By 1987, the drilling of 130,980.3 m was accomplished in Aynak copper deposit, 25,424 core samples were collected, 3,716.1 m of anticipating drift were completed and 26,871.9 m^3 of trenches were dug up. The part of Central Aynak from the above mentioned work was 59655.1 m drilling, obscure number of drill holes, 3015.1 m drift, 12,024 m^3 of exploration trench, 12,600 core samples, 6679 samples from trenching and for the Western Aynak 177 boreholes with a length of 67,244.2 m, 2 drifts distanced in 781 m, 3,578.7 m^3 of exploration trench, 12,824 core samples and 1914 samples from trenching [2].

3.13. Data Verification

With a view to data verification from former exploration work and to get further elaborated exploration data, MCC-JCL accomplished verification investigation and additional investigation for Central and Western Aynak in 2008-2011. These verification work includes 36 drill holes in length of 17,752.01 m, 2,872 samples for elementary inquiry, 323 samples for combined inquiry, in which, for the central Aynak verification investigating drill holes and additional investigating drill holes amounted to 6 and 9 respectively with a total length of 6,680.2 m, 775 samples for elementary inquiry and 104 samples for combined analysis. While for Western Aynak, 17 drill holes for additional inquiry and 4 verification investigating drill holes in a total length of 1,1071.81 m, 2097 samples for elementary analysis and 219 samples for combined analysis were accomplished [2] .

The exploration work for Aynak copper deposit was accomplished in 1970s and 1980s. According to the information recorded shows that the average recovery rate of ore for Central Aynak is 70.46% and 77% for ore-body territory. According to the grade of main ore-body which is greater or equal to 0.4 expresses that the average ore recovery rates for primary, mixed and oxidized ore are 71%, 64% and 63% respectively. Generally, it means that the rate of recovery for ore is on lower side [2].

In the Central Aynak, from the interior and exterior control examination of the chemical test results of the samples, it has been proved that both quality and accuracy of the ore are certified, while generally, in Western Aynak the medium error is within the acceptable range but in some samples the error is exceeding. Therefore, for resources and reserves calculation a coefficient method is applied for qualification [2].

The ore recovery rate is proved by MCC-JCL and furthermore, the quality and reliability of the ore are also approved and mentioned that both are satisfied [2].

To conclude, former exploration, verification and additional exploration have described the actualization, occurrence and spatial distribution of the ore body. The prospection grid in central Aynak is 100 m-150 m*100 m-150 m, and resource for attested level accounts 2/3 and in the open-pit area, the ore body prospection grade is gratified completely for the design requirement. Particularly, it has been attested by the verification investigation that the quality of the geological exploration is good [2].

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Chapter 4. Material and Methodology

The task of this chapter is to use the available data and apply a reasonable method to get appropriate results. A 3D geological model and a 3D block model (3DBM) have been generated to calculate mineral resources and reserves. In addition, pit optimization and open-pit design have been developed. The block model covers the mineral contents of Sulphide Copper (CTID) and Oxide Copper (COID). Generally, 132 drill holes were considered during modelling which are related to Central Aynak Deposit.

4.1. Work Methodology

Firstly, it was strongly required to check the drill holes' database to get trustworthy and verified data for the deposit modelling, resource and reserve estimation and production planning.

There were several problems and mistakes identified and corrected during the checking process. The database contains the drill holes related to both Aynak Central and Aynak Western deposits. After the examination of the data, 132 drill holes were recognized related to Aynak Central part and 18 drill holes to Aynak Western part. Working on Aynak Western has been ignored because of less available data and Aynak Central deposit has been chosen for the thesis. Then, the bench-height method was used to composite the copper samples.

A 3D geological model was generated from Cuto (Sulphide copper) and Cuox (Oxidized Copper) content using Minesight to represent the estimation domain. A 3D Block Model (3DBM) has been created and features and densities were assigned to the block model. Then, the 3DBM was used for reserve calculation and production planning purposes including optimization and open pit design.

The estimation methods used for block model are Inverse Distance Weighting (IDW) and Ordinary Kriging (OK). IDW routine was selected to calculate reserves. The reason for choosing this method was that during literature review, it has been discovered that IDW is the most common method used in various feasibility studies including MCC Feasibility Study Report of Aynak, technical report and preliminary assessment of the harper creek project and etc. [2] [39] [40].

4.2. Received Data

A Data base of drill holes was taken from the Ministry of Mines and Petroleum of Afghanistan (MoMP) at the inception of the thesis. As per confidentiality of the data, the data was only used for educational purposes.

This database was named as Aynak_boreholes.mdb and the structure of the database including several excel sheets was as follow:

Table 4.2-1 Structure of tblanalysis excel spreadsheet in database

No.	Item	Value	Description
1	BH_Number	100	Borehole identification number
2	Top_depth	101 m	Top depth of sample
3	Bottom_depth	103.5 m	Bottom depth of sample
4	Interval	2.5 m	Length of sample
5	Sample_No	115	Sample identification number
6	Cu_total	0.5%	Sulphide copper grade
7	Cu_ox	0.29%	Oxidized copper grade
8	Oxidation	50	N/A
9	Date_entered	27.04.05	Date when the data is entered into database
10	Entered_by	najeeb	A person who entered the data into database

Table 4.2-1 shows the structure of the data analysed in laboratory and named as tblanalysis.xlsx in the database.

Table 4.2-2 Structure of tblBorehole excel spreadsheet in database

No.	Item	Value	Description
1	Started date	03.07.74	Starting date of the bore hole
2	Finished date	05.12.74	Finishing date of the bore hole
3	X	92460,6	East coordinates
4	Y	28282	North coordinates
5	H	2443,7	Elevation
6	Azimuth	2443,7	Azimuth
7	Inclination	-65	Inclination or dip angle
8	Recovery	77	
9	Depth	330,5	Depth of the bore hole
10	Scan	#M:\Borehole_tiffs\n-1.tif#	
11	Date entered	Date_entered	Date on which the data entered into database

Table 4.2-2 depicts the format of data used for the boreholes and named as tblBoreholes.xlsx in the database.

Table 4.2-3 represents the information about CDs on which the important data have been righted while these CDs were not available for writing the thesis. This file has named as tblCD.xlsx in the database.

Table 4.2-3 Structure of tblCD file in Database

CD_number	BH_Number	File_name	Date_created	Created_by	Comments
1	1	Borehole -1.tif	06.12.04	USGS	
2	11	n-10.tif	14.12.04	USGS	This CD also contains the end of one BH? 2?
2	12-H	n-11.tif	22.12.04	USGS	
2	16	n-16.tif	19.12.04	USGS	
2	19-H	n-19.tif	22.12.04	USGS	
3	41-V	n-41tif.tif	22.12.04	USGS	
3	7	n-7.tif	17.12.04	USGS	

Table 4.2-4 represents the information about inclination and azimuth of the drill holes and has named as tblDownhole_azim_dip.xlsx in database.

Table 4.2-4 Format of tblDownhole_azim_dip file in database

BH_Number	Depth	Inclination	Azimuth
1	0	-65	290
1	20	-60,25	292
1	40	-61	300
1	60	-59	300
1	80	-60	302
1	100	-60,75	302
1	120	-61	302
1	140	-61,33000183	295
1	160	-61,5	290
1	180	-62,5	290

Table 4.2-5 depicts data about the geology of the deposit and has named as tblGeology.xlsx in the database.

Table 4.2-5 Structure of the tblGeology in the database

No.	Item	Value	Description
1	BH_Number	1	Bore hole identification number
2	Unit_no	1	Sample unit number
3	Top_depth	0	Top depth of the sample
4	Bottom_depth	12,4	Bottom depth of the sample
5	Thickness	12,4	Length of the sample
6	Age	N	Geological age
7	Lithology	Arenaceous argillaceous deposits	Lithology
8	Description		Description about lithology
9	Banding_angle	50	
10	Date_entered		Date on which data entered into data base
11	Entered_by		A person who entered data into data base

4.3. Data Examination and Verification

Data examination and confirmation were crucial to create an appropriate geological 3D model and estimate resource and reserve quantities. Therefore, after reviewing and validating the data, the structure and formats of the required data are as following.

4.3.1. Collar Table

The characteristics and structure of the collar ASCII file is recorded as below in Table 4.3-1. All the required and important features were included in this file to get a reasonable result.

Table 4.3-1 Structure of the collar file used to create PCF file in Minesight

Column Number	Name	Remarks
1	BH_Number	BH-Number is used for the hole identification
2	X	East coordinates
3	Y	North coordinates
4	Z	Elevation coordinates
5	Depth	Length of the drill hole (m)
6	Azimuth	Azimuth angle of the drill hole
7	Inclination	Dip angle of the drill hole
8	Started date	Shows the starting date of drilling of the hole.
9	Date entered	Shows the date of entering information into the database
10	Entered by	Shows who has entered the information into the database

Table 4.3-1 involves all the information about 132 drill holes with a total length of 51882 meters. All the drill holes have different depth.

4.3.2. Survey Table

The structure and features of the survey ASCII table are listed in the following Table 4.3-2. This file contains all the crucial and required components needed for the modelling.
Table 4.3-2 comprises information about 132 drill holes including length, dip and azimuth of the drill holes.

Table 4.3-2 Structure of survey file used to create PCF file in Minesight

Column Number	**Name**	**Remarks**
1	BH_Number	Identification of the drill hole
2	Depth	length of the drill hole
3	Dip	Dip or inclination angle of the drill hole

4.3.3. Assay Table

The structure and features of the assay ASCII file are listed in Table 4.3-3. This table compromises all the necessary and principal components of the deposit that are used for modelling purposes. Table 4.3-3 depicts information about 132 drill holes and 12471 samples. Samples were tested for sulphide copper (Cu_total), oxidized copper (Cu_ox) and oxidation.

Table 4.3-3 Structure of the assay file used to create PCF file in MineSight

Column Number	Name	Remarks
1	BH_Number	Identification of the drill hole
2	Top_depth	Starting length of the sample
3	Bottom_depth	Finished length of the sample
4	Interval	Interval or thickness of the sample
5	Sample_No	Number of the sample
6	Cu_total	Total quantity of sulphide copper
7	Cu_ox	Quantity of oxidized copper
8	Oxidation	The quantity of Oxidation

4.3.4. Geology Table

The structure and characteristics of the geology ASCII file are recorded in Table 4.3-4. This table shows the entire compulsory and crucial aspects of the deposit which were used for modelling to have an appropriate model of the deposit.

Table 4.3-4 Structure of the geology file used to create PCF file in MineSight

Column Number	Name	Remarks
1	BH_Number	Identification of the hole
2	From	The top length of the drill hole
3	To	The bottom length of the drill hole
4	Thickness	Thickness or length of each lithology in a drill hole
5	Lithology	Lithology of the drill holes
6	Lith_abr	An abbreviation assigned to each lithology to make it sample for understanding and also for the Minesight software because the software can read few letters.
7	Nr	It shows the number of each lithology.

Table 4.3-4 represents data about 132 drill holes and 2322 lithology for all the drill holes. Lithology were abbreviated into four letters because Minesight software can only read few letters. Moreover, they were abbreviated to easily understand each lithology during describing each drill hole individually.

4.3.5. Grouping of Lithology

The Lithologies were grouped into different groups according to their types, characteristics and geological formations. Grouping has been done to have small number of lithologies for better understanding. Furthermore, the MineSight software does not support large number of lithologies and large names. Therefore, the lithologies were abbreviated as well. The grouped lithologies and their abbreviations are shown in Table 4.3-5.

Table 4.3-5 Grouping of the lithologies based on their characteristics and geological formation

Lithology	Group	Lithology	Group	Lithology	Group
Arenaceous-argillaceous deposits	san	Quartz vein	qua	Quartz-biotite-carbonate schist	sch
Unconsolidated deposits	san	Quartz schist	qua	Biotite-dolomite schist	Sch
Sand	san	Quartz-feldspar-mica schist	qua	Biotite-quartz-dolomite schist	sch
Tectonic breccia	bre	Quartzite	qua	Quartz-biotite-dolomite schist	sch
Conglomerate	bre	Quartz-biotite schist	qua	Tremolite-dolomite schist	sch
Tectonic breccia	bre	Quartzite with quartz-feldspar-biotite rock	qua	Quartz-sericite-biotite schist	sch
Breccia	bre	Loam	loa	Quartz-biotite-dolomite schist	sch
Breccia	bre	Amphibolite	amp	Sericite-quartz schist	sch
Dolomite-quartz-feldspar rock	dol	Marl	cla	Quartz-dolomite schist	sch
Dolomite-quartz-feldspar rock	dol	Clay	cla	Sericite-dolomite schist	sch
Dolomite-quartz-feldspar rock	dol	Clay	cla	Sericite-biotite-quartz schist	sch
Dolomite-quartz-feldspar rock	dol	Slag	slag	Dolomite marble	mar
Biotite-hornblende gneiss	gne	Dolomite marble with quartz schist	mar	Dolomite marble with microquartzite	mar
Talus deposits	tal	Calcite vein	mar	Dolomite marble with sericite-quartz schist	mar
unidentified	x	Marble	mar	Dolomite marble with microquartzite	mar

4.3.6. Topography File

The original topography file was not provided along with the database. After downloading the topography file of the area of interest, the problem was that the drill holes were not located in the exact area not even in Afghanistan. The coordinate system used for drill holes was unknown.

To solve the above problem mentioned, reports including MCC Feasibility Study Report of Aynak Copper Mine 2015, Interim Report to the Ministry of Mines and Industries on the Aynak Copper Deposit Logar, Afghanistan by Watts, Griffis and McQuat Ltd, Toronto,1976, and Ainak Mining in the democratic Republic of Afghanistan-Feasibility study by State Institute for Designing of Non-Ferrous Metallurgical Enterprises, Moscow 1980 related to Aynak Copper area have been read but could not find useful information related to the coordinate system. The only possibility was the Ministry of Mines and Petroleum of Afghanistan and the ministry has been contacted to get information regarding the coordinate system. But unfortunately, after a while it has been told that they do not know about the coordinate system.

To figure out the problem, it could be a possibility that some digits are missing from the drill holes' coordinates and it has been assumed that the drill holes coordinate system is UTM 42 N. A topography file has been downloaded in UTM 42 N. After comparing the figures to the drill holes' coordinates, the correction figures of X = 50000 and Y = 3700000 have been discovered and added to the drill holes' coordinates to register the drill holes into the exact area of interest.

The next problem was to import the topography file into Minesight and the best option is to have a topography file in DXF or DWG formats. In this regard, a topography file was downloaded and changed into points in ArcGIS. Then, Surfer software had been used to create a DXF file. After that, the file was imported into Minesight and had given the possibility to create a 3D triangulation, confirmed and adopted to the project area.

4.4. Geological Modelling

To generate a geological model of an ore body using Minesight, the following steps were necessary and had considered:

4.4.1. Initializing PCF File

To start working on a project in Minesight Compass, a Project Control File (PCF) aynk10 and a Minesight Compass Project (aynk.prj) file have been initialized. A Minesight Compass project has been created from the existing PCF file.

4.4.2. Initializing Project Files

All project files including assay file (aynk11) and survey file (aynk12) have been created efficiently using Minesight Compass Project File Editor.

4.4.3. Initializing a Minesight 3D Project

A Minesight project called Aynak Copper Deposit Modelling has been created and then, PCF or project limits have been determined carefully and accurately. Because once the limits have been determined at the beginning of the Minesight Project, later the limits cannot be changed.

4.4.4. Drill Hole 3D Display

Using the CONSCA (**CON**vert, **S**urvey, **C**ollar, **A**ssay) program the data has been converted to the required format and then loaded into the Minesight for a 3D display. This 3D display provides options for colour, cut-off definition, labelling and some other options. Below Figure 4.4-1 shows the 3D display of Aynak Copper Deposit drill holes.

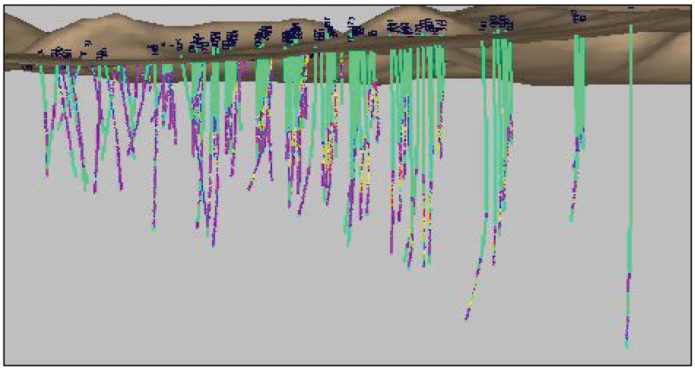


Figure 4.4-1: 3D display of Aynak drill holes

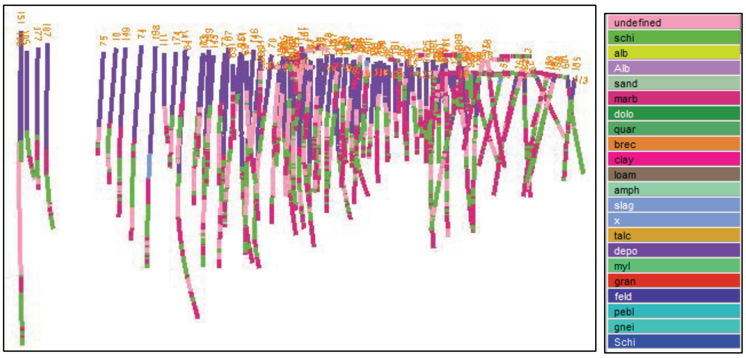


Figure 4.4-2: 3D drill holes view based on lithology

Figure 4.4-2 shows the 3D drill holes' view of drill holes in which the lithology's are grouped.

