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: ____________________

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Sayed Zabihullah Shadab,

Freiberg, Germany 15.11. 2017

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Introduction

Considering the great demands for minerals, mining companies desire to have the technology, which is cost-effective and applicable to the site conditions. It is known to all, and also it is addressed in different articles and scientific papers, that materials handling constitute 45 -60 % of total operating costs in large open pit mines [1]. Based on these situations, firms manufacturing equipment that have less operating costs and meet the mining company's requirements.

While conducting a pre-feasibility study of a mineral deposit, it is vital to consider site specifications and geological conditions of the mineral deposit, in order to determine which technology will be applicable, and the haw many numbers of equipment will be needed because it has a direct link to the overall mining capital costs of the deposit.

Proper selection of the technology or equipment during the pre-feasibility stage results to a positive outcome, because the findings of pre-feasibility makes the foundation for the actual and detailed feasibility study.

The most important tool among others used for the extraction and development of a mineral deposit are the technologies or machineries. Considering site specifications and geological conditions of the mineral deposit, different possible technologies are available. Cost is a key element in equipment selection procedure because of that, efforts are being made to select the most appropriate and cost effective technology for the development of the mineral deposit.

In this thesis, some of the equipment used in the development of a surface mine are discussed and compared briefly. For the mechanical breakage, equipment such as surface miners and rippers are discussed. Rotary and percussive drill rigs, shovels, and dump trucks are considered for drilling and blasting, and materials handling. The discussion is focused on their types, applicability in different types of rocks, productivity, and the advantage and disadvantages of the individual technology.

After evaluation of equipment for the mining technology, the most feasible and appropriate technology was selected based on the geological and mechanical properties (such as hardness, strength, Geological Strength Index, and few other factors) of rocks and soils. The same procedure was also followed for the loading and transport technologies.

Chapter 1. Description of the Area of Interest

Geology and site conditions, climate, general information about the area such as infrastructures, local resources, accessibility, and reliability of geological data have been studied in current chapter.

1.1. Location and Brief Geology

The Aynak Copper deposit is located in Mohammad Agha district of Logar Province, South-South-East of Kabul. According to the archeological evidence that has been found in the area, the extraction of copper has been done in previous times. The Aynak copper mine was divided into three different parts of central, west and south. [2]

The Aynak copper was discovered by Afghan-Soviet geologists in 1970s, and the total amount of the deposit was reported to be about 240 Mt containing 2.3% copper. Further geological studies have been conducted during 1970 – 1980 through drillings and digging some trenches and audits. Some geological studies have been done by Metallurgical Corporation of China (MCC) for prove of previous studies. Aynak copper deposit is located in Kabul block. The length of the Kabul block is approximately 200 km and is 50 km wide, the location of deposit is illustrated in Figure 1.1-1 [3] [4].