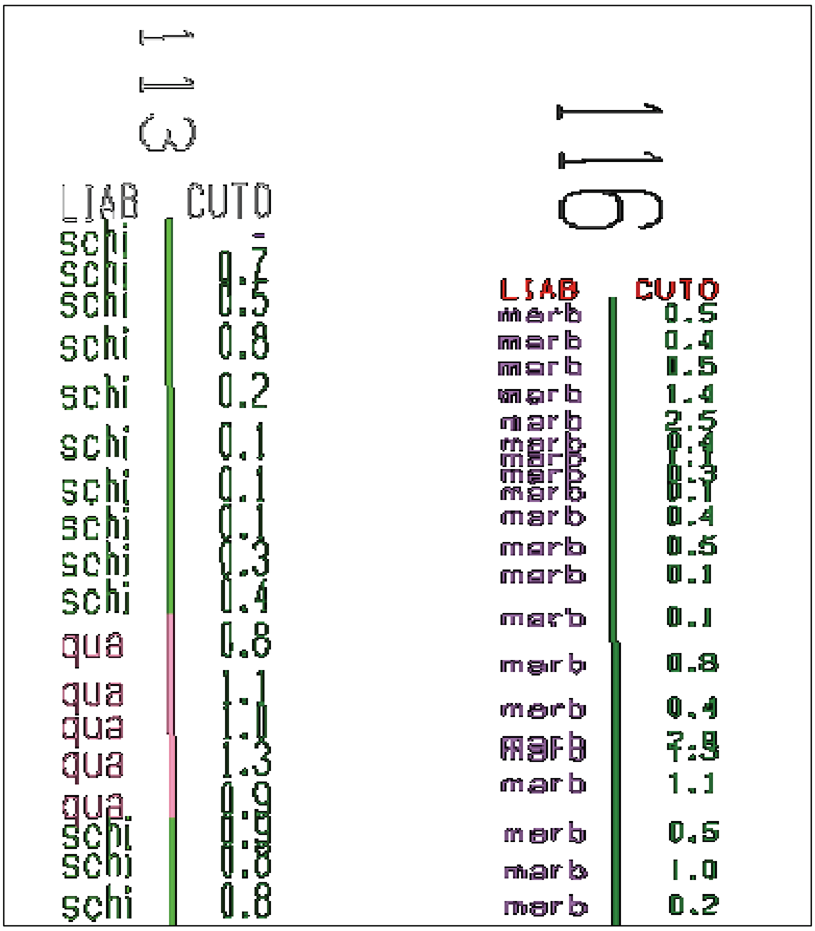


Figure 4.4-3: Drill holes 113 and 116 are shown in detail for better understanding

To illustrate drill holes in details, drill holes number 113 and 116 in Figure 4.4-3 are shown as the representatives for the entire drill holes. It shows the percentage of sulphide copper and the lithology of the drill holes mentioned.

4.4.5. Compositing

A method by which assay data are linked to make weighted average or composite grade representative of intervals longer than their own is called compositing. Bench compositing is a technique mostly used for resources modelling in open-pit mining today. For huge and uniform deposits, the compositing interval is the bench height and constant elevation. The causes for and the advantages of compositing includes that the uneven length assay samples should be composited to specify representative data for analysis. Furthermore, it decreases the number of data and needed time for calculation and incorporates dilution. It also decreases disordered variation due to massive or tiny assay values [9].

Considering the necessity and benefits of compositing, the assay data have been composited on bench height basis to decrease the amount of data used. Moreover, to decrease the calculation time and to prepare data for modelling and to provide constant assistance for geo-statistics [41] [42].

A file called aynk09 has been initialized using Minesight Compass Editor. Below is the 3D display of composites shown in Figure 4.4-4.

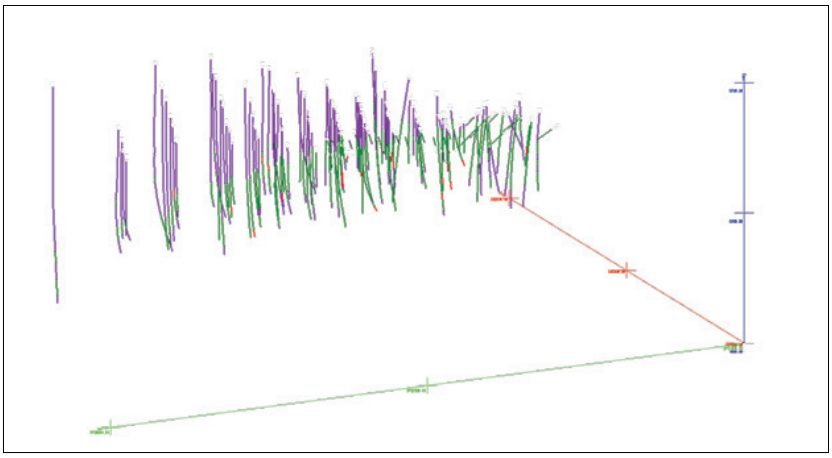


Figure 4.4-4: 3D composite display of drill holes

4.4.6. Grid Setting

Grid set was created by defining an origin, azimuth and dip. Grid set was used to step through data by restricting the view using volume clipping. Furthermore, it was used to slice the data for interpretation. Grid set shown in Figure 4.4-5 has been used in this research. For better realization, Figure 4.4-6 represents the function of the grid set using Minesight which has given the possibility to interpret the data easily [41] [42].

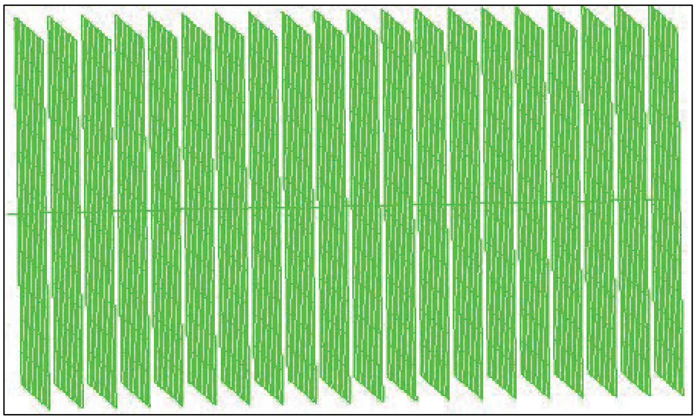


Figure 4.4-5 Grid set 3D view in Minesight

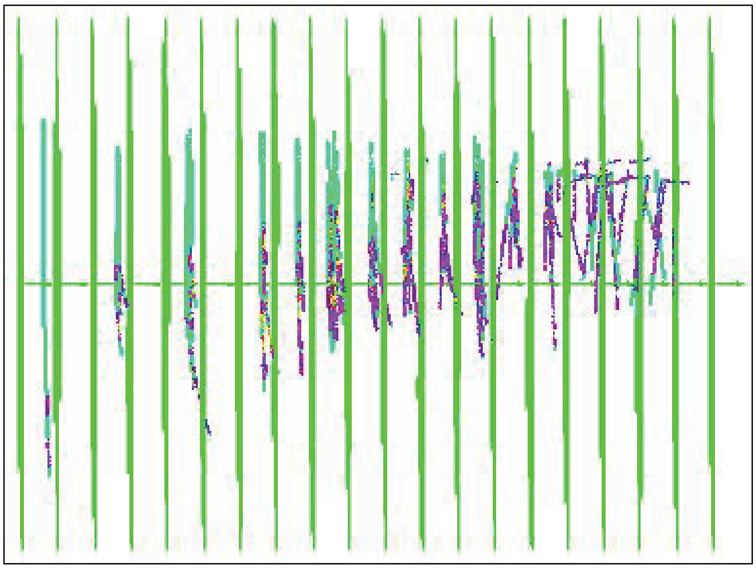


Figure 4.4-6: Grid set used to restrict the visibility using clipping volume for interpretation

4.4.7. Ore Body Modelling

The following polygons shown in Figure 4.4-7 were created using the smooth polygons option to generate the 3D model of the ore body. The polygons were created by connecting the mineralized body with a minimum grade of 0.1%.

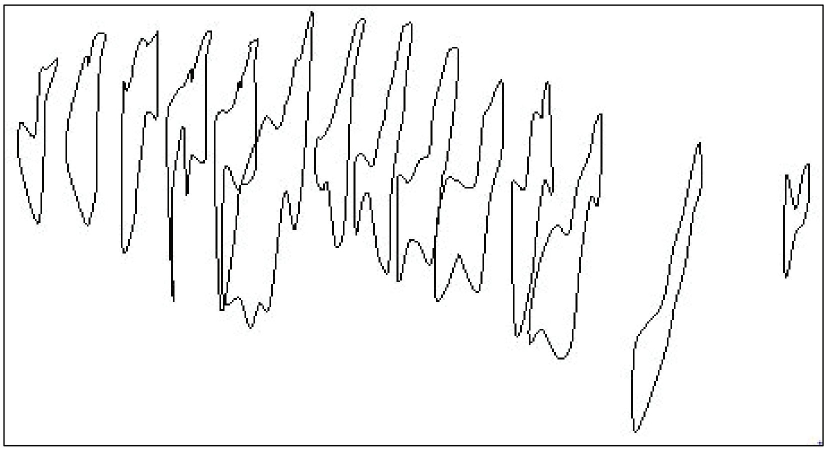


Figure 4.4-7 polygons created to build 3D ore body model

After then, the polygons were connected using the Linker Tool and in some points strong nodes were created to generate a reasonable 3D model. Furthermore, the polylines were checked for openings, direction and endpoints. Moreover, the 3D ore body model was verified for openings and self-intersecting faces. Figure 4.4-8 shows the 3D ore body model along with the topography and drill holes [41] [42].

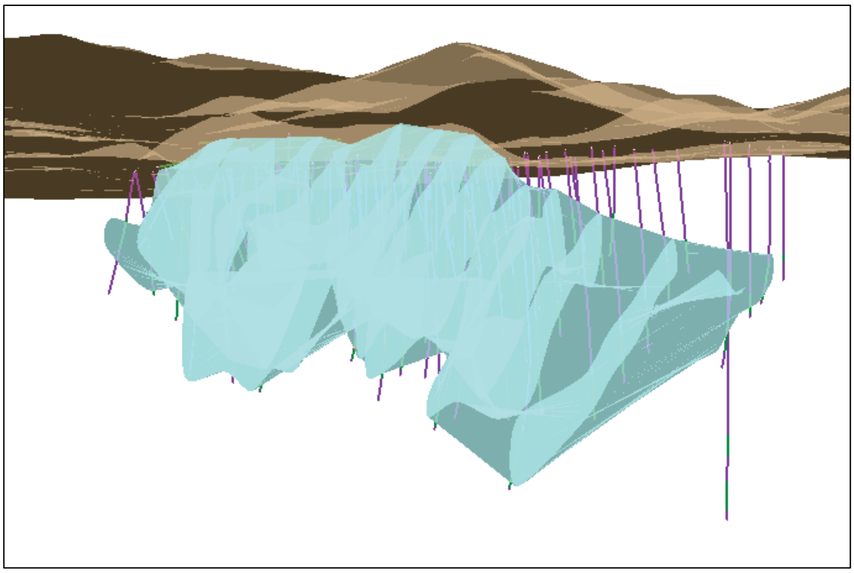


Figure 4.4-8:3D view of ore body

4.5. Block Modelling

The 3D block model (3DBM) is the fundamental component for resource assessment and mine design. The 3DBM is used for storing geologic codes and interpolating composite samples [41] [42].

4.5.1. Initialization of 3DBM File

To create a block model in Minesight, a model file aynk15 with a dimension of 25*25*10 has been initialized. The dimensions of block model were selected based on the average drill hole interval and average pit slope angle. The drilling grid was assumed 100 m and the average pit slope was assumed to be outside the range 45°-55° [5] [21].

4.5.2. Assigning of Values to 3DBM

It is strongly needed to specify values to a block model. There are no required items for block model [41]. In this case the following values have been added to 3DBM.

- CTID – copper total grade from inverse distance weighting interpolation
- CTK – copper total grade from Kriging interpolation
- COID – copper oxide value from inverse distance weighting interpolation
- COK – copper oxide value from Kriging interpolation
- SG – specific gravity
- TOPO – percentage of a block below topography
- MCOST – mining cost

- NCOMP – numbers of composites
- COST2
- COST3
- COST4

} Available options for extra costs that may be added later

- XTRA1
- XTRA2
- XTRA3
- XTRA4
- XTRA5
- XTRA6
- XTRA7
- XTRA8
- XTRA9
- XTRA10

} Available options for additional values that may be added later

4.5.3. Adding Topography to 3DBM

First of all, topography has been gridded into a surface file GSF 13. Block model has been constrained by topography to guarantee that resource calculations do not cover the blocks that are above the surface. In addition, the blocks below surface will get a value of 100% and blocks in the air will get a value of 0% [41] [42].

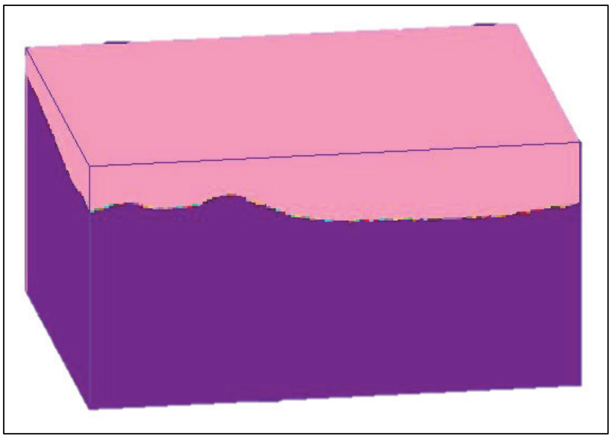


Figure 4.5-1: Model view with added topography

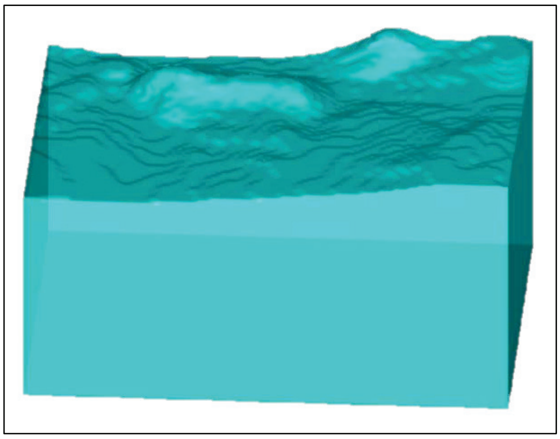


Figure 4.5-2: Block model view with constrained

4.5.4. Interpolation of the Model

To deliver the composite grades or qualities into a 3D block model, the only procedure is to interpolate the model. There are several technics of interpolation existed in Minesight. From the available methods, two routines of interpolation including Inverse Distance Weighting (IDW) and Ordinary Kriging have been used to interpolate the aynk15 block model [42] [41].

Inverse Distance Weighting (IDW)

IDW technic has been done through a procedure p62001.dat which can be accessed via Minesight compass. The important search parameters were described as follows [41] [42]:

- Search distance from block on Model-X
- Search distance from block on Model-Y
- Search distance from block on Model-Z
- Max 3-D distance from block to accept data

The following interpolation control items were described.

- CTID for mine model and cuto for composite
- COID for mine model and cuox for composite

2D and 3D views of block model after the application of IDW interpolation are shown below. Figure 4.5-3 and Figure 4.5-4 represent the block models which explain the blocks and their grades of sulphide copper interpolated by Inverse Distance Weighting technic in Aynak Copper Deposit.

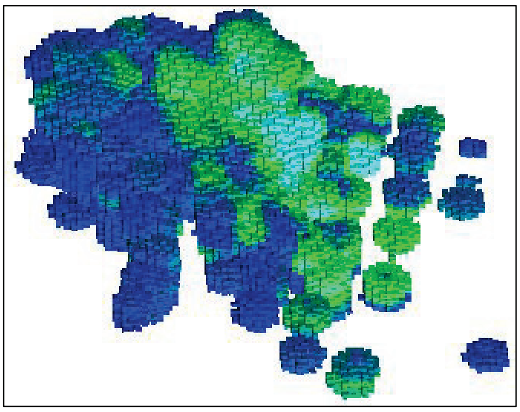


Figure 4.5-3: 3D view of CTID block model by IDW

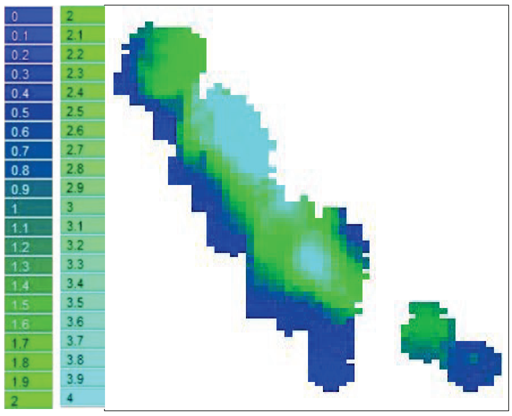


Figure 4.5-4: Cross section of CTID block model by IDW

Figure 4.5-5 and Figure 4.5-6 depict the block models of Aynak Central Copper Deposit interpolated by Inverse Distance Weighting technic. Furthermore, they represent the blocks where oxide copper is existed along with their grades which are shown in different colours

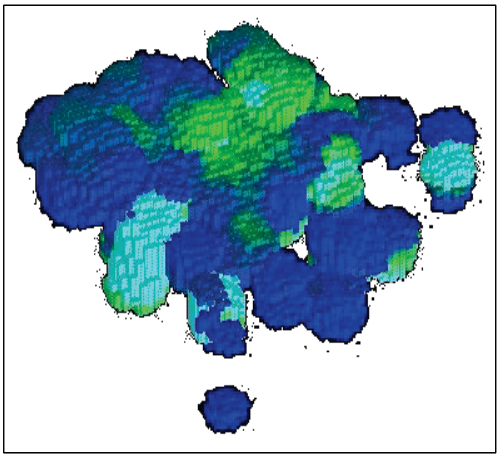


Figure 4.5-5: 3D view of COID block model by IDW

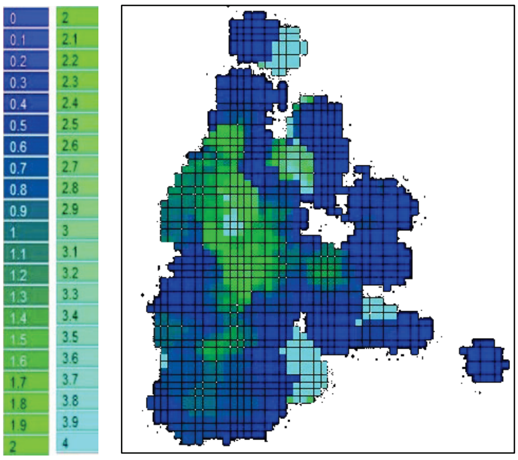


Figure 4.5-6: 2D view of COID block model by IDW

<u>Ordinary Kriging</u>

Ordinary Kriging technic has been applied through a procedure p62401.dat. The procedure can be accessed via Minesight compass. The following main search parameters were defined [41] [42]:

- Search distance from block on Model-X
- Search distance from block on Model-Y
- Search distance from block on Model-Z
- Max 3-D distance from block to accept data

The control items below mentioned were defined to the interpolation method:

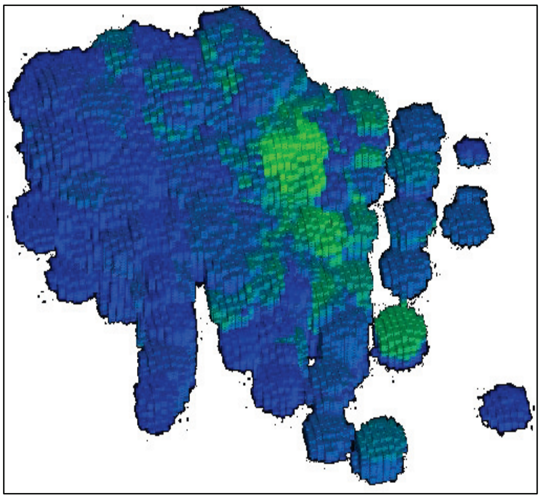
- CTK for mine model and cuto for composite
- COK for mine model and cuox for composite

The following crucial variogram parameters were also described:

- Model type
- Nugget effect
- Sill
- Range along major axis
- Range along minor axis

– Range along vertical axis

2D and 3D displays of the block model after the application of the Ordinary Kriging routine are shown below.



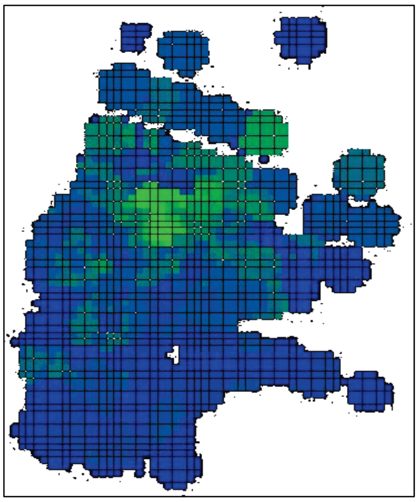
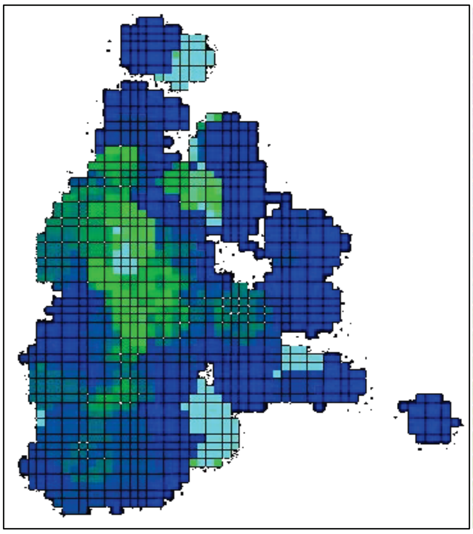


Figure 4.5-7: 2D view of CTK block model by OK

Figure 4.5-8: 3D view of CTK block model by OK

Figure 4.5-7 and Figure 4.5-8 give information about the block models of Aynak Central Copper Deposit interpolated by ordinary kriging method. Moreover, they illustrate the blocks with sulphide copper in the ACCD and their grades which is represented by different colors.



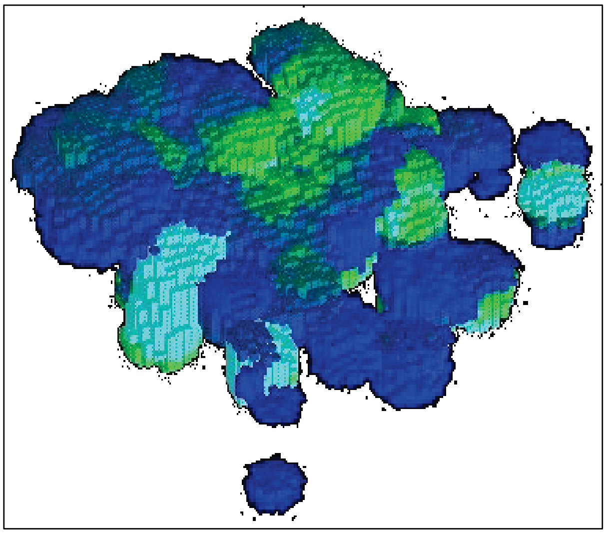


Figure 4.5-9: 2D view of COK block model by OK

Figure 4.5-10: 3D view of COK block model by OK

Figure 4.5-9 and Figure 4.5-10 display the block models of ACCD interpolated by ordinary kriging method in 2D and 3D respectively. Furthermore, they give information about the blocks and the grades of oxide copper which are shown in different colors.

4.5.5. Grade Shells

Grade shell is a closed and three-dimensional image of a code or absolute content reclaimed from a 3D block model has been created. It is used to grant the clue about the location of specific grade values or geologic codes in the model. Generally, it has been generated only for imagination purposes and not for volume or reserves estimations. Below are the grade shells produced to indicate where the blocks are located in the 3DBM by using IDW and OK interpolation methods [41] [42].

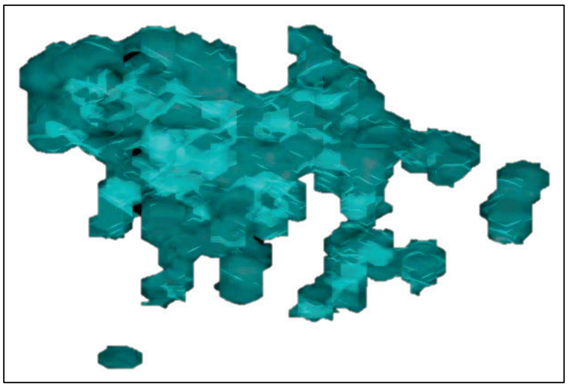


Figure 4.5-11: Grade shell of COID by IDW

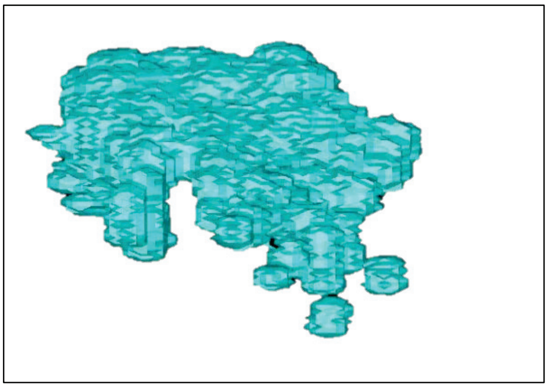


Figure 4.5-12: Grade shell of CTID by IDW

Figure 4.5-11 and Figure 4.5-12 depict the grade shells of sulphide copper and oxide copper in Aynak Central Copper Deposit respectively. In addition, they give information only about the location of blocks which contain copper with the defined cut-off in the entire 3D block model.

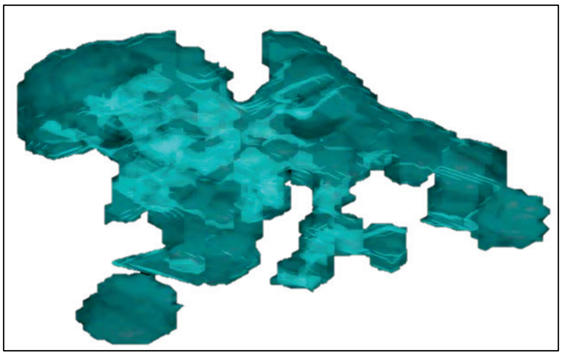


Figure 4.5-13: Grade shell of COK by OK

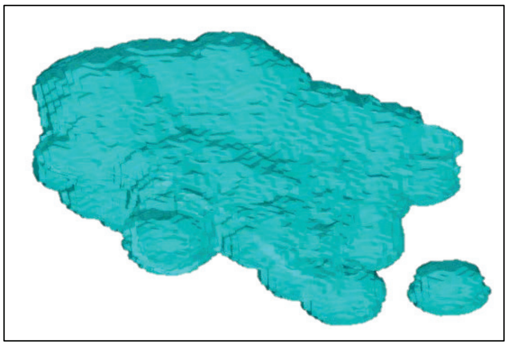


Figure 4.5-14: Grade shell of CTK by OK

Figure 4.5-13 and Figure 4.5-14 illustrate the grade shells of sulphide copper and oxide copper in Aynak Central Copper Deposit respectively. Furthermore, they just show the location of the blocks which consist of copper in the entire block model.

4.5.6. Quick Model Statistics

Quick model statistics have been created to produce brief and rapid figures for inspecting purposes. These statistics describe the model quantitatively and are valuable to examine different visible features such as grade distribution, probability and total tonnages in each cut-off grade.

The Quick Model Stats Tool in Minesight (MS3D) is a mechanism that allows us to confirm and analyse differences in the model by adjusting interpolation criteria. Furthermore, it helps to recognize large conflicts before creating extra comprehensive model statistics and graphs [41] [42].

The results obtained from this tool will be discussed in detail in Chapter 5.

4.5.7. Model Statistics Using MSDA

MineSight Data Analyst (MSDA) has been run to get more detailed information including statistics and resource report from the model. This procedure can be only applied after the model has been coded and interpolated, and all calculations are completed. Statistics describe the model quantitatively and help to evaluate different features such as grade distribution, total tonnages, and probability [41] [42].

The various types of statistics created from the model will be discussed in detail in Chapter 5.

4.5.8. Reporting Model Resources

A special tool called MineSight Reserve (MSReserve) has been used to report model reserves. It gives a possibility to generate rapid and advanced charts, graphs and reports in various patterns [41].

A reserve logic file called Aynkreserve.resx has been created to calculate resources based on the specific aynak15 block model from a range of coordinates.

The Joint Ore Reserves Committee (JORC) Code has been used to classify the reserves into Inferred, Indicated and Measured resources.

Sulphide Copper (CTID) and Oxide Copper (COID) have been estimated with a cut-off starting from 0.1 to 10 with an increment of 0.1 until 5 and after that with an increment of 0.5.

Copper with cut-off of less than 0.1 has been counted as waste. More details are discussed in Chapter 5 in Table 5.2-1.

4.6. Open Pit Optimization

MineSight Economic Planner (MSEP) tool was used for optimization. To find out an ultimate pit with MineSight, the following two new files with the required values were created.

First, a new GSF file which includes new values with the number of desired pits must be added to the old or existed GSF file. In this study, a file called Aynk13.pto with 50 pits was created with the following values explained in Table 4.6-1.

Table 4.6-1 Characteristics of Aynk13.pto file added for optimization

No	Item	Description	Minimum	Maximum	Precision
1	TOPOG	Starting surface grid	1500	2700	0.1
2	PIT00-PIT50	Pit surfaces	1500	2700	0.1
3	RCODE	Surface limit code	0	100	1
4	SLPS1-SLPS5	Surfaces for slope definition	1500	2700	0.1
5	SLPC1-SLPS5	Slope codes for surfaces	0	100	1

Second, a new model file has to be created with required and desired values. In this research, Aynak15.pto was initialized as well and the following values described in Table 4.6-2 had been assigned to the existing model fie aynk15.

Table 4.6-2 Features of Aynk15.pto file used for pit optimization

No	Item	Description	Minimum	Maximum	Precision
1	TOPO	Topography	0	100	0.1
2	CTID	Sulphide copper by IDW	0	50	0.1
3	COID	Oxidized copper by IDW	0	50	0.1
4	CTK	Sulphide copper by OK	0	50	0.1
5	COK	Oxidized copper by OK	0	50	0.1
6	MCOST	Mining cost	0	1.5	0.1
7	SG	Specific gravity	0	8.96	0.1
8	SC	Stripping cost	0	1.5	0.1
9	DIL	Dilution rate	0	5	0.1
10	LOSS	Loss rate	0	5	0.1
11	NCOMP	Number of composites	0	100	1
12	PC	Processing cost	0	13	0.1
13	REC	Recovery rate	0	90	0.1
14	OC	Other cost	0	25	0.1
15	CUTOF	Cut-off	0	0.3	0.1
16	PP	Product price	0	6700	0.1
17	DR	Discount rate	0	8	0.1
18	MNRL	Material code	1	1	0.1
19	DEST	Destination	0	0	0.1
20	NVPB	Net value per block	0	0	0.1
21	NVPT	Net value per ton	0	0	0.1
22	OUTPU	Output rate	0	11500000	0.1

4.6.1. Pit Limit Determination

The Open-pit limit has been determined by considering the ore body appearance, ore grade, joint fissures, structures, and surface topography. Furthermore, the following fundamentals were considered during outlining the open pit limit [2].

- The mineral resources or ore must be detected as much as possible into the limit to decrease Ore loss.
- The final pit slope angle must be kept at level that could assure the slope stability to avoid geological accidents.
- The geometric shape of the open pit limit must be friendly to maintain the slope stable and as well to assure that the development and haulage system is organized within the stable area of the slope.

4.6.2. Open Pit Optimization Method

MineSight software was used to optimize the open pit. Lerch's Grossman (LG) technic which is currently the most common method applied for open pit optimization, had been used to determine the open pit limit.

The LG technic guarantees an optimal result. This method needs a larger amount of calculation which could be decreased by re-blocking; which means that the blocks used for open pit design will be combined into larger blocks. Since the outcome is optimal, therefore, LG technic is used to determine the final pit limit [43].

4.6.3. Economic Parameters

In order to detect as much Ore as possible into the open pit limit and extract the maximum resources, this study focuses on the principle that the direct cost of mine production must determine the limit of the open pit. The economic parameters utilized in this study are shown below in

Table 4.6-3 and these parameters were assumed or taken from MCC Aynak Feasibility Study Report, 2015.

Table 4.6-3 List of economic parameters [2]

No	Name	Value	Unit
1	Mining Cost	1.5	$/t
2	Processing Cost	13	$/t
3	Price	3	$/lb
4	Recovery	90	%
5	Factor	22.05	
6	SG/TF	2.2	
7	1^{st} Grade	CTID	%

8	2nd Grade	COID	%
9	Material Code	MNRL	
10	Discount Rate	8	%
11	Variable Mining Cost	0.01	$/Bench
12	Minimum Slope	35	°
13	Output	11.5	10^6 tons

4.6.4. Final Slope Angle

Conforming to the present geological information, the mine area has slightly complex geotechnical environment. Based on the slope stability analysis outcomes and considering the distinct between the bed rock and Tertiary rock formations, the final slope angles are determined by various elevation.

The slope angle of bedrock below the elevation of 2190 m is determined at 44° and of Tertiary and Quaternary rocks higher than 2190 m is set at 36°, with a highest angle of 37°. Please see Figure 4.7-1 for details [2].

4.7. Open Pit Mine Design

4.7.1. Rock Mechanic Studies

The theoretical and applied science of the mechanical behavior of the rock is called rock mechanics. For experimental basis, it is mainly involved with rock masses on the scale that appears in engineering and mining activity. On the other hand, it might be considered as the study of the characteristics and behavior of accessible rock masses caused by changes in stresses or other conditions [44].

Geological Settings of Rock Mass Stability

Conforming to the Aynak Central Detailed Exploration Report, the platform complex relies upon the following three structures [2]:

- Lower complex: consists of gneiss, crystalline schist, amphibolite and marble
- Middle complex: comprises igneous and terrestrial sediments of Carboniferous and Early Permian
- Upper complex: compose the overburden of the platform of Late Permian-Triassic period

Based on the Feasibility Report of MCC, 2015, the mine area is divided into two rock formations– semi-hard rock of Neogene and rocks of Upper-Proterozoic age. Furthermore, the mine area is comprised of three tectonic layers from bottom to top, i.e. lower layer, middle layer and top layer [2].

Physical and Mechanical Properties of Ore and Rock

According to the feasibility report of MCC, the assessment results of physical and mechanical characteristics of rocks are explained below in Table 4.7-1 and Table 4.7-2 [2].

Table 4.7-1 Physical and mechanical properties of upper-Proterozoic rocks [2]

Lithology	Bulk specific gravity (g/cm3)	Compressive strength (Mpas)
Quatz-feldespar ore	2.72-2.74	101-113
Marble	2.83-2.84	143.7-145.6
Quartzite	2.62-2.71	126-137
Shale	2.77-25.78	74-76
Amphibolite	2.83-2.86	101.7-107

Table 4.7-2 Physical and mechanical properties of Tertiary rock formation [2]

Rock type	**Natural humidity (%)**	**Specific gravity (g/cm^3)**	**Compressive strength (Mpas)**	**Cohesive force (Mpas)**
Clay	7	1.98-1.99	2.3-2.4	0.1-0.3
Marlite	13.4	2-2.02	3.08-3.2	0.31-0.4
Siltstone	16	2.06-2.07	2.5-2.52	0.27-0.41
Sandstone	10.8	1.95	1.99-2.03	0.2-0.4
Limestone	17.2	2.09	6.6-7.9	0.1-0.8
Pelite	11.74	2.01-2.02	3.14-3.23	0.2-0.33
Conglomerate and cumulate gravel	5.3	1.94-1.95		0.001

4.7.2. Preliminary Geotechnical Classification of Rock Mass

The rock mass of Aynak mine area is divided into four geotechnical rock formations which are shown below in the Table 4.7-3 [2].