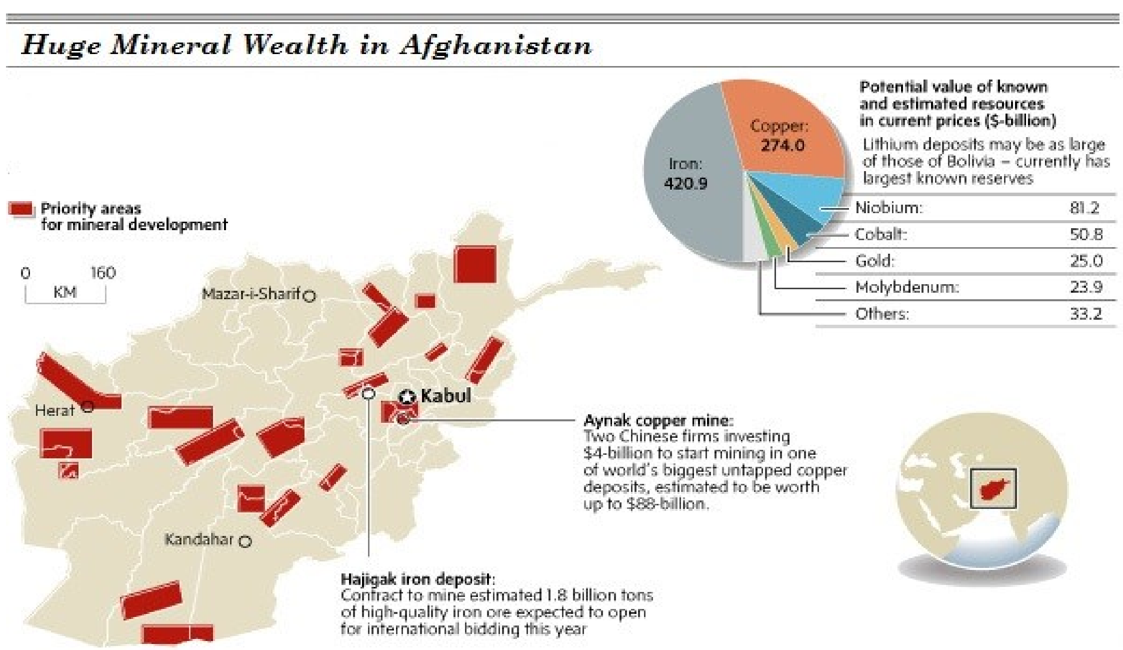


Figure 1.1-1 Location of Aynak Copper Deposit [5]

1.1.1. Genesis of Copper Deposit

According to BENHAM et al. (1995), the genesis of Aynak copper deposit has not been studied in depth, but in reference to the style, size, and grade of the deposit, it resembles Zambian copper belt [3].

The metallogenic model illustrated in Figure 1.1-2, shows that the leaching of copper was most likely occurred through the underlying volcanic rocks and were circulated via brines, evaporitic rocks and seawater [3].

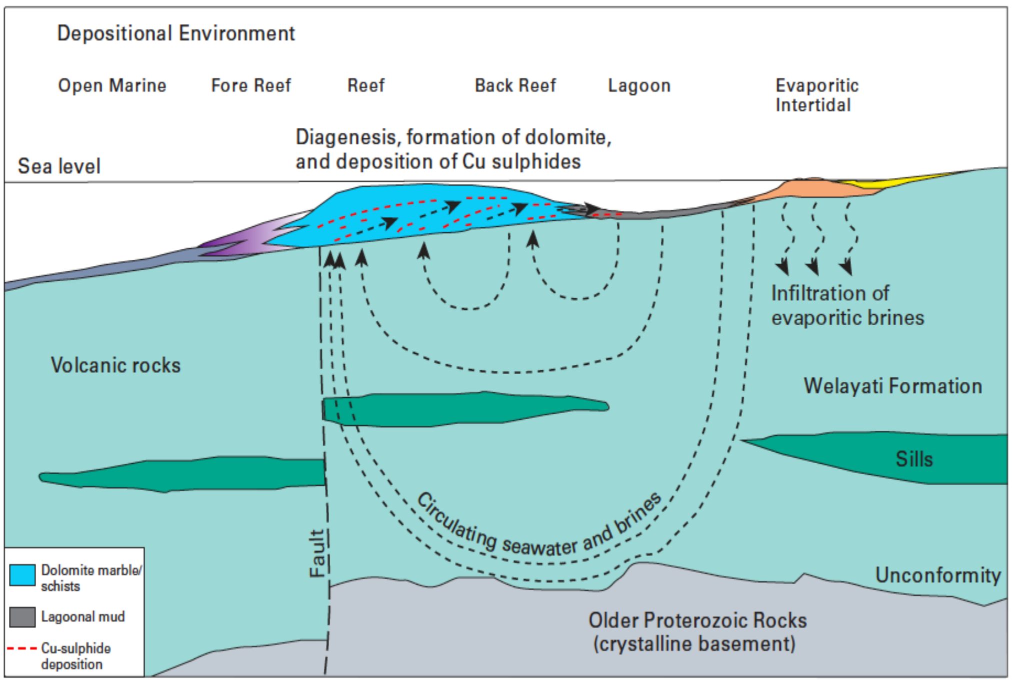


Figure 1.1-2 Possible Metallogenic Model for Aynak Copper Mineralization [3]

1.1.2. Mineralization

Stratabound and/or stratiform sediment-hosted copper deposits contain huge amount of copper [6]. According to the British Geological Survey (BGS), mineralization in the Aynak Copper area as follows: “Mineralization is strata-bound and characterized by chalcopyrite and bornite distributed in dolomite marble and quartz-biotite-dolomite schists of the *Loy Khwar Formation*” [7]. The Ore body of Aynak deposit in the central part mainly consists of bornite and, chalcopyrite exists only in very small amounts in the middle and lower parts, while in the western Aynak unlike the central part, the mineralization comprises of 80 % of chalcopyrite and 20 % of bornite [7].