Table 4.7-3 Characteristics of geotechnical rock formations [2]

Rock formation	Characteristics
Quaternary loose sediments	It comprises sandstone-clayish conglomerate and grit gravel and maximum thickness of 130m.
Neogene half-hard rocks	It contains conglomerate, grit gravel, pelite, siltstone, and marl and its thickness is 650m.
Proterozoic hard rocks	This formation mostly contains silicate marble, carbonate-mica schist, and quartz. The UCS ranges between 40-214 Mpas.
Intrusive mass rocks	This formation contains hornblendite which consists of hornblende, plagioclase and garnet

Based on the geotechnical zoning and hydrological and topographic markers, the Aynak copper deposit has been divided into four geotechnical zones, which are described below in the Table 4.7-4 [2].

Table 4.7-4 Characteristics of geotechnical zones in Aynak Central area [2]

Zones	Characteristics
Pre-Cambrian rock bed	This zone contains the late Tertiary and Quaternary products with an elevation of 2100m-2500m or 2700m. The UCS is 74.6-145.6 Mpas.
Soil eroded and accumulated plain	This zone comprises Tertiary sedimentary stratum. The exact surface elevation is 2230m-2400m and the deflection angle is 3°-5°.
Mountain slope	The slope inclination is 20°-35° and separated by alluvial stream channel and its height is 3m-18m.
Valley alluvial-diluvial cone	This zone has been wiped by west-hill river current or by gutter current. It is stored with loam clay and soil has been marginally salinized.

Slope stability analysis

Based on the feasibility study report of MCC, the Aynak Central Open-pit has been categorized into six engineering geological zones. Furthermore, for the slope stability analysis, the seismic basic intensity of the mine region is believed to be 8°. The lowest threshold safety factor (F_s) of the slope considering the seismic load is 1.2.

Conforming to the outcomes of the rock mechanics study of Aynak central, the slope angles of each zone could be outlined as displayed in figure when the elevation of pit bottom is 1965 m. If the elevation of pit bottom varies, there will be changes in the total slope angle. When the slope height increases by 50 m-100 m and 100 m-150 m then the total slope angle must decrease by 1°-2° and 2°-3° respectively. The slope angles for Central Aynak Open-pit are shown in Figure 4.7-1 [2].

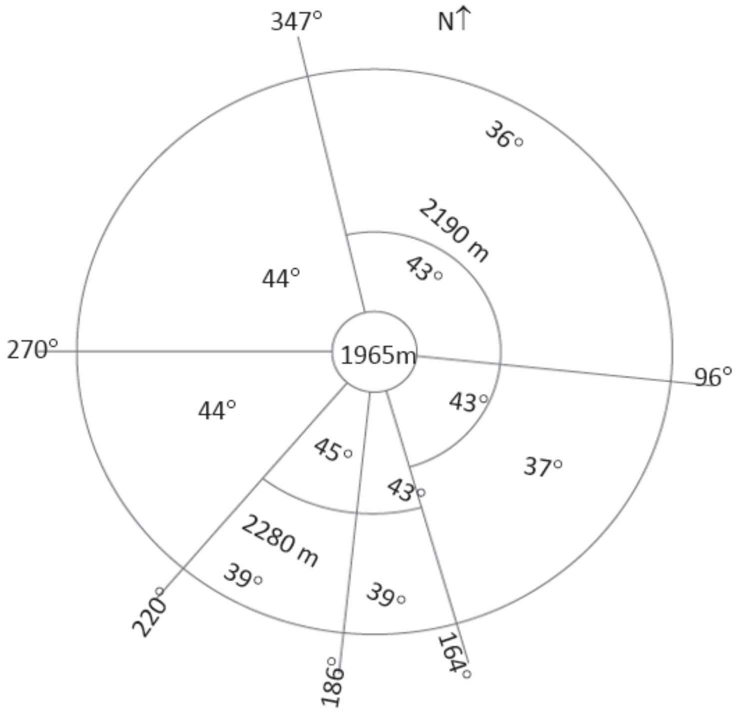


Figure 4.7-1: Slope angles in the zones of Aynak Central Mine [2]

Parameters of rock mass intensity of the six zones classified in Aynak Central have been described below in the Table 4.7-5 [2]

Table 4.7-5 Parameters of rock mass intensity [2]

Geological zones	Engineering fetrofabrics	GSI	Average height of rock stratum(m)	Compressive Strength (Mpas)
Zone 1	Amphibolite	46	264	0.89
	Carbonaceous quartz schist	56	507	1.85

Zone 2	Amphibolite	47	253	0.89
	Carbonaceous quartz schist	40	73	0.35
Zone 3	Gravel	55	37	0.11
	Argillaceous siltstone	66	100	0.3
	Sandy mudstone	65	145	0.45
	Mid-fine sandstone	77	175	0.34
	Marlstone	68	214	0.67
	Glutenite	85	252	0.49
	Breccia	85	279	0.70
	Amphibolite	48	458	1.31
	Carbonaceous quartz schist	57	576	2.06

Table 4.7-5 Parameters of rock mass intensity(continued)

Zone	Engineering fetrofabrics	GSI	Avg. height of rock stratum (m)	Compressive strength (Mpas)
Zone 4	Gravel	-	-	-
	Argillaceous siltstone	51	57	0.15
	Sandy mudstone	53	101	0.27
	Mid-fine sandstone	73	131	0.26
	Marlstone	65	170	0.55
	Glutenite	90	208	0.49
	Breccia	77	235	0.53
	Amphibolite	46	426	1.18
	Carbonaceous quartz schist	58	545	2.05

Geological zones	Engineering fetrofabrics	GSI	Av. Height of rock stratum(m)	Compressive strength (Mpas)
Zone 5	Gravel	-	-	-
	Argillaceous siltstone	-	-	-
	Sandy mudstone	40	36	0.11
	Mid-fine sandstone	73	80	0.20
	Marlstone	65	104	0.42
	Glutenite	81	143	0.32
	Breccia	71	170	0.38
	Amphibolite	44	328	0.95
	Carbonaceous quartz schist	48	462	1.39
Zone 6	Gravel	-	-	-
	Argillaceous siltstone	-	-	-
	Sandy mudstone	38	31	0.09
	Mid-fine sandstone	71	75	0.12

Marlstone	61	100	0.38
Glutenite	76	138	0.28
Breccia	70	165	0.36
Amphibolite	40	186	0.60
Carbonaceous quartz schist	39	152	0.54

Slope Monitoring System

It is necessary to monitor the slope because the final slope height of Aynak Copper mine is higher than 700 m and the depth of Tertiary rock mass is higher than 300 m. The nature of the rock mass is nearly poor; therefore, it is required to monitor high and steep slopes, rock mass and its fracture slopes.

A slope displacement monitoring system will be authorized at relevant time due to production advancement and the monitoring position will depend on the formation of boundary benches of the slope. The ongoing general slope deformation systems are Theodolite, total station, GPS and microwave radar [2].

4.7.3. Mining Method and Sequence

The exploration quality of Central Aynak Copper deposit is comparatively high and the quantity of total resources are 342,268,750.00 tons. The proportion of Inferred resources is 208.6 million tons which accounts for 60.9% of total resources, 30.7% of the total resources are related to Indicated category which accounts for 105.4 million tons and 28.2 million tons of the total resources are related to Measured class which accounts for 8.4% of the total resources.

The practical environment of Central Aynak is quite good, appropriate and acceptable for the development plan. Central Aynak deposit comprises the ore bodies that are lenticular in shape, strike NNS and tend in the direction of ES with a dip angle of 35°-40°. Moreover, the strike length is around 1850 m and the width is 1200 m.

The highest and lowest levels of ore body existence are 2530 m and 1610 m respectively and the outcrop of the ore body is 2400 m-2600m. The Overburden thickness is 50-80 m and maximum of 120 m [1]. Furthermore, the ore body existence is flat, therefore, it is suitable for open pit mining. Both the geological exploration quality and the ore degree of the Central Aynak deposit are high, this study recommends open pit mining method.

A short pre-production period of 2 and half years is specified to achieve the planned production rate quickly based on clarified process flow and satisfactory working circumstances for Open-pit mining to accomplish the full production.

4.7.4. Designing of the Open-Pit

A special tool in MineSight called Pit Expansion tool was used to design the final open pit of Central Aynak Copper Mine. The procedure is to set the elevation and then digitize a base string according to the ultimate pit. Parameters explained in Table 4.7-6 and Table 4.7-7 were required and considered in designing the open-pit and road respectively.

Table 4.7-6 Items specified for designing the road

No.	Item	Value	Unit
1	Level	1850-2550	m
2	Width	30	m
3	Grade (Fraction)	0.08-0.1	%
4	Direction	1	clockwise

Table 4.7-7 Necessary parameters used in designing open-pit

No.	Item	Value	Unit
1	Elevation	1850-2550	m
2	Bench Height	15	m
3	Berm	20-30	m
4	Face Slope	70-80	°
5	Pit Slope	39-45	°

4.7.5. Material Destination

Material are categorized into two groups based on economic parameters specified for destination. First group is Ore which will be sent to mill because they can pay for all the cost including mining cost, processing cost and etc. Second group is waste which will be transported to dump because they cannot reimburse the expenses which will be likely spent for extracting the material. Table 4.7-8 indicates the economic parameters defined for destinations.

Table 4.7-8 Economic parameters for destinations

Destination	Material	Grade	Price	Processing cost ($/t)	Mining cost ($/t)	Recovery (%)	Factor	SG/TF
Mill	Ore	CTID	3.0	13	1.5	90	22.05	2.0
		COID	3.0	1.5	1.5	80	22.05	2.0
Dump	waste	CTID	0	1.5	1.5	0	22.05	2.0
		COID	0	1.5	1.5	0	22.05	2.0

Chapter 5. Results and Discussions

In this chapter, results and statistics obtained from assays, compositing, and block model were described and discussed. In addition, mineral resource classification, pit optimization, production capacity, waste disposal and push backs were analyzed.

5.1. Statistics

In this part, the crucial statistics obtained from assays, composites and 3DBM will be discussed briefly.

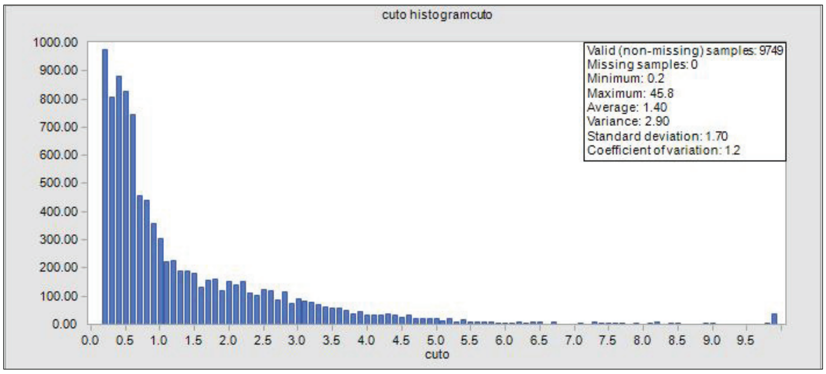


Figure 5.1-1 Sulphide copper assay histogram

Figure 5.1-1 shows the histogram of assays data. It depicts that high number of samples have less than 1.5% of copper grade. Furthermore, it indicates that in assay data, the copper grade varies significantly and ranges from 0.1% to 10%. The standard deviation for sulphide copper in assay data is 1.70.

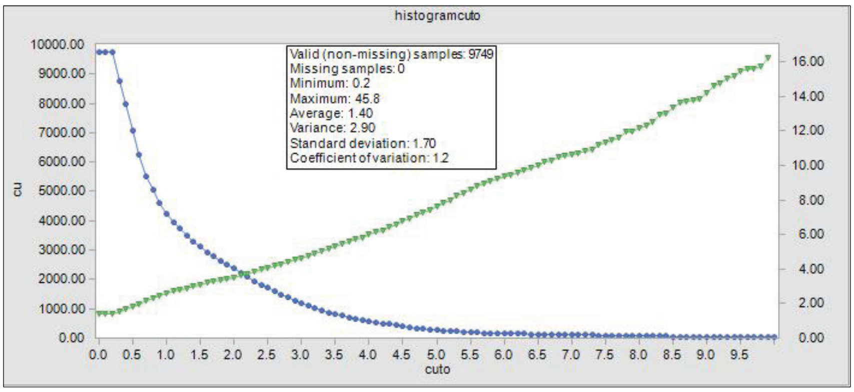


Figure 5.1-2 Sulphide copper grade tonnage curve based on assay data

Figure 5.1-2 represents the grade tonnage curve of assay data related to sulphide copper. It indicates that higher quantity of sulphide copper has low grade. On the other hand, around 1000 tons have copper grade of higher than 16%.

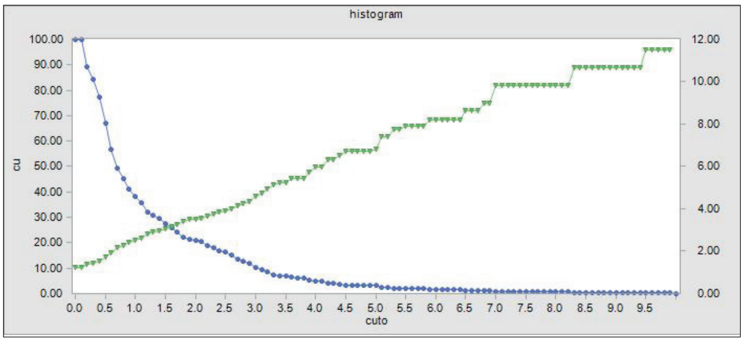


Figure 5.1-3 Grade tonnage curve of sulphide copper based on composites

Figure 5.1-3 represents the grade tonnage curve of sulphide copper in composites. It demonstrates that less 10% of sulphide copper has copper grade between 10-12%.

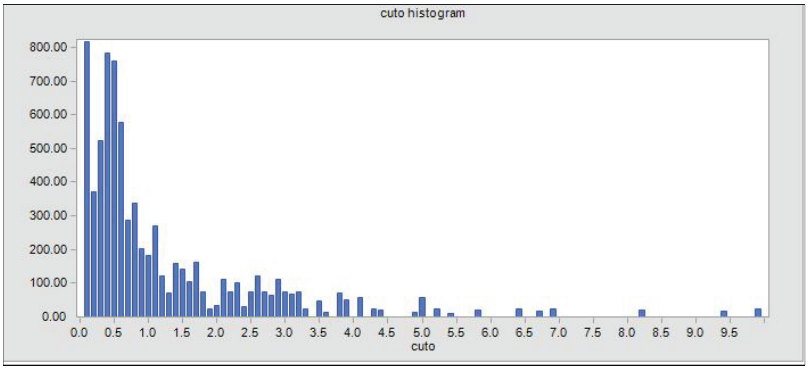


Figure 5.1-4 Sulphide copper histogram based on composites

Figure 5.1-4 graph shows sulphide copper histogram. Generally, it represents an incredible fluctuation in the distribution of copper grade based on compositing. This bar graph demonstrates that high number of samples have copper grade less than 4% in contrast to the less number samples which represent high grade up to 10%.

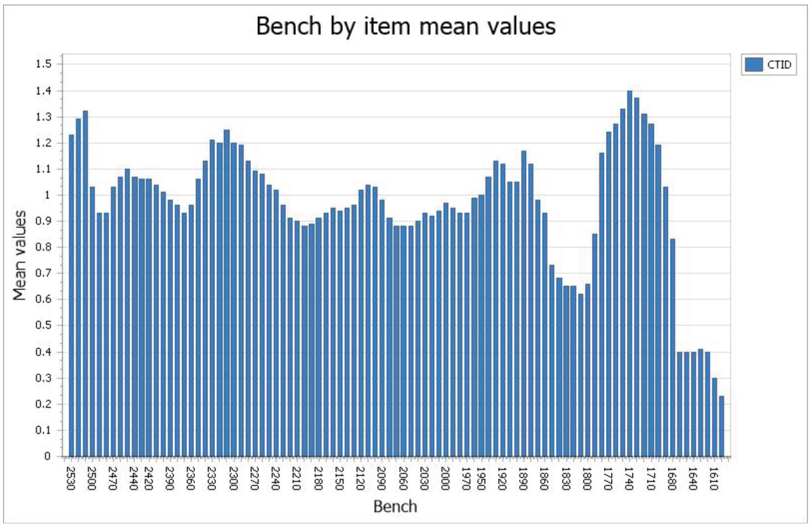


Figure 5.1-5: CTID Benches by mean values

Figure 5.1-5 above shows the mean values of Sulphide Copper (CTID) per bench by applying ordinary kriging technic. X axis and Y axis illustrate the elevation of benches and mean values respectively. Generally, it depicts a fluctuation in the mean values per bench. Additionally, it shows that the lower benches have the smaller mean values in comparison to the benches starting from 1700 m and above.

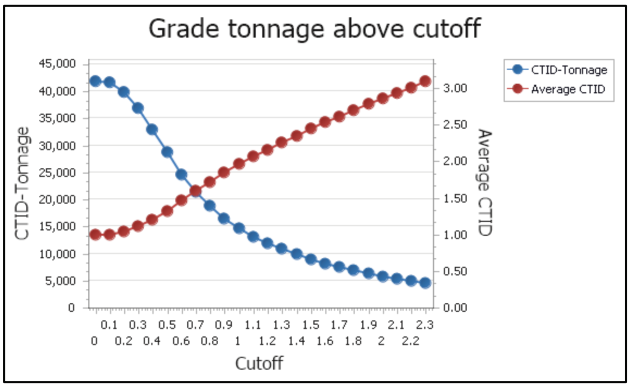


Figure 5.1-6: CTID grade tonnage above cutoff

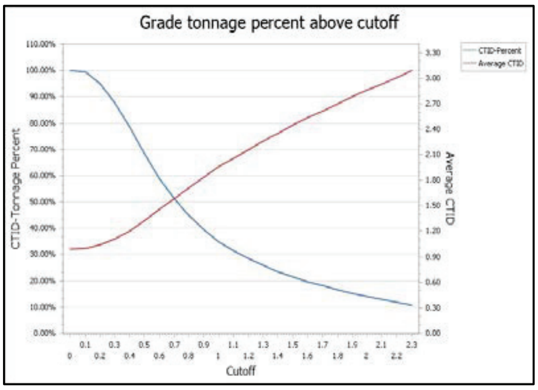


Figure 5.1-7: CTID grade tonnage percent above cutoff

Figure 5.1-6 and Figure 5.1-7 represent the grade tonnage curves of sulphide copper in application of IDW method. Graph on left side shows quantity in tons and on right side shows percentage respectively. Looking into the graph on left side, it can be said that the higher tonnage has lower grade. Figure 5.1-7 indicates that around 10% of sulphide copper is higher than an average grade of 2.3%.