Primary mineral zoning can be seen within the deposit. Cobalt concentrations also increase in the peripheral areas and in some places, evidence of cobaltite can be found. A small amount of smaltite occurs in combination with pyrite and chalcopyrite [7]. The cobalt concentration in a southeastern and central part of central Aynak is less and it lies between 0.004 and 0.013% [7].

According to the British Geological Survey (BGS), the secondary mineral zoning is located in the oxidized part of the deposit. In the oxidized zone, the malachite can be found together with small quantities of the following minerals [7]: Azurite, chalcocite, covellite, cuprite, and native copper.

Limited information is available about the concentration of the gold in the Aynak deposit. Gustave et al. (1979) and BGS reported that 606 samples of the ore were analyzed for gold. Only 256 samples of the 606 contained detectable concentrations [7]. The result of the analysis of these samples is indicated in below.

Number of Samples	**Concentration of Gold**
141 Samples	0.2 ppm
78 Samples	0.4 ppm
14 Samples	0.8 ppm
1 Samples	1.0 ppm

1.2. Accessibility

Unfortunately, Afghanistan does not have railway as one of the most basic modes of transport as compared to many countries and thus the only way to get to Aynak is through normal roads.

The distance from Kabul to the mine site is about 49 km. Accessibility to Aynak copper deposit is divided into two parts or sections. The first section is maintained via Kabul-Khost highway, which is an asphalt road, and it accommodates high traffic. The second section is via an unpaved road and crossing through a village. Afghanistan-Pakistan border is located in south of Kabul-Khost highway [8].

1.3. Infrastructure

The overall infrastructure is inadequate in Afghanistan. In transportation section, there is no railway at all, neither for daily commuting nor for good transportation. The only mean of transportation found in the country is road transport on either asphalt roads or gravel roads, which are not constructed as according to international standards.

Another vital and key element for the development of the mine is electricity. Regretfully, there is no electricity in Aynak Copper field. Currently, Afghanistan is buying the electricity from its neighboring countries such as Tajikistan and Uzbekistan. According to the mining contract which is granted to Metallurgical Corporation of China (MCC), a coal-fired power generation plant will be built in the area of Tala. The main reason for building this power plant is the abundance of coal deposits in the northern part of the country [8].

A new town will be built for the people who will be displaced from their villages near the mining area. There is no regional airport in the area, the only and nearby airport to the area is Kabul airport, located approximately 50 km away [8].

The area also lacks of water pipelines and the company has to drill some water wells or bring the water from the Logar River. Besides the sewage, there is neither internet connections nor landlines where the mine is situated, thus limiting communication.

Having no good infrastructures in the country has caused different drawbacks for the Ministry of Mines and Petroleum of Afghanistan (MoMP) during negotiations of mining Contracts. The drawbacks are explained in the following:

Based on Minerals Law of Aghanistan, the companies are obliged to provide some socieal services to the local communities, and Government of Afghanistan is also responsible to prepaer the required infrastructure for the mining companies.

In negotiation period of mining contracts, both sides (the MoMP and Mining Company) try to achieve as much as possible benefits from the opposite side. Since Afghanistan does not have adequate infrastructure, it cannot argue to gain more benefits from the companies which have interest on investing in mining sector of Afghanistan, i.e. in local communities, there is need for the clinics, schools, and other necessory facilities, which should be built by Mining Company according to the contract. Companies argue that the Government of Afghansitan can not provide to them required infrastructure, thus compaies do not provied any social services [9].

1.4. Local Resources

The following local resources are available in the Aynak area:

- Work force: work force is available in the area with a very low salary for example 300 dollar per month or less than this amount.
- Skilled labor: though there is insufficiency of skilled labor in the area but few not updated workforce like truck drivers, driller and excavator drivers can be found.
- Provincial Hospital: a state hospital is available in the center of Logar province, Pul-e-Alam.

1.5. Climate

The mine site is hit by continental climate, there is lack of rain and the area is dry. Sometimes during winter, heavy snow and cold weather could be expected. The annual average temperature and the monthly average temperature is stated in Table 1.5-1 [10].

Table 1.5-1 Measurement of temperature of the area [8]

Temperature of the area	
Annual Average Temperature	+10.5 to +12.8 ℃
Monthly Average Temperature in Winter	-5 to -7 ℃
Monthly Average Temperature in Summer	+23 to +24 ℃
Lowest Temperature	-37 to -42 ℃
Highest Temperature	+36 to +36.6 ℃
Average Annual Rain Fall	197.2 to 229 mm

Regarding water table in the Aynak area, it can be stated that the water level below surface is located between (4-42) meters [11].

1.6. History

Undeniably, that Afghanistan is a country, rich in mineral resources. Based on the archeological studies, the first extraction of mineral resources in Afghanistan was performed four thousand years B.C [12]. Aynak is one of the different areas where extraction of minerals was performed in the past. The exploitation of copper from Aynak area was performed 2000 years ago and approximately has supplied the required copper of people for the 8 decades [12]. Evidences like small audits and surface excavations existed in the area, also Copper coins and skeletons were found in the area before recent exploration works. The issued copper coins in the 1^{st} and 4^{th} century was made by Kushan dynasty [7].

The government of Afghanistan and the people of the country are looking forward to have huge benefits from Aynak Copper deposit. As the unemployment is increasing in the country and the economic situation of the country is in down, extraction of Aynak copper is the opportunity to create employment for the people of Afghanistan.

1.7. Reliability of Geological Information

The data were collected from Aynak Department of MoMP. The Afghan-Soviet geologists had done the drillings in 1987. Regarding the reliability of the data, it can be said that the Drill Hole data is quite reliable because the drilling logs of Afghan-Soviet geologists were proved by MCC through drilling 40 additional exploration wells near to old ones [13].

Chapter 2. Surface Mining Potential and Methodology

In this chapter, the mining method, location of the waste dump, annual production and schedule of the mine, overburden removal, and operating efficiency of the equipment have been discussed.

2.1. Surface Mining Methods

Surface mining method are classified into the following two classes: [14]

- Mechanical
- Aqueous

Mechanical Methods will be discussed in detail in the following, as these will be used for the Aynak deposit. The aqueous class is dependent on water i.e. water is used for mining and processing of mineral through jetting, slurring, and dissolving [15].

Aqueous class is further divided into two subclasses which are indicated in the following [14]:

- Placer
- Solution

Hydraulicking and Dredging comprises the placer mining methods while borehole mining and leaching is used in solution methods. The aqueous methods are mostly used for gold, tin ore, diamonds, sand and gravel, and other heavy minerals, thus that aqueous methods cannot be used for mining of this deposit [15].

The following are the extraction methods in the mechanical class; “open pit mining and open cast method, quarrying, and Auger or high wall mining” [14]. More than 80 percent of the surface mining in United States is performed by the above-mentioned methods. [14]

The following is the definition of open pit and open cast mining methods according to Hartman (2002):“Open pit and open cast mining methods are the essential methods among four mechanical methods and also counted as most important methods between all eight surface mining methods. Open pit mining is done through one or numbers of horizontal benches. The removal of ore and overburden is performed by the benches which vary in size starting from 9 meters up to 30 meters” [14].

The number of benches relate to the thickness of the deposit. The thickness of deposit and the number of benches increase with thickness of the deposit. The shape of the open pit may look like an inverted cone when the deposit gets deep. If the deposit gets deep, benches are added. Shallow deposits (thickness is between 15 to 45 meters), can be extracted by single bench [16].

The height of the bench should be designed in a way that the excavator beam can reach the bench height, thus the excavator puts some limitations to the height of bench. While designing the bench, the maneuvering of haulage trucks, excavator, shovel, and fly rock from other benches should be taken into consideration. The slope of the bench is determined considering rock or soil mechanics of the area. Various rock test suck as UCS, tensile strength, and few others should be done. Common practice of bench height, width, and slope according to Hartman (2002) is given in Table 2.1-1 [17].