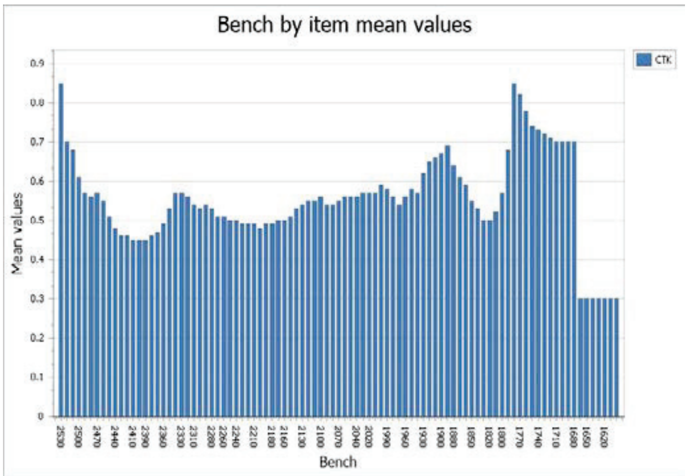


Figure 5.1-8 CTK benches by item mean value

Figure 5.1-8 deals with the mean values of sulphide copper per bench. Ordinary kriging technique was applied. It represents that the benches in higher elevation has higher mean value than the benches located in lower elevation. The higher mean value is 0.85 and lower mean value is 0.29.

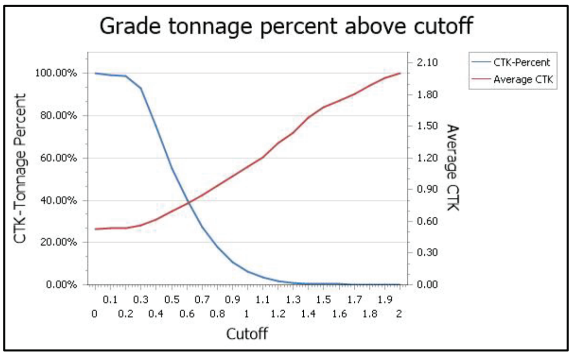


Figure 5.1-9 CTK grade tonnage percent above cutoff

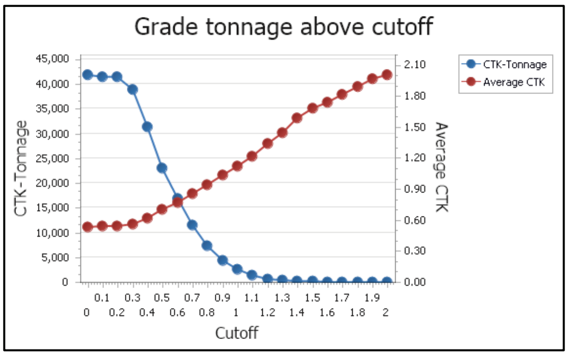


Figure 5.1-10 CTK grade tonnage above cutoff

Figure 5.1-9 and Figure 5.1-10 demonstrate the grade tonnage curve of sulphide copper in percentage and tons respectively. Noticing the graph on left side, it can be said that the higher percentage has low grade with lower than 0.1%. The graph o right side visualizes that less than 5,000 tons have grade higher than 2%.

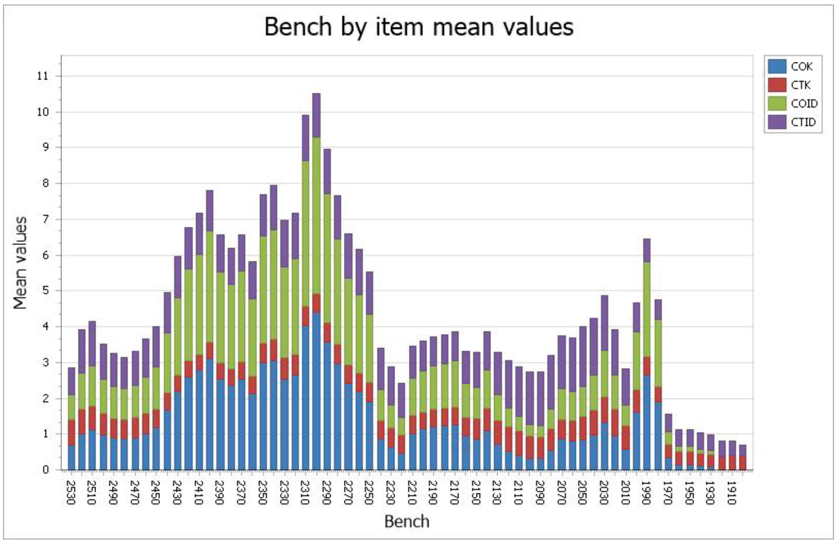


Figure 5.1-11: Sulphide and oxidized copper comparison by item mean values per bench, applying IDW and OK

Figure 5.1-11 indicates the comparison of inverse distance weighting and ordinary kriging methods applied for sulphide and oxidized copper. It shows that mean values for oxidized copper by kriging and oxidized copper by inverse distance are nearly similar to each other. It shows homogenous distribution of the oxidized copper grade. On the other hand, the mean value differs for sulphide copper by inverse distance weighting and ordinary kriging. The IDW approach represents higher mean value than OK procedure.

	Minimum	Maximum	Std. Devn.
TOPO			
CTID	0.10	8.90	1.05
COID	0.10	30.00	5.11
CTK	0.10	2.00	0.25
COK	0.10	30.00	5.11

Figure 5.1-12: Basic model report

Figure 5.1-12 represents the basic statistics about the block model. It shows that the minimum and maximum grades for Oxidized Copper (CO) by IDW and OK are equal and represent an equivalent standard deviation of 5.11. However, it is not similar in case of Sulphide Copper (CT). It illustrates that the maximum grade by IDW and OK is 8.9% and 2% respectively and the standard deviation for CTID is 1.05 and for CTK is 0.25.

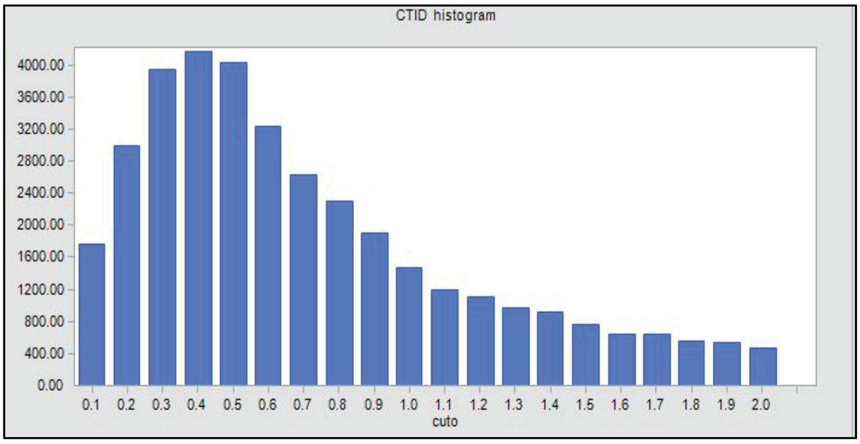


Figure 5.1-13: CTID histogram utilizing IDW approach

Figure 5.1-13 illustrates the sulphide copper distribution according to the number of samples based on IDW approach. The vertical axis shows the number of samples and horizontal axis represents the copper grade. Looking to the bar graph, generally, it shows that huge number of samples indicate lower grade of less than 1%. A small number of samples appear having grade between 1-2%.

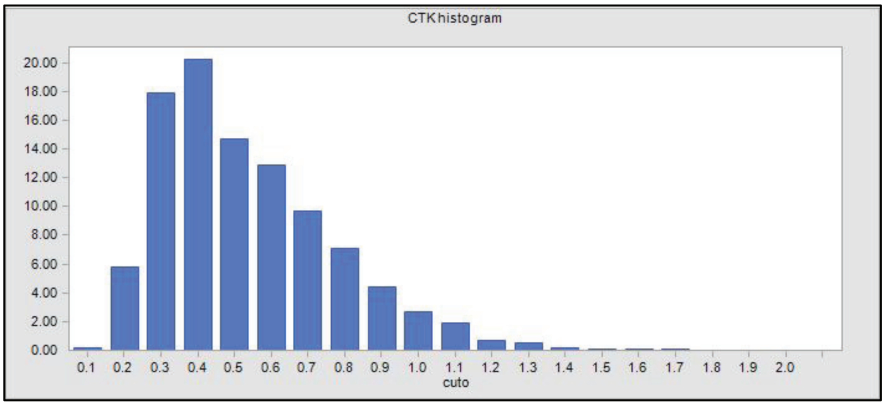


Figure 5.1-14: Sulphide copper histogram applying ordinary kriging method

Figure 5.1-14 shows the sulphide copper histogram in regard to the application of ordinary kriging procedure. It depicts that high number of samples have copper grade of less than 0.8%. On the other hand, it visualizes that very low number of samples have grade between 0.9-1.7%.

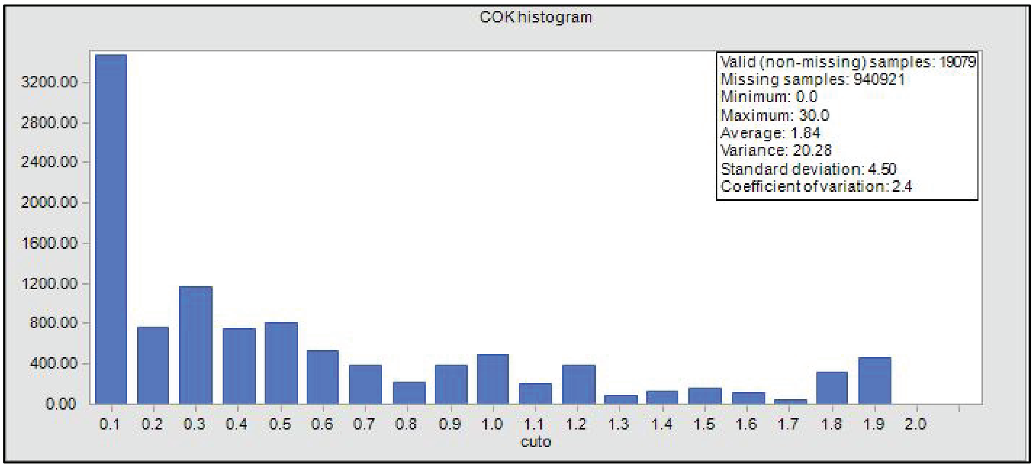


Figure 5.1-15: Oxidized copper histogram applying ordinary kriging

Figure 5.1-15 illustrates oxidized copper histogram in regard to the application of ordinary kriging procedure. Generally, it shows that the distribution of grade varies significantly. It represents an average grade of 1.84 with a standard deviation of 4.5.

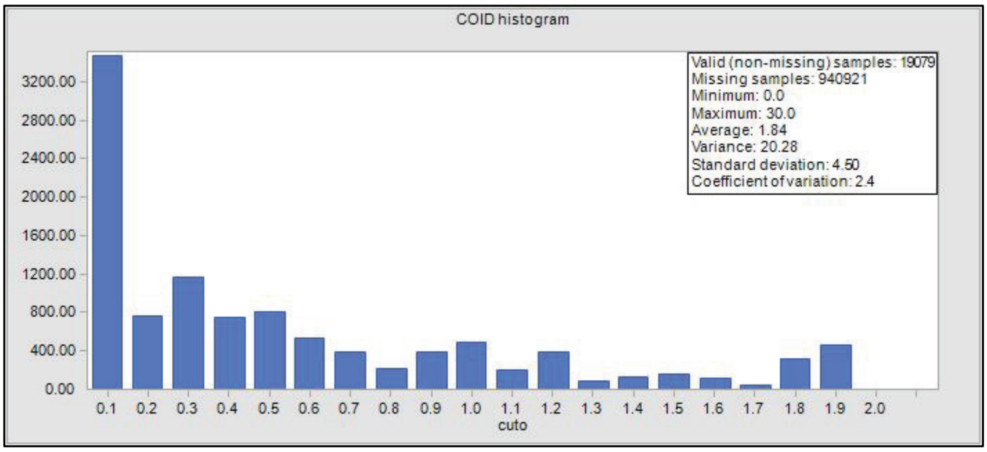


Figure 5.1-16: Oxidized copper histogram by application of inverse distance weighting technique

Figure 5.1-16 demonstrates the histogram of oxidized copper respecting to the application of IDW approach. Mainly, it depicts a great variation in the distribution of copper grades concerning the number of samples. Looking to the bar graph, it seems that a lot of samples have copper grade of less than 1.2%. The average grade of oxidized copper is 1.84 and the standard deviation is 4.5.

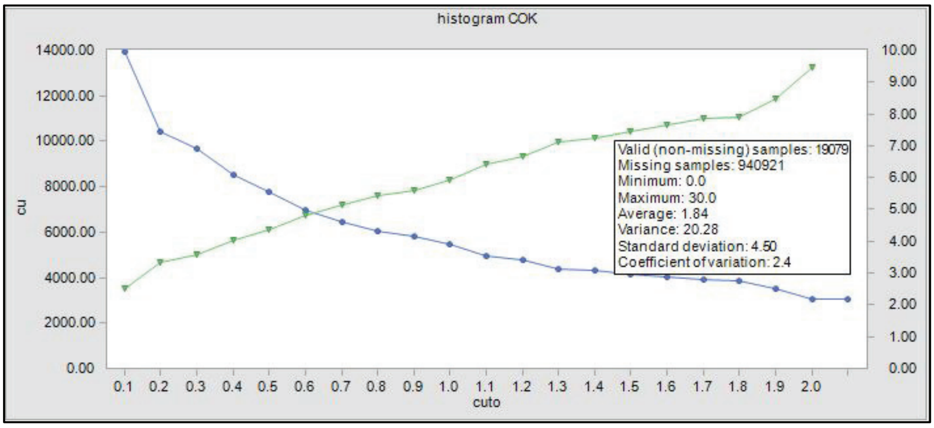


Figure 5.1-17: Oxididized copper grade tonnage curve by applying ordinary kriging

Figure 5.1-17 shows the grade tonnage curve of oxidized copper respecting to the application of ordinary kriging approach. Commonly, it indicates that high quantity of copper has a low grade of 0.1% and less than 40, 000 tons have an average grade higher than 2%.

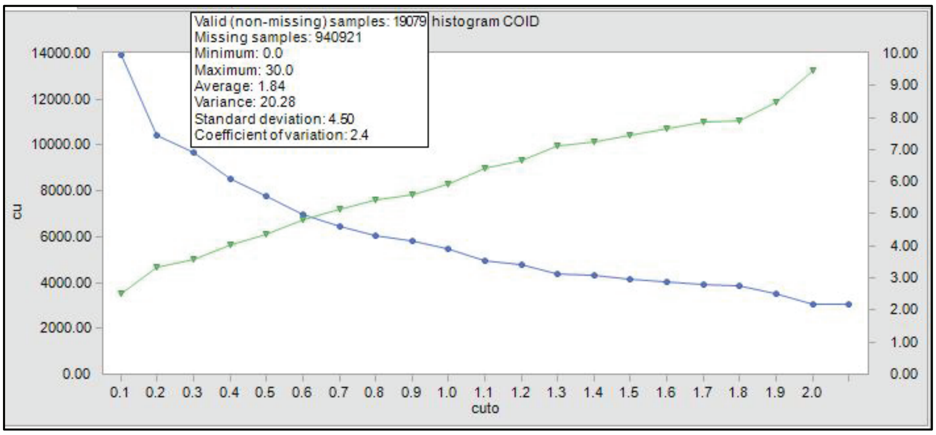


Figure 5.1-18: Oxidized copper grade tonnage curve by applying inverse distance

Figure 5.1-18 demonstrates the grade tonnage curve of oxidized copper in regard to the application of inverse distance weighting technique. Generally, it can be said that the higher tonnage has a low grade. And low quantity of oxidized copper has high grade higher than 2%. Furthermore, average grade is 1.84 and standard deviation is 4.5.

5.2. Mineral Resources

Mineral resources were calculated using MSReserve tool and had classified into three categories based on JORC code.

Table 5.2-1 Mineral resources reported from 3D block model

Type	Class	Ore Tonnage (t)	Cu grade (%)	Cu Metal (t)
Sulphide Copper (CTID)	Inferred	132,188,281.25		
	Indicated	65,347,968.75		
	Measured	16,956,406.25		
	Total	214,930,156.25	1.83	3,933,221.85
Oxidized Copper (COID)	Inferred	76,409,218.75		
	Indicated	39,691,718.75		
	Measured	11,237,656.25		
	Total	127,338,593.8	1.69	2,152,022.24
Sub-total	Total Inferred	208,597,499.50		
	Total Indicated	105,039,687.50		
	Total Measured	28,194,062.50		
Total	CTID + COID	342,268,750.00	1.77	6,058,156.88

As it can be seen in Table 3.1-1, the total tonnage of mineral resources in Aynak Central part is 342.3 million tons with an average grade of 1.77% which is equivalent to 6.05 million tons of copper metal. The proportion of Inferred resources are 208.6 million tons which accounts for 60.9% of the total resources. Indicated resources represent 30.7% of total resources which is equal to 105.04 million tons and Measured resources count for 28.2 million tons which is equivalent to 8.4% of the total resources.