Table 2.1-1 Common Practice for Bench Dimensions [17]

Mineral	**Bench Dimensions**		
	Height, ft. (m)	**Width, ft. (m)**	**Slope**
Copper	40-60 (12-18)	80-125 (24-38)	50°-60°
Iron	30-45 (9-14)	60-100 (18-30)	60°-70°
Nonmetallic	40-100 (12-30)	60-150 (18-45)	50°-60°
Coal (Western U.S.)	50-75 (15-23)	50-100 (15-30)	60°-70°

Open cast (Strip) mining is same as open pit mining with a very specific difference that the overburden which is extracted from open cast mining is not transported to waste dump instead is cast directly in the mined out areas. This method is mostly used in coal mining and for the extraction of approximately horizontal deposits.

Quarrying is a mining method that is used mostly for the extraction of dimension stones and it is defined by Hartman (2002) as:"a mining method same as open pit methods but the benches (which is called faces in this method) are smaller and vertical." The dimension stone that is produced by quarrying method is generally used in monuments, decorations and many others. [17].

Auger or high wall mining is defined by Hartman (2002) as:"a mining method that recovers coal or other minerals from under the highwall when the ultimate stripping ratio has been achieved" [17]. In this method, the holes parallel to the coal seam are drilled into the coal seam and coal is excavated from the holes. Amongst all surface mining methods, open pit mining method is the most used method in the mining of copper deposits.

It has the following advantages over other methods [18].

- Flexibility
- Small shutdown expenses
- Complete extraction of the ore inside the pit limits
- Less number of workforce is needed
- Expeditious production

Considering the above mentioned facts and the type of deposit (stockwerk deposit) and the deposit model which is currently being modeled by other colleague of mine, open pit mining methods seems to be suitable for the extraction of Aynak Copper deposit.

2.2. Overburden Removal

Overburden removal is one of the initial steps to prepare any mine for the production, and it implies to the removal of an overlain layer to expose the ore. It is vital for the planning department of mine to bring the expenditures of overburden removal to a lower possible cost because generally a company or respective mine cannot have any return interest from the costs spent for the overburden removal [16].

Based on the characteristics of the rock, different techniques are used for the excavation of over burden e.g. ripping, dozing, and drill and blast. Generally, ripping is used for topsoil and soft over burden and drill and blast method is used for medium hard and hard materials.

Several equipment is used for the excavation of overburden, like draglines, power shovels, dozer/front-end loader, and bucket wheel excavators. Transportation of overburden is accomplished by trucks, conveyor belts, and front-end loaders [19].

The selected machine must effectively cooperate with overall site conditions. It is pertinent to use the same equipment for both ore and the waste because it brings down the maintenance requirements and ultimately, results in the decrease of the operating costs [19].

Overburden removal in open pit is conducted by two methods; dumping the overburden in pit and outside of the pit. In most cases, the overburden is transported outside of the pit because it takes up lots of space and creates a disturbance to other operations. As a rule of thumb, “mine the next ore”, mining companies try to postpone the stripping of waste as much as possible in order to decreas the costs in the commencement of operations.

For Aynak Copper deposit a comparison will be made for the excavation of overburden in order to identify whether mechanical cutting is suitable or the drill and blast procedure(?), but for the transportation of overburden the self rear dumpt trucks will be used because of its following characteristics [20]:

- Versatility, differenet material can be transported by trucks
- Good gradeaility
- Googd traction
- Ability to perform in unfavorable road condtion
- Ability to maneuver in small areas
- Flexibility almost to every condition
- Suitability to withstand the impact of loading
- Appropriate for dumping of material at the edge of waste dumps

2.3. Waste deposition (horizontal, vertical and total distance)

During development of the mine, besides extracting the main mineral (Ore), the mass amount of the rocks (waste rocks) which is deposited with the minerals, should be extracted. As rule of thumbs, there is an economic return for the extraction of minerals, but in the case of waste rocks, generally, there would be no economic return for the costs that will be consumed for the handling of material.

Mine wastes are of different types e.g. overburden, waste rocks, tailings, slags, mine water, water treatment sludge and gaseous water. Amongst all of types of wastes, only the overburden has less impact on the environment but the rest have a severe impact based on their components. Based on their properties different methods are applied to avoid environmental contaminations [21].