In regard to the copper type, sulphide copper depicts 214.9 million tons which is equal to 62.8% and the remaining 127.34 million tons are related to oxide copper which represents 37.2% of the total resources.

5.3. Ultimate Pit

The most essential technical properties of the final pit are described below in Table 5.3-1.This table shows the total amount of recoverable reserves which are around 329,008,501 tons with an average copper grade of 1.62%. Furthermore, the final depth of the pit is 730 m and the stripping ratio is 3.1 t/t.

Table 5.3-1 Main technical properties of Aynak central open-pit

Item	Description	Value	Unit
Stope elevation	highest	2550	m
	lowest level	1850	m
Stope depth	Total depth	700	m
Surface dimension	Max. length	2083	m
	Max. width	1357	m
Floor dimension	Max. length	258	m
	Max. width	163	m
Ore tonnage		329,008,501	t
Waste tonnage		1,020,640,514	t
Ore and waste tonnage		1,349,649,015	t
Average stripping ratio		3.1	t/t

The 3D view of final pit is shown in Figure 5.3-1, which represent how the final pit looks like and the ore blocks that might be mined. The section of the final pit is displayed in Figure 5.3-2, which shows the optimized limits of the final pit.

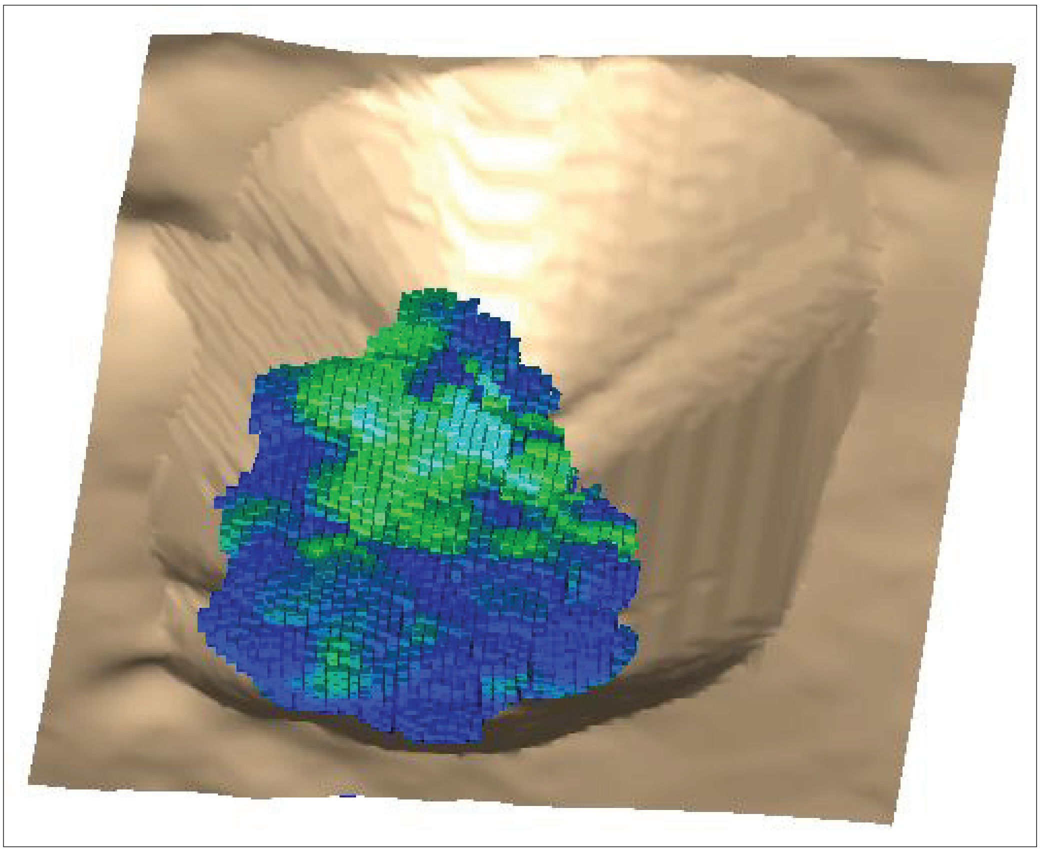


Figure 5.3-1: 3D view of Aynak Central final pit

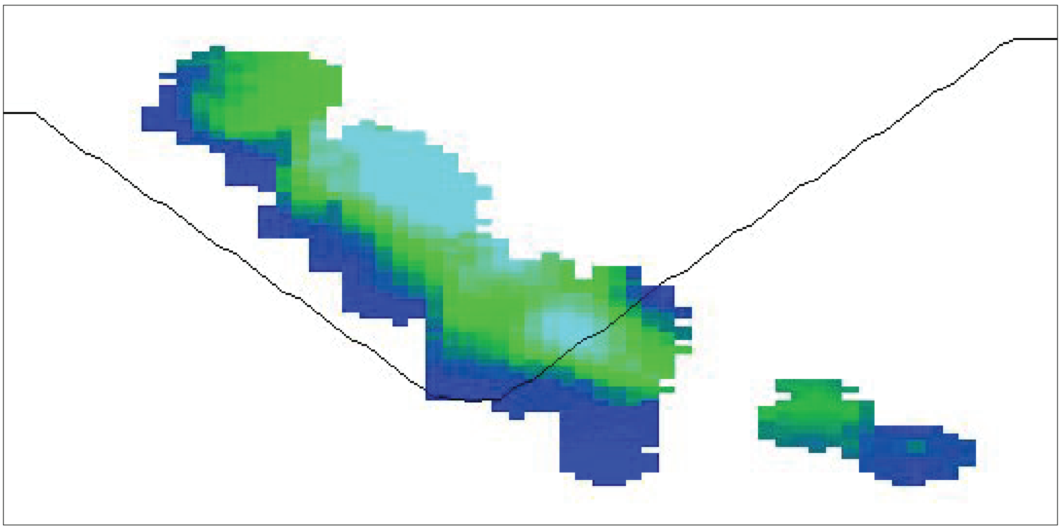


Figure 5.3-2: Cross section of the Aynak Central ultimate pit

5.4. Aynak Final Open-Pit

The final open-pit of Aynak central will look like as it is displayed below in Figure 5.4-1. The total tonnage of copper ore exploited within the entire production period is around 329,008,501 tons which is equal to 5,823,450 tons of copper metal.

The recovery rate is 96% and the stripping ratio is 3.1 t/t. Aynak Central open-pit is around 2100 m long and 1360 m wide. The mine life is estimated around 25 years. The overall slope angle is 45°. The height and width of the benches are around 15 m and 30 m respectively and the slope angle for benches varies depending on the rock formation and calculated between the range from 60-65°.

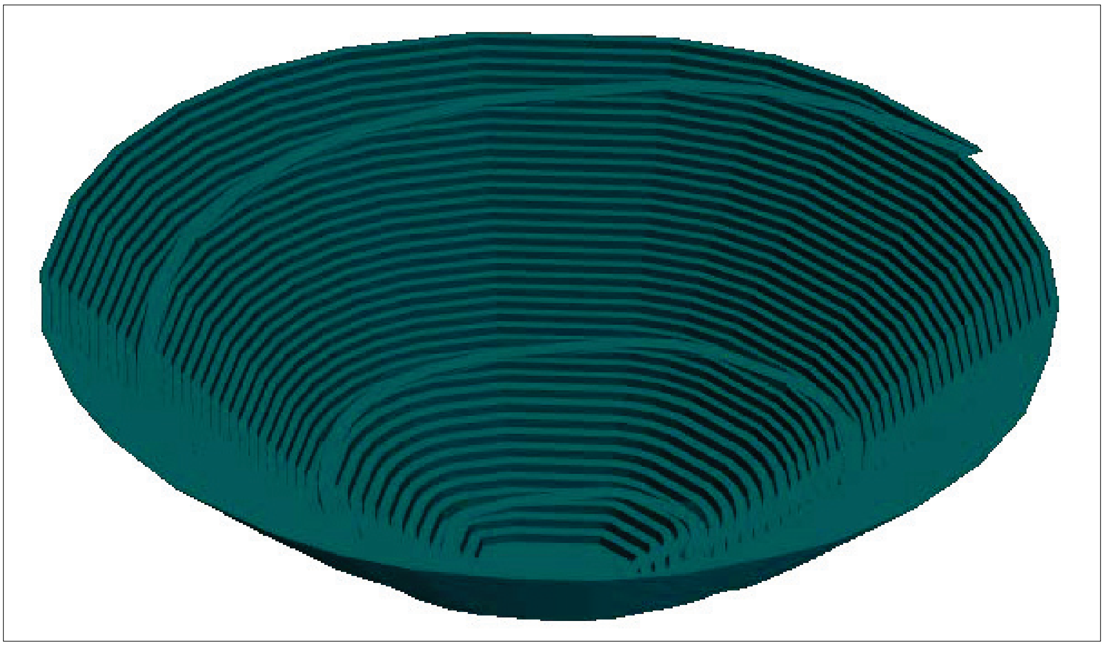


Figure 5.4-1: 3D display of Aynak Central Open-Pit

5.4.1. Transportation of Ore

The production rate of ore is 40,000 t/d which is equivalent to 13.2 million t/a and the total quantity of ore produced during the entire service life of Aynak Central Open-Pit is 329,068,501 million tons.

Ore will be transported by 218 t electric-wheel self-dumping trucks to a semi-mobile crushing station in the open-pit. After then redeployed by a 1,580-m long conveyor-belt to the stockpile at the concentrator industrial site [2].

5.5. Waste Management

Considering the difference in type and state of the waste material, different individual sites are considered to manage waste in the Aynak Central Open-Pit area. The following norms, regulations and standards will be carefully considered [2].

- Code for Designing of Tailing Facilities (GB50863-2013)
- Safety Technical Regulations of the Tailing Ponds (AQ2006-2005)
- Design Code for Rolled Earth-rock Fill Dam (SL274-2001)
- Technical Regulations for the Tailing Ponds Safety Monitoring (AQ2030-2010)
- Code for Seismic Design of Special Structures (GB50191-2012)
- Standard for Pollution Control on the Storage and Disposal Site for General Industrial Solid Wastes (GB18599-2001)

5.5.1. Waste Dump

Waste Dump will be constructed for the solid wastes including overburden and mining waste rocks. The total amount of waste rocks in the Aynak Central Open-Pit is around 1,020.7 million tons.

The waste dump is built on the north side of the open-pit and its area is around 6.3 million m^2. The crest elevation of dump site is 2609 m and the slope angle is 37°. The waste dump is around 2450 m wide and stockpiling height is 101 m. The capacity of waste dump is around 378.3 million m^3 and is quite suitable for stockpiling all the waste generated for the entire service life of the mine.

The waste rocks will be directly transported by 218 t self-dumping trucks from open-pit to the waste dump. There will be four 372 kW bulldozers for accumulating, flattening, and rolling-over the waste rock on the dump. The waste haulage road is 26 m wide and the road bed width is 32 m [2].

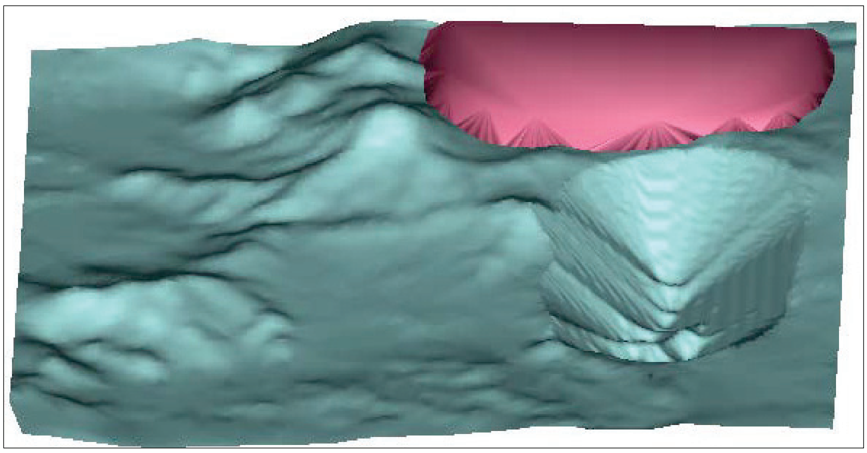


Figure 5.5-1: Waste dump location

5.5.2. Tailings Pond

The total quantity of tailings produced during the entire life of the Aynak Central Open-Pit from processing plant is assumed to be 400 million m^3. Tailing pond will be built on the east-west side and 500 m far from the open-pit. The crest elevation of tailing pond is 2456 m and its area is around 7.2 million m^2. The tailings will be transported to tailings pond through pipelines.

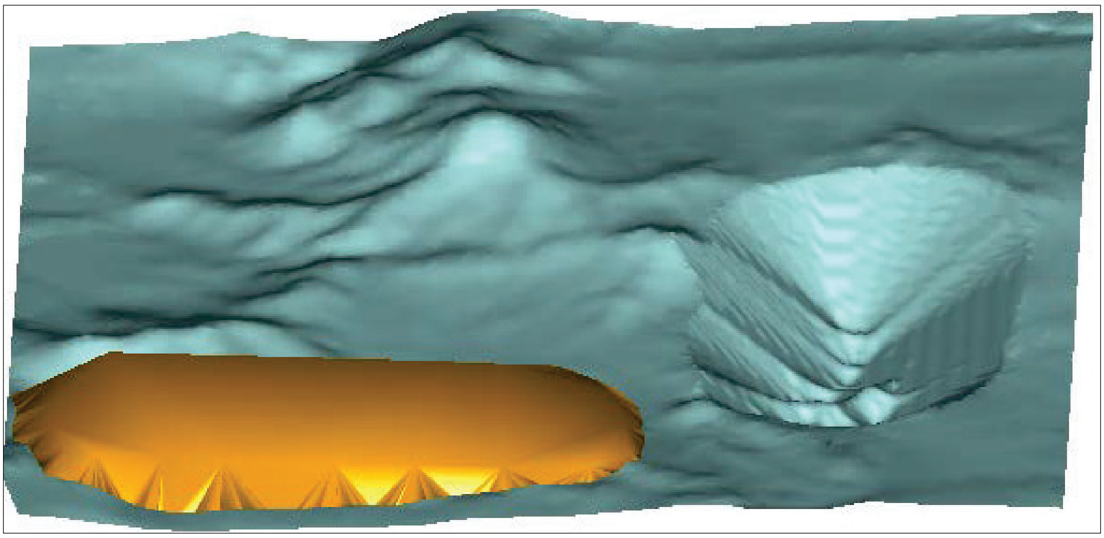


Figure 5.5-2: Location of tailings pond

5.5.3. Waste Water Management

According to MCC-JCL Feasibility Study Report of Aynak Central Open-Pit, the total water usage during the entire period of open-pit is 208,442million m^3/d which does not involve the unpredicted water consumption of 4,352 m^3/day. The share of fresh water for production is 241,48 million m^3/d, domestic water volume is 75,361 300 m^3/d, circulating water is estimated around 104,553 m^3/d, return water capacity is 75,361 m^3/d, secondary water quantity is 4,080 m^3/d and the water recycling rate is 88.27% [2].

The waste water produced from engineering machines repairing shop must be de-greased and delivered to stope purification station for beneficiation and consumed as the new water in the production area of plant. The domestic waste is handled in septic tank and biochemical processing accessory and after treatment, the water is utilized for landscaping and irrigation. The fresh water for production is evaporated and disappeared totally [2].

5.6. Production planning

Production planning is the essential task for feasibility study and for later production. In this thesis, the focus will be on production rate determination, mine life, push backs and sequencing.

5.6.1. Production capacity

The quantity planned for Open pit mining is 40,000 t/d which is equal to 13,2 million t/a including sulphide and oxidized ore. Conforming to the technical environment of the deposit. The production capacity is validated as below [2]:

- validation of the annual production quantity based on the expansion rate of

$$A = P.V.\frac{\eta}{h.(1-e)} \qquad \textbf{(5.6-1)}$$

  mining activities.

Where,
A = annual production quantity that could be likely attained
P = average ore quantity per bench, there are 46 benches from 2550 m to 1850 m, so average ore quantity per bench is around 7.2 million tons.
V = deepening rate of the mining activities which is 25m-35m in the world large open pit mines
H = height of bench
η = ore recovery rate
e = ratio of mixed waste
The open pit production quantity that can be achieved is calculated below:
A=$\frac{7.2*30*0.96}{15*(1-0.04)} = 13.3$ million t/a
As it can be seen from the calculation, within ultimate pit limit a production capacity of 13.2 million t/a can be achieved.

- Validation of the open pit production capacity based on the number of excavating equipment organized in mining face.

The surface dimensions of the pit are 2083 m in length and 1357 m in width and pit bottom is 258 m long and 163 m wide. Conforming to real production experiences in similar mines at least one bench can be ensured and at least one 35 m^3 power shovel would be organized for mining. Based on MCC feasibility Study Report 2015, the length of mining front is more than 300 m and truck transportation is selected. One WK-35 type of electric shovel with a bucket size of 35 m^3 is selected and its annual efficiency is 18 million tons per set [2].

$$A = N.n.Q \qquad (5.6\text{-}2)$$

Where,
A = annual production quantity of the pit that could be likely attained
N = number of electric shovel
Q = annual production quantity of electric shovel
n = number of exploration stage during synchronous work
The open pit production quantity that can be achieved is calculated below:
A=N*n*Q=1*1*1800=18 million t/a.
As it can be seen from the calculation, the production capacity of 13.2 million t/a can be achieved in intermediate mine.

5.6.2. Working system and mine life

Working system of Aynak Central open pit is designed in such a manner that an annual production of 13.2 million t/a could be achieved. There are 330 working days per year, three shifts working per day and each shift is designed for eight hours. The mine life is calculated using the equation (5.6-3).

$$T = \frac{Q.\eta}{A(1-e)} \qquad (5.6\text{-}3)$$

Where,
T = mine life
Q = Total ore tonnage
A = annual production rate
η = ore recovery
e = ratio of mixed waste

$$T = \frac{329 * 0.96}{13.2(1-0.04)} = 22.9 \approx 23$$

The mine life of Central Aynak Open-pit is estimated about 23 years.