In some mining methods, there is the possibility of depositing the waste inside the area that has been mined out, but in open pit mining method, the waste material should be dumped outside the pit. In order to decrease the transportation costs, the distance to waste dump should be selected as short as possible considering factors affecting the location of waste dump.

In Aynak copper deposit, it is supposed that waste dump will be located to the south of the deposit and the total distance from the pit to the waste dumps is assumed 5 km. The waste is directly to be sent to the waste dump [10].

2.4. Open Pit Schedule

Based on MCC reports, the amount of ore which is feasible to mine is $140*10^6$ t with an average stripping ratio of 7.6 [8].

Considering the given stripping ratio the amount of waste to be extracted can be calculated using Equation (2.4-1):

$$Stripping\ Ratio = \frac{Waste}{Ore} \qquad \textbf{(2.4-1)}$$

$$\text{Waste} = \text{Stripping Raio} \times \text{Ore} = 7.6 \times 140 * 10^6\ \text{t} = 1{,}064 * 10^6\ t \cong 1{,}050 * 10^6\ \text{t}$$

Total amounts:

ORE: $140*10^6$ t Planned Annual Production of Ore: $7*10^6$ t

WASTE: $1,050*10^6$ t ″ Annual Production of Waste: $52*10^6$ t

The life of mine is assumed to be 20 years. It is assumed that the annual production of ore is $7*10^6$ t and for waste rocks, it is $52*10^6$ t. Table 2.4-1 and Table 2.4-2 show the annual production of ore and waste.

Table 2.4-1 Annual productions of ore and waste

Description	**1st year**	**2nd year**	**3rd year**	**4th year**	**5th year**	**6th year**	**7th year**	**8th year**	**9th year**	**10th year**
Ore	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$
Waste	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$

Table 2.4-2 Annual productions of ore and waste

Description	11th year	12th year	13th year	14th year	15th year	16th year	17th year	18th year	19th year	20th year
Ore	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$	$7*10^6$
Waste	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$	$52*10^6$

2.5. Required Performance and Operating Hours

Aynak mine is scheduled to operate 355 days per year and three shifts per day. Table 2.5-1 and Table 2.5-2 show the required performance for the planned production of ore and waste, respectively.

Table 2.5-1 Required performance (ore)

Description	Amount [Metric Tons]
Annual Production	$7*10^6$
Daily Production	19,800
Per shift Production	6,600
Hourly Production	825

Table 2.5-2 Required performance (waste)

Description	Amount [Metric Tons]
Annual Production	$52*10^6$
Daily Production	146,500
Per shift Production	48,900
Hourly Production	6,112

Operating hours are calculated considering the religious and national holidays in Afghanistan. Operating hours are given in Table 2.5-3.

Table 2.5-3 Working Time and Operating Hours

Calendar days	**Total days**	**365 days**
Not usable time	Weekends	0 days
	Mine shutdown	0 days
	Public holidays	10 days
	Scheduled ordinary time	355 days
Scheduled time per day	Repair/unscheduled time	- days
	Hours per shift	8 hours
	Shifts per day	3 shifts
	Scheduled time per day	24 hours/days
Total hours	Scheduled hours per year	8520 hours
Unavailable hours	Planned maintenance (assumption)	500 hours
	Unplanned maintenance (assumption)	500 hours
	Others (break down, supply delay, administrative etc.)	250
	Subtotal unavailable hours	**1250 hours**
Available hours	**Available hours**	**7270 hours**
	Equivalent to	965 shifts/year
Non-cycling hours	Potty brakes, traffic delays, possibly weather delays, waiting time, etc.	250 hours/year
Efficient Operating Hours	**Operating time**	7020 hours/year
Availability and Efficiency factors	**Availability Factor** $(\frac{Available\ hours}{Total\ hours})$	**85 %**
	Efficiency Factor $(\frac{Efficient\ operating\ hours}{Available\ hours})$	**96.5 %**
	Overall Efficiency Factor	82 %

Efficiency factor is calculated in order to determine the amount of time for which the equipment is used efficiently. When efficiency factor is determined, it will be considered in the subsequent calculation procedures while defining the number of equipment.

Chapter 3. Equipment Selection

The task of this chapter is to identify the feasible technologies for the mining and materials handling in Aynak Copper deposit considering the site conditions, geology of the deposit, and other relevant factors.