5.6.3. Pre-Production Schedule for Open-Pit Mining

The pre-production schedule of Central Aynak Open-pit mining covers three main sections [2]:

- To construct a skyward road that allows us to set up a connection among open pit, ore crushing station, waste dump and other different working faces. In addition, this connection authorizes the mining equipment and stripping equipment to gain access into the active faces readily and to deliver the extracted ore and waste to primary crushing station and waste dump.
- To set up the belt conveyor between primary crushing station and the concentrator to deliver the crushed ore to stockpile at concentrator.
- To remove some waste meaning to provide sufficient working area for mining and stripping equipment and to produce adequate development and extraction reserves.

The preproduction work of open pit mining includes the building of a 3-km skyward road and installation of semi-mobile crusher. The pre-production time of open pit is 3 years [1].

5.6.4. Production Schedule for Aynak Open-Pit Mine

The mine will be set for production right away after the mine pre-production is finished. Firstly, production schedule was designed for pushbacks using MSEP tool and through MSOPIT-Design Pits procedure. This tool gives information about the number of pushbacks, periods, ore and waste quantity. Three pushbacks per year were defined to be mined per year. This calculation assumes that each pushback is mined top down sequentially and all economic material is processed.

Table 5.6-1 Cumulative summary of Aynak central open-pit

Period	**Pushback**	**Cumulative net value (10^3)**	**Cumulative ore (10^3t)**	**Cumulative ore & waste (10^3 t)**	**Cumulative present value (10^3)**	**Cum% value**
1	1	549,601	15,938	46,738	525,295	8.96
2	1	882,715	26,525	79,638	819,727	14.40
3	1	140,3068	44,250	140,900	1,243,497	22.88
4	1	1,762,401	56,888	179,450	1,511,158	28.74
5	2	2,145,795	70,550	213,850	1,775,601	35.00
6	2	2,549,774	84,888	243,900	2,032,451	41.59
7	2	2,965,067	100,025	270,188	2,74,788	48.36
8	2	3,177,592	108,062	282,100	2,390,656	51.83
9	2	3,576,182	123,725	303,238	2,593,575	58.33
10	2	3,929,084	138,562	321,300	2,757,951	64.08
11	2	4,208,623	151,000	336,312	2,878,191	68.64
12	2	4,409,143	160,475	348,188	2,959,094	71.91
13	2	4,609,673	171,850	362,775	3,035,359	75.18
14	3	4,745,178	186,725	595,512	3,082,044	77.39
15	3	4,965,237	198,888	617,688	3,153,582	80.98
16	3	5,197,671	212,088	638,538	3,223,720	84.77
17	3	5,440,774	225,912	659,362	3,291,586	88.74
18	4	5,600,723	237,625	673,538	3,333,081	91.35
19	4	5,737,797	251,512	831,475	3,365,255	93.58
20	9	5,853,387	264,450	959,462	3,391,493	95.47
21	9	6,051,694	277,275	990,650	3,432,461	98.70
22	9	6,131,172	289,238	1,138,850	3,447,764	100.00

Table 5.6-1 illustrates cumulative present value and net value for each year. It also gives information about the cumulative ore production and cumulative ore/waste tonnage.

In order to design mine schedule on yearly basis, a production rate of 13.2 million/a was specified. Based on this production rate, the mine will operate for 22 years and other relevant information is demonstrated below in Table 5.6-2.

Table 5.6-2 Yearly schedule summary of Aynak central open-pit. Unit for value and ton is 10^3.

Period	Net value	Ore (t/a)	Ore & waste (t/a)	Present value	%Total NPV	Strip ratio
1	459,972	13200	38,718	442,516	12.835	1.933
2	418,868	13200	40,518	373,838	23.678	2.070
3	390,381	13200	45,529	321,903	33.014	2.449
4	377,811	13200	42,693	288,534	41.383	2.234
5	370,281	13200	35,455	261,916	48.980	1.686
6	372,501	13200	29,679	244,124	56.060	1.248
7	367,559	13200	25,060	223,411	62.540	0.898
8	355,107	13200	20,798	198,914	68.310	0.576
9	340,126	13200	18,305	176,976	73.443	0.387
10	322,974	13200	16,732	155,915	77.965	0.268
11	305,117	13200	15,808	136,182	81.915	0.198
12	285,404	13200	16,233	117,763	85.330	0.230
13	239,820	13200	16,946	91,980	87.998	0.284
14	102,352	13200	22,7901	35,543	89.029	16.265
15	241,175	13200	25,859	79,049	91.322	0.959
16	232,240	13200	20,871	70,457	93.366	0.581
17	233,909	13200	20,196	65,755	95.273	0.530
18	184,824	13200	16,206	48,226	96.671	0.228
19	125,549	13200	15,7068	29,504	97.527	10.899
20	119,878	13200	12,6607	27,368	98.321	8.591
21	204,857	13200	33,368	42,387	99.550	1.528
22	804,68	12038	148,300	15,500	100.00	11.320

Table 5.6-2 depicts that the production quantity in the first year will be 13.2 million tons. The production rate will remain equal to the planned amount of 13.2 million t/a until the end of 21st year of the mine. After then, the production rate will decrease by 12 million tons in the last year of mine service life. The mine life will come to a closure in the end of 22nd year excluding the pre-production time. The entire service life of the Aynak Central Open-Pit Mine will last 25 years including pre-production time.

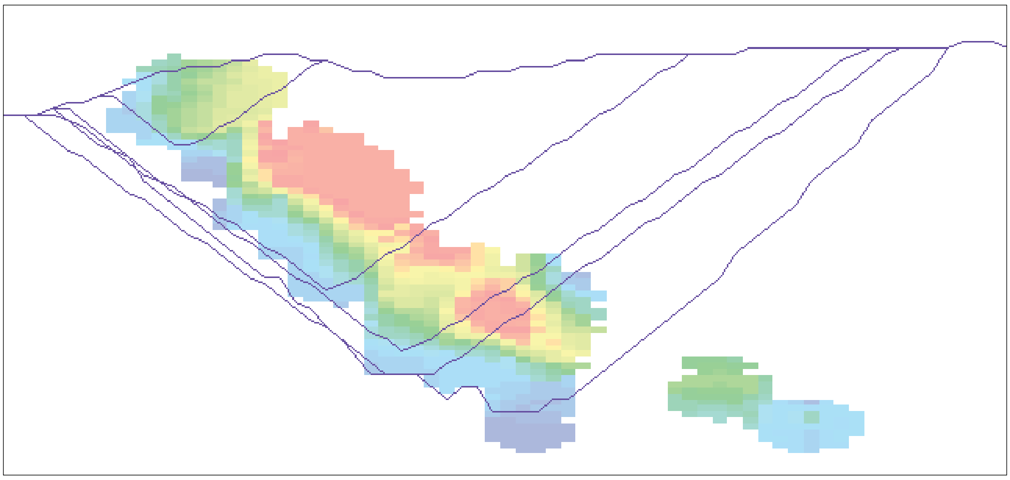


Figure 5.6-1: Cross section of pushbacks in Aynak central open-pit

Figure 5.6-1 demonstrates the cross section of pushbacks for clear understanding. Looking into Figure 5.6-1 the first pushback will be mined in 4 years and the second pushback will last for 8 years. Furthermore, the mining of 3rd, 4th and 5th pushback will take 4, 2 and 3 years respectively.

5.6.5. Mine Development, Equipment and Haulage System

The road development technique will be applied for the Aynak Central open-pit mine. Because the open-pit has an ellipse-shape with equivalent long axis and short axis. A haulage way will be organized with spiral and turn-backs in order to decrease the quantity of stripping produced by side expansion. The access point is located at the north side of the open-pit with an elevation of 2550 m.

According to MCC Feasibility Study Report on Aynak 2015, the ore will be delivered by the dump truck, Semi-mobile crusher, conveyor belt and the waste will be transported by the trucks.

Aynak open-pit will be developed by horizontal bench mining technique with a bench height of 15 m. Mining face is on the gentle slope and stripping face is on the inclined slope. The slope angle of active bench is 70°-75° and at the last stage of mining face work, two benches will be combined into one with a bench height of 30 m and bench slope angle of 60°-75°.

Conforming to MCC Feasibility Report 2015, Rotary driller with a diameter of 250 mm, Electric Shovel with 35 m^3 bucket and 218 t dump truck will be used for drilling, shoveling and transportation respectively [1].

In order to understand the final layout of Aynak central open-pit for production planning, it is presented below in Figure 5.6-2.

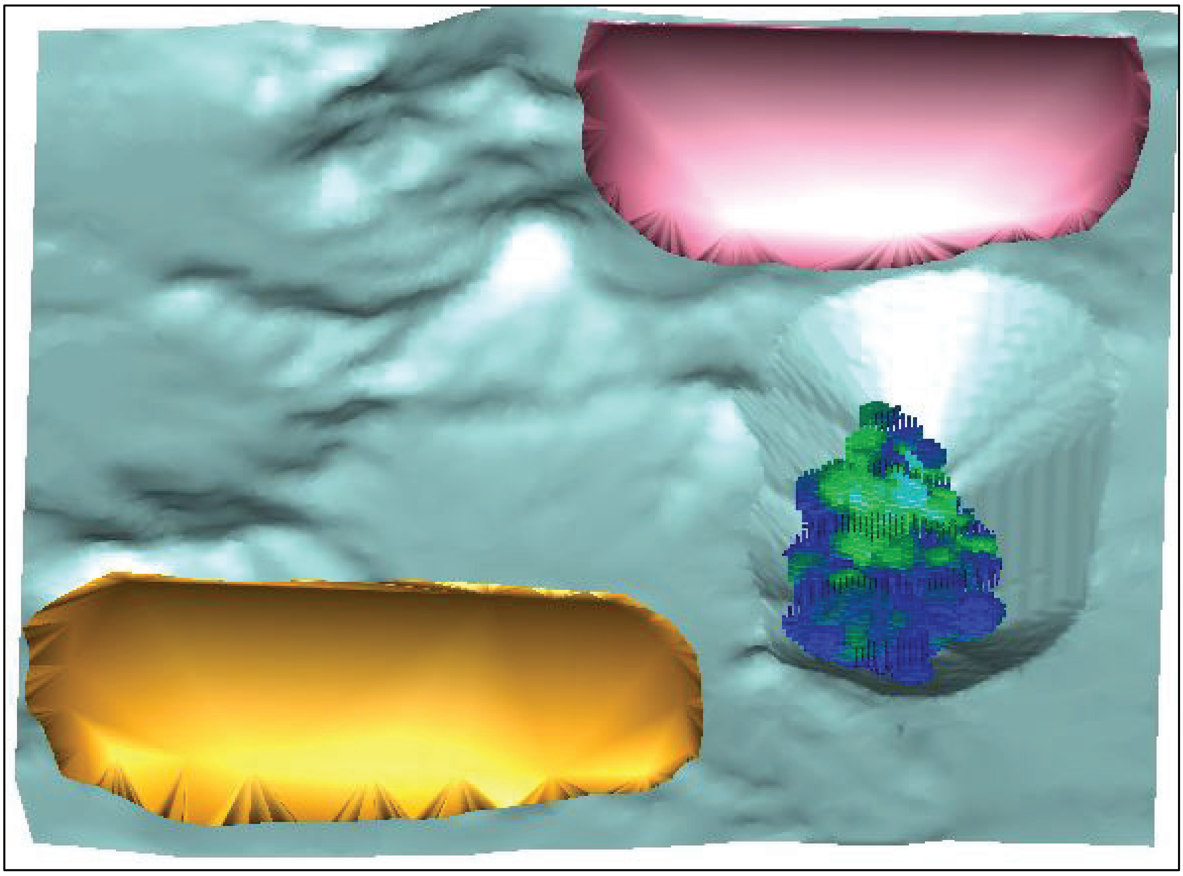


Figure 5.6-2 Layout of Aynak central open-pit mine site

Chapter 6. Financial Evaluation of the Deposit

This chapter mainly focused on the economic assessment of the deposit, sensitivity analysis, and economic viability of the deposit.

6.1. Investment Estimation

In this part, all the costs, investment, taxes rates, royalty rate, interest rate and etc. were taken from MCC Feasibility Report on Aynak Mine 2015 [2].

6.1.1. Capital Cost

The capital costs (CAPEX) estimated for Central Aynak Open pit mine is 3,077.022 million USD in the initial stage. Based on the feasibility study report of MCC, 2015, the capital costs and expenses are described briefly below in Table 6.1-1.

Table 6.1-1 Summary of the capital costs and expenses of the open pit mine [2]

No.	Description	Capital cost (10^3\$)	Share in total cost (%)
1.1	Main production facilities	959,080.43	31.17
1.1.1	Open pit mining area and explosive magazine	550,617.83	17.89
1.1.2	Sulphide concentrator	230,192.29	7.48
1.1.3	Oxide concentrator	68,052.18	2.21
1.1.4	Tailings facilities	110,218.13	3.58
1.2	Auxiliary production facilities	9,850.24	0.32
1.3	Utility systems and external infrastructure facilities	270,685.79	8.80
1.4	Administrative and welfare facilities	99,866.39	3.25
1.5	Other expenses	549,483.13	17.86
1.6	Mineral royalty	808,000.00	26.26
1.7	Contingency @ 20%	380,055.80	12.35
	cost estimated for the project	**3,077,021.78**	**100.00**

6.1.2. Working Capital

The working capital necessary after the project is put into operation is assumed based on the feasibility study report of MCC, 2015. The working capital estimated for Aynak Central Open-Pit is 81,690,000 USD [2].

6.1.3. Interest Amount

The interest amount during construction time which is mentioned in the feasibility study report of MCC, 2015 is 178,260,000 USD [2].

6.1.4. Total Costs of the Project

The total cost calculated for the project including capital cost, interest during construction time and working capital is 3,336.972 million USD. This amount is assumed for this study in conformity with the MCC Feasibility Study Report of Aynak Central Copper Mine [2].

6.2. Cost Estimation

6.2.1. Mining Cost

The mining cost covers the drilling, blasting, shovelling, transportation, drainage, geological and auxiliary operation cost. These costs were assumed for this study and taken from MCC Feasibility Study Report, 2015. The principles of cost calculation are as follows [2]:

1. Auxiliary materials: The local price will be determined for auxiliary material if the local supply fulfil the Aynak mine demand. Otherwise, the price will be decided in the relation with the present price in China or Pakistan.
2. Fuel: The fuel costs are estimated as 1.45 USD/Kg for diesel, 1.35 USD/Kg for gasoline and 649 USD/t for heavy oil.
3. Power: A price of 215 USD/ K. Kwh is calculated for power supply considering heavy oil power plant.
4. Salary and Welfare: The annual amount of wages and welfare is estimated 16.399 million USD.
5. Maintenance cost: The maintenance cost is calculated based on project cost and the repair rate. For building and mechanical equipment, the repair rate is 2% and 5% respectively.

6. Other costs: The other costs calculated for mine production are 406,000 USD.

The annual expenses of the Open pit mining are 213.774 million USD, which is equal to 18,510 USD per unit ore and 1.21 USD per ton ore rock.
A cost of 13.27 USD/t is calculated as the average mining cost during the operation time.

6.2.2. Dressing Beneficiation Cost

The dressing beneficiation cost involves the crushing, grinding, centralized cleaning, concentrate, screening and tailings relieving costs.

The basis for beneficiation cost calculation are similar to the basis of mining cost estimation. The average beneficiation cost is 60 million USD, which is equivalent to 12.5 USD/t conforming to the MCC Feasibility Study Report, 2015 [2].

6.2.3. Auxiliary Cost

The auxiliary cost covers the additional production expenses of the public auxiliary facilities for the electricity, water and heating.

The principles for auxiliary cost calculation are the same used for the other cost calculations including mining costs and beneficiation costs in this chapter.

According to the MCC Feasibility Study Report 2015, the additional production cost of the auxiliary facilities is estimated as 29.748 million USD, which is equal to 2.58 USD/t [2].

6.2.4. Management Cost

The management cost involves the executive salary, office expenditures, design consultation fee, environmental monitoring cost, communication and information service bill, security charges, and insurance premium and community expenditures.

According to the MCC Feasibility Study Report 2015, the computed cost for management amounts to 62.866 million USD, which is similar to 5.44 USD/t ore [2].

6.2.5. Sales Cost

The sales cost comprises of the transportation cost and other shipping appropriate costs during the sales. According to the MCC Feasibility Study Report 2015, the average sales cost is 147.794 million USD, which is equal to 12.8 USD/t [2].

6.3. Final Analysis

6.3.1. Production Output and Business Income

The final and entire production of Aynak Central Open-pit is 5,593,144.5 t of copper metal. The product price determined for this deposit is 6000 USD/t. The total business income is 34,677,496,005 USD.

6.3.2. Taxes

Various types of taxes were considered according to the law for minerals and are listed below.

Royalty Rate

Conforming to the MCC Feasibility Study Report 2015, the royalty rate is determined on the basis of product price in the market. Currently, the product price is higher than 4409 USD/t for which the royalty rate specified in contract is 19.5% [2].

Contract Tax

The contract tax is calculated based on the bought material, fuel, power and other cost. According to the MCC Feasibility Study Report 2015, the contract tax of 7% has to be paid by the suppliers and contractors [2].

Interest Tax

According to the MCC Feasibility Study Report 2015, Apart from the available money of the company, investment of 2,153.935 million USD is needed and should be borrowed from a bank. The interest tax of 20% has to be paid in conformity with the rules [2].

Income Tax

According to the MCC Feasibility Study Report 2015, the income tax rate is 20% [2].

6.4. Overall Evaluation of the Deposit

Based on the calculations and assumptions in Table 6.4-1, generally, it seems that the deposit will pay all the costs and expenses. Furthermore, the company will gain profit from the project and the government as well for the taxes and royalty [2].

Table 6.4-1 Overall average evaluation of the deposit

No.	Item	Description	Total (USD)
1	Net revenues	Calculated with software from ultimate pit	1,012,640,908
2	Income tax		202,528,181
3	Post-tax Profit	Net profit	810,112,727

6.5. Sensitivity Analysis

Sensitivity analysis for the deposit was done on different basis, considering the environment and nature of the deposit and interest area. The results of the analysis are briefly described below.

1. Changes in copper price can highly affect the profit of the deposit and has a direct relation to the earnings.
2. According to the mineral law and mining contract, the royalty has a huge impact on the profit because rise in copper price can increase the royalty tax of the deposit.
3. Increase in the price of heavy oil can influence the earning because the power supply and other big facilities require huge amount of heavy oil.
4. Upturn in the construction investment and operation cost doubles the expenses and can extremely change the profit.
5. Lack of industrial foundation, lengthy range of land shipment, extremely high material prices, and security issues will have unfavorable impacts on the economic profit of the project.
6. Unstable situation of the interest area will negatively influence the economic and social benefits of the project. Because it is a long-term project with huge investment, large number of work force, huge equipment, structures and properties need security.

Chapter 7. Conclusion and Recommendations

This chapter brings the current study to an end and conclude all the important issues briefly. Additionally, the relevant recommendations were reported based on the outcome of the study.

7.1. Conclusion

Aynak copper deposit is one of the biggest deposit in the world. The total mineral resources estimated for Aynak Central based on 3DBM are around 342,268,750.00 tons with an average grade of 1.77%. The mineral resources for Aynak Western were not calculated because of the low available data. Therefore, Aynak Western deposit had been ignored.

Central Aynak deposit includes the ore bodies that are lenticular in shape, strike NNS and tend in the direction of ES with a dip angle of 35°-40°. Moreover, the strike length is around 1850 m and the width is 1200 m. Aynak Central deposit will be exploited by Open-pit mining because of the reliable resources with high grade and suitable mining condition.

The Aynak Central Open-pit is designed for 40,000 t/d which is equivalent to 13.2 million t/a and the mine service life is 23 years. The mineable resources are 329,008,501 tons of ore with an average cu grade of 1.7% which is similar to 5,593,144.5 tons of copper metal. The amount of waste is around 1,020.7 million tons. The stripping ratio is 3.1 t/t and the recovery rate is 96 %.

The main technical indications of the Aynak Central Open-pit are as follows: The surface length and width of the final Pit are 2083 m and 1357 m respectively. The floor dimensions of the final pit are 258 m long and 163 m wide and the final depth of the pit is 700 m. The overall slope angle is around 45°.

7.2. Recommendations

After completing the current study, the following recommendations will be made for further work and to get reasonable information in the future.

1. A huge exploratory work was done by the Soviet Union geologists, but a data base was received that contains 150 drill holes. Among which 132 had related to Aynak central and the remaining 18 drill holes were related to Aynak west. Therefore, further supplementary exploration is required to for better assessment of the Aynak west.

2. In the available database, there is no information about the associated contents alongside copper such as silver (Ag), sulfur (S) and cobalt (Co) which are mentioned in the geological reports. Therefore, an additional investigation is necessary to determine the amount of the mentioned elements and it will increase the benefit of the project and government as well.
3. Darband and Jawhar are the two other copper deposits located in the vicinity of Aynak Copper deposit. Doing exploration and drilling work for further analysis will expand the copper resources in the area. In addition, taking advantage of the Aynak facilities will pave the way to be exploited in the future immediately after the Aynak Central and Western are mined.
4. A detailed rock mechanic study is recommended to get reliable information. Because of deepening the depth of mining, the final slope stability needs the verification work in each profile. Furthermore, the western Aynak will be exploited by underground mining.
5. Ground water has an influence on the stability of slopes therefore, monitoring the ground water is required.
6. Recognition of the coordinate system of the drill holes because it is required to register the drill holes into the exact area for better understanding and for relevant resource estimation.

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List of Abbreviations and Symbols

°C *Degree Centigrade*
$ *United States of America Dollar*
% *Percent*
.mdb *Access file extension*
.prj *MineSight Project file Extension*
.xlsx *Excel file extension*
+ *Plus or Positive*
3D *Three Dimensional*
3DBM *Three-Dimensional Block Model*
ACCD *Aynak Central Copper Deposit*
ACCP *Aynak Central Copper Deposit*
Ag *Silver*
AGS *Afghanistan Geological Survey*
amp *Amphibolite*
ASCII *American Standard Code for Information Interchange*
Aynak15.pto *Aynak model file used for optimization in MineSight*
aynk09 *Aynak Composite file used in MineSight*
aynk11 *Assay File for Aynak Copper Deposit*
aynk12 *Survey file of Aynak used in MineSight*
Aynk13.pto *Aynak topography file used for optimization*
aynk15 *Aynak model file used in MineSight*
Aynkreserve.resx *Aynak Reserve file used in MineSight*
B.C *Before Christian*
BESR *Breakeven Stripping Ratio*
BM *Block Model*
bre *Breccia*
CD *Compact Disc*
CIF *Cost, Insurance and Freight*
cla *Clay*
Co *Cobalt*
COID *Oxide Copper Interpolated by Inverse Distance Weighting*
CONSCA *CONvert, Survey, Collar, Assay*
CTID *Sulphide Copper Interpolated by Inverse Distance Weighting*
Cu *Copper*
Cuox *Oxidized Copper*
Cuto *Sulphide Copper*
d *day*
dol *Dolomite*
DWG *Drawing (CAD programs filename extension)*
DXF *Drawing Exchange File*
ES *East South, East South*
FOB *Free on Board*
F_s *Safety Factor*
g/cm^3 *gram per centimetre cubed*
gne *Gneiss*
GPS *Global Positioning System*
GSF *Gridded Surface File*
GSF 13 *Aynak Gridded Surface Topography File used in MineSight*
IDC *Inverse Distance Cubed*
IDS *Inverse Distance Squared*
IDW *Inverse Distance Weighting*
IRR *Internal Rate of Return*
JCC *Jiangxi Copper Corporation*
JORC *Joint Ore Reserves Committee, Joint Ore Reserves Committee*
km *Kilometre*
km^2 *Kilometre Squared*
lbs *Pounds*
LG *Lerchs-Grossman*
LME *London Metal Exchange*
loa *Loam*
lt *Long Ton*
Ltd *Limited*
m meter
m/sec *metre per second*
m^3 *metre cubed*
MCC *Metallurgical Corporation China*
MCOST *Mining Cost*
mg *milligram*

mm	*Millimetre*
MoMP	*Ministry of Mines and Petroleum*
MS3D	*MineSight Three-Dimensional*
MSDA	*MineSight Data Analyst*
MSEP	*MineSight Economic Planner*
mt	*Metric Ton*
N	*North, Newton*
NCOMP	*Number of Composites*
NNS	*North-North South*
NPV	*Net Present Value*
OK	*Ordinary Kriging*
PCF	*Project Control File*
qua	*Quartzite*
S	*Sulfur*
san	*Sand*
sch	*Schist*
SG	*Specific Gravity*
SRK	*Steffen, Robertson, and Kirsten*
st	*Short Ton*
t	*Ton*
tal	*Talus deposits*
TOPO	*Topography*
USA	*United States of America*
USSR	*Union of Soviet Socialist Republics*
UTM	*Universal Transverse Mercator*

Appendix A Characteristics of Received Data

BH_Number	Top_depth	Bottom_depth	Interval	Sample_No	Cu_total	Cu_ox	Oxidation	Date_entered	Entered_by
100	101	103,5	2,5	11557	0,5	0,29	58	27.04.05	najeeb
100	103,5	104,2	0,7	11558	0,11	0,03	27	27.04.05	najeeb
100	104,2	106,2	2	11559	0,43	0,3	70	27.04.05	najeeb
100	106,2	108,2	2	11560	0,9	0,57	63	27.04.05	najeeb
100	108,2	110,2	2	11561	0,38	0,2	53	27.04.05	najeeb
100	110,2	112,2	2	11562	0,28	0,14	50	27.04.05	najeeb
100	112,2	120,4	8,2	11563	0,91	0,67	74	27.04.05	najeeb
100	120,4	121,8	1,4	11564	4,49	3,71	83	27.04.05	najeeb
100	121,8	123,6	1,8	11565	2,64	1,26	48	27.04.05	najeeb
100	123,6	125,8	2,2	11566	0,56	0,88	34	27.04.05	najeeb

Table 7.2-1:Structure of tblanalysis excel spreadsheet in database

Started date	Finished date	X (m)	Y (m)	Height (m)	Azimuth (°)	Inclination (°)	Recovery(%)	Depth (m)	Scan	Date_entered	Entered_by
03.07.74	05.12.74	92460,6	28282	2443,7	290	-65	77	330,5	#M:\Borehole_tiffs\n-1.tif#	07.02.05	J S Coats
14.03.75	13.07.75	92845,2	28381,9	2486,5	290	-75	84	441,5	#M:\Borehole_tiffs\n-10.tif#	07.02.05	J S Coats
15.08.76	06.09.76	91941,9	28261,1	2424,87		-90		219,3		27.04.05	najeeb
05.09.76	13.12.76	92083,4	28438,5	2461,22	290	-79	75	320,2		27.04.05	Omar+Mirahmad
01.08.76	23.11.76	92056,7	28244,1	2445,28	290	-79	78	397,9		30.04.05	amiri and aziz
24.06.77	28.08.77	93125,53	28200,96	2474,26	289	-89	51	406,7		30.04.05	najeeb
25.08.75	01.10.75	93150,23	28118,07	2465,8	0	-90		341,7		30.04.05	najeeb
10.11.75	26.02.77	93050,5	28400	2501,5	78	-89	87	579,7		30.04.05	najeeb
05.12.76	14.02.77	91559,36	28228,48	2395,75	312	-70	85	405,5		30.04.05	najeeb
15.10.76	27.02.77	91489,2	28354,7	2404,51	123	-75	71	322,7		01.05.05	Peter Dunkley

Table 7.2-2: Structure of tblBorehole excel spreadsheet in database

BH_Number	Unit_no	Top_depth	Bottom_depth	Thickness	Age	Lithology	Description	Banding_angle	Date_entered	Entered_by
1	1	0	12,4	12,4	N	Arenaceous-argillaceous deposits	Arenaceous pebbled (sic) deposits with argillaceous-carbonate cement take place to the depth of 12.4 metres			
1	2	12,4	117,75	105,35	N	Arenaceous-argillaceous deposits	In the interval of 12.4 to 117.75 metres predominant is yellowish-grey arenaceous clay with thin interbeds of rather compact mica sandstone and, more rarely, of conglomerates.			
1	3	117,75	152,4	34,65	6	Quartz-biotite-dolomite schist	Dark grey to greyish-brown coaly quartz-biotite-dolomite slate mainly of breccia-like appearance. In the upper portion of the layer sulphide mineralization consists of pyrite semi-oxidised grains, native copper and cuprite, while in the lower portion predominant are chalcocite and chalcopyrite	**50**		

Table 7.2-3: Structure of the tblGeology in the database

Appendix B Rock Mechanic Study Details

Rock type	Natural humidity (%)	Bulk specific gravity (g/cm3)	Porosity factor (%)	Compressive strength (Mpas)		Cohesive force (Mpas)		Internal frictional angle (°)	
				Natural state	After air-dried	Natural state	After air-dried	Natural state	After air-dried
Clay	7	1.98-1.99	27-36 (Ave: 30.2)	2.3-2.4	7.7-8.03	0.1-0.3	0.75-1.14	46.2-48.7	40.8-42.8
Marlite	**13.4**	**2-2.02**	**23-35 (Ave: 28.5)**	**3.08-3.2**	**7.7-8.3**	**0.31-0.4**	**0.34-0.75**	**44.6-49.2**	40.6-44.1
Siltstone	**16**	**2.06-2.07**		**2.5-2.52**	**6.4-6.5**	**0.27-0.41**	**0.78-1.11**	**41-44**	37.2-40.3
Sandstone	**10.8**	**1.95**	**26-34 (Ave: 26.8)**	**1.99-2.03**	**6.54-7.1**	**0.2-0.4**	**0.16-0.55**	**44.2-47.1**	41.4-44.9
Limestone	**17.2**	**2.09**		**6.6-7.9**	**13.23-14.9**	**0.1-0.8**	**0.1-1.33**	**43.7-50.7**	38.8-43.4
Pelite	**11.74**	**2.01-2.02**	**30-33 (Ave:30.7)**	**3.14-3.23**	**7.3-7.9**	**0.2-0.33**	**0.6-0.94**	**45-48.6**	43.2-46.7
Conglomerate and cumulate gravel	**5.3**	**1.94-1.95**				**0.001**			**48.7-49.2**

Table 7.2-4:Physical and mechanical properties of Tertiary rock formation [2]

Zones	Characteristics
Pre-Cambrian rock bed	This zone contains the late Tertiary and Quaternary products and the mountain profile with an elevation of 2100m-2500m or 2700m. The mountain fluctuation is around 400m and the mountain ridge is isolated by canyon and pass. The mountain pass is dangerously steep. The UCS is 74.6-145.6 Mpas and the strength coefficient is 8.17-13.22.
Soil eroded and accumulated plain	This zone comprises Tertiary sedimentary stratum and the exact surface elevation is 2230m-2400m and the deflection angle is 3°-5°. In this zone, conglomerate ore body, sand soil, siltstone, clay, mud rock, lime mud, and gravel appear.
Mountain slope	This zone is distributed nearly the slope of mountain area edge and area of denudation. The slope inclination is 20°-35° and separated by alluvial stream channel and its height is 3m-18m. The slope sedimentary layer is filled with Friable debris and sub-clay. The base of sedimentary layer of the area is mainly consisted of stony rock and some post-Tertiary clay.
Valley alluvial-diluvial cone	This zone has been wiped by west-hill river current or by gutter current. It involves west-hill river bed-where the sedimentary layer is built of clay rubble and sandy soil rubble mixed with boulders (5%-20%) and the stream channel-made by the washing process of water flow. This zone is stored with loam clay and soil has been marginally salinized.

Table 7.2-5: Characteristics of geotechnical zones in Aynak Central area [2]

Geological zones	Engineering fetrofabrics	GSI	D	Mb	S	Av. Height of rock stratum(m)	φ (°)	C (Mpas)
Zone 1	Amphibolite	46	1.0	0.53	0.00012	264	30.9	0.89
	Carbonaceous quartz schist	56	1.0	1.08	0.00065	507	32.8	1.85
Zone 2	Amphibolite	47	1.0	0.57	0.00015	253	31.8	0.89
	Carbonaceous quartz schist	40	1.0	0.34	0.00005	73	37.9	0.35
Zone 3	Gravel	55	0.7	0.84	0.0015	37	27.7	0.11
	Argillaceous siltstone	66	0.7	1.54	0.0072	100	28.2	0.3
	Sandy mudstone	65	0.7	1.46	0.0063	145	29.0	0.45
	Mid-fine sandstone	77	0.7	4.24	0.0357	175	22.3	0.34
	Marlstone	68	0.7	1.72	0.0097	214	29.0	0.67
	Glutenite	85	0.7	6.58	0.1137	252	22.2	0.49
	Breccia	85	0.7	6.58	0.1137	279	26.3	0.70
	Amphibolite	48	1.0	0.61	0.00017	458	28.0	1.31
	Carbonaceous quartz schist	57	1.0	1.16	0.00077	576	32.4	2.06
Zone 4	Gravel	-	-	-	-	-	-	-
	Argillaceous siltstone	51	0.7	0.68	0.0008	57	26.0	0.15
	Sandy mudstone	53	0.7	0.76	0.0011	101	26.6	0.27
	Mid-fine sandstone	73	0.7	3.40	0.0200	131	22.8	0.26
	Marlstone	65	0.7	1.46	0.0063	170	29.4	0.55
	Glutenite	90	0.7	8.66	0.2347	208	25.4	0.49
	Breccia	77	0.7	4.36	00384	235	24.6	0.53
	Amphibolite	46	1.0	0.53	0.00012	426	27.4	1.18
	Carbonaceous quartz schist	58	1.0	1.24	0.00091	545	35.8	2.05

Table 7.2-6: Parameters of rock mass intensity [2] (continued)

Geological zones	Engineering fetrofabrics	GSI	D	Mb	S	Av. Height of rock stratum(m)	φ (°)	C (Mpas)
Zone 5	Gravel	-	-	-	-	-	-	-
	Argillaceous siltstone	-	-	-	-	-	-	-
	Sandy mudstone	40	0.7	0.37	0.00017	36	28.3	0.11
	Mid-fine sandstone	73	0.7	3.40	0.0200	80	26.2	0.20
	Marlstone	65	0.7	1.46	0.0063	104	33.1	0.42
	Glutenite	81	0.7	5.28	0.0637	143	24.5	0.32
	Breccia	71	0.7	3.05	0.0150	170	24.3	0.38
	Amphibolite	44	1.0	0.46	0.00009	328	28.1	0.95
	Carbonaceous quartz schist	48	1.0	0.61	0.00017	462	28.9	1.39
Zone 6	Gravel	-	-	-	-	-	-	-
	Argillaceous siltstone	-	-	-	-	-	-	-
	Sandy mudstone	38	0.7	0.33	0.00013	31	28.4	0.09
	Mid-fine sandstone	71	0.7	1.07	0.00095	75	18.7	0.12
	Marlstone	61	0.7	1.17	0.0035	100	31.7	0.38
	Glutenite	76	0.7	4.01	0.0308	138	22.9	0.28
	Breccia	70	0.7	2.89	0.0129	165	24.1	0.36
	Amphibolite	40	1.0	0.34	0.00005	186	29.9	0.60
	Carbonaceous quartz schist	39	1.0	0.32	0.00004	152	31.7	0.54

Table 4.8-5: *Parameters of rock mass intensity*

Appendix C Reports Wrote on CD

Appendix C-1: Sulphide copper resorce estimation report
Appendix C-2: Oxidized copper resource estimation report
Appendix C-3: Ultimate pit determination report
Appendix C-4: MSEP MultiV pits report
Appendix C-5: MSOPIT pushbacks report
Appendix C-6: MineSight project file of the thesis