3.1. Possible Mining Technologies

Currently various technologies exist in the mining industry which are possible for the development of a mineral deposit, such as drill and blast, rippers, surface miners, dredges, scrapers, and other state of the art technologies. All of the above-mentioned technologies are discussed as following:

3.1.1. Drill and Blast

In prehistoric time, men were using different methods for fragmenting rock (rock penetration) like water quenching and fire making. Rock fragmentation with the help of explosive was applied in the 18th century when black powder was prepared. Easiness are provided in the area of rock fragmentation (material breakage) when Alfred Nobel created dynamite in 1867. After the creation of dynamite, the usage of ANFO (Ammonium Nitrate Fuel Oil) started in 1955. As time passed by, many other new technologies emerged in the field of explosives [22].

Drilling is a process of creating holes on the surface with a small diameter and different depths. Drilling is performed for different objectives, like production, exploration and technical aspects. In production drilling the drilled holes are filled with explosives to achieve good fragmentation results. In exploration drilling, the purpose is to obtain the cores for geological analysis and resource/reserve estimation. In technical drilling, the purpose is to get information about the slope stability, drainage, foundation testing, and few other purposes [23].

Drilling and blasting are the most important parts of the production cycle during surface and underground operations. According to Hartman (1987), the production cycle of a mine constitutes four parts, namely drilling, blasting, loading, and hauling. The importance of drilling and blasting is shown in Figure 3.1-1, since it constitutes the half of the production cycle [16].

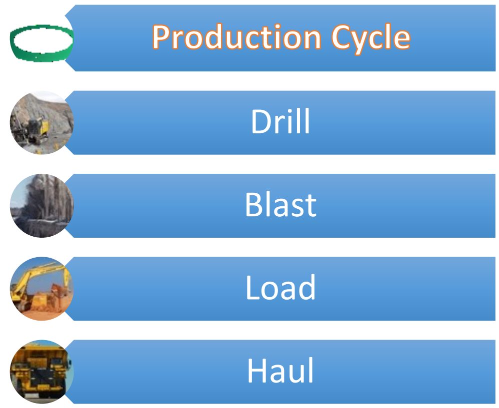


Figure 3.1-1 Basic production cycle [17]

3.1.1.1. Drilling Methods

Technologies' development is very rapid and the manufacturers developed the products and equipped the machinery with state of the art technologies in the current century. Applying autonomous system for dump trucks and camera system for detection of rock interface in surface miners can be mentioned as state of the art features in machineries.

There are two major drilling methods:

- Percussive Drilling
- Rotary Drilling

Another method is percussive-rotary drilling, which is a combination of the above-mentioned two methods. Figure 3.1-2 and Figure 3.1-3 show two types of drill rigs.

Figure 3.1-2 MD 6240 Rotary Drill [24]

Figure 3.1-3 Cat MD1550c Top hammer drill (percussive) [24]

In percussive drill rig, the rock breakage (penetration) is achieved through a combination of rotation and percussive, transmitted by the drill bit to the rock. This is used for the smaller diameter holes. For holes with bigger diameter, the hammer is located down-the-hole or in-the-hole closely above the bit in order to avoid losses of energy in the string or rods. Earlier, the percussive drills were powered by compressed air, but in mid-1970 the hydraulic drill replaced the pneumatic ones. The advantages of hydraulic drill rigs compared to the pneumatic drill rigs are the better penetration rate and less moving parts [25].

In rotary drilling, the torque is applied at the end of the rod in order to rotate the bit that is installed at the other end of the string. Bit rotation causes the breakage of material. Bit rotation and penetration is powered by diesel or electricity. Different bits are used in the rotary drilling and the most popular one is a tri-cone bit, suitable for the holes having diameter of 150 - 444 mm. There are three types of rotary drill bits:"Milled-tooth tri-cone bit, TC Tricone bit, and PDC insert drag bits" [25].

As the main purpose of drilling and blasting is to break or fragment the rock mass, so it is important to mention the characteristics of rocks affecting the drilling process. According to Jimeno et al. (1995), the physical properties of rock/rock mass like Hardness, strength, elasticity, plasticity, abrasiveness and texture have an effect on drilling process and also the selection of drilling method.

A detailed description of the above-mentioned rock/rock mass properties is beyond the scope of this thesis. For detailed information the reader is referred to "Drilling and Blasting of Rocks" by Jimeno et al. (1995) [22].

3.1.1.2. Types of Rock Drills (Drilling Rigs)

After discussing the drilling methods, now the question raises about the types of drill rigs. Mainly there are two different types of drill rig:

- Pneumatic drill rigs
- Hydraulic drill rigs

They differ according to the mean of energy transmission. In pneumatic drilling rigs, the compressed air is used as a mean of energy transfer while in hydraulic drilling rigs, hydraulic oil is used. Hydraulic rock drills came into existence in sixties and beginning of seventies, and developed very robust [22].

According to Jimeno et al. (1995), several reasons like lower cost and significant drilling capacity are behind the idea why hydraulic rock drills are better than pneumatic drilling rigs. More or less the same reasons like low energy consumption, lower costs, greater drilling capacity, better environmental conditions, and flexibility in operation were mentioned by Marshal (1985) [26].

Lots of efforts have been made by manufacturers in order to bring automation in drill rigs and to optimize the drilling process. Computer-assisted, computer–monitored and few other features can be name as part of drill automation process [27]. As result of these efforts, they developed a computer-controlled drilling system. These features can be seen in the drilling rigs which are used for surface mining that they are equipped with the global positioning system for controlling drill hole locations [28].

3.1.1.3. Blasting

When it comes to blasting, first, it is necessary to talk about the explosives, its types and their classification, its properties, and the initiation systems.

Explosives used for mining and construction purposes are classified as follows [28]:

- High explosives
- Blasting agents

High explosives are classified based on the high velocity of detonation (VOD), pressure, and density. Blasting agents are the mixture of fuel and oxidizer, and none of these elements is explosive individually. The initiation of blasting agent should be done by a primer detonator like ANFO [29].

Regulatory authorities in the United States classify explosives namely [30]:

- High explosives
- Low explosives
- Blasting agents

Regulatory authorities defined the blasting agents, as an explosive that cannot be detonated by a number 8 blasting cap. Number 8 blasting cap is defined as a cap that has 0.4-0.45g Pentaerythritol Tetra nitrate (PETN) base charge [30].

After understanding the different classifications, it is important to note the major properties of the explosives. According to Olofsson (1990), the properties of explosives are: “Velocity of Detonation (VOD), strength, detonation stability, sensitiveness, density, and many other properties.”

Generally, the firing is classified into three different types [29]:

- electric detonators
- none- electric detonators
- electronic detonator.

Cap and fuse detonators were used extensively in the blasting. With the invention of the electric detonators, the hazards from blasting decreased and blaster could start initiation from a protected area.

Non-electric detonators were not used as much as the electric ones. By inventing the shock tube in 1990, non-electric detonators become the mostly used detonators. Nowadays, non-electric detonators, especially shock tube detonators, are used all over the world and it is proved to be safe initiation system from staff harming point of view during blasting operations. As advantages of the nonelectric detonators, the following points can be mentioned [29]:

- easy handling
- no leakage while blasting
- Accuracy of blasting process

The third initiation system is electronic, which has a programmable chip. This chip manages the delay time of the initiation through sending some codes. The main difference between electric and electronic detonators is a micro chip. Pyrotechnic elements are replaced by the micro chip in electronic detonators. The problem with this type of initiation system is, that they are very expensive [29].

3.1.1.4. Pros and Cons of the Drill and Blast

Each technology has pros and cons and so does the drilling and blasting. Pros of drilling and blasting are [31]:

- Cost effective
- Applicable to any condition (rock properties)
- Any geology
- Flexibility

The cons of drilling and blasting are the following [31]:

- Ground vibration
- Air shock

- Fly rock
- Slow advance rate
- Cyclic process
- Transport, handle and storage

Besides the above-mentioned disadvantages, it can be stated that political and security situation in Afghanistan is critical, especially in the area where the Aynak copper deposit is located. There is a chance that the explosives could be seized by Taliban (an insurgent group which is in the opposition of Government).

3.1.1.5. The Drill Selection Process

Before selectin of drill for the blasthole drilling purposes, the properties of the material which have impact on the drillability rate should be taken into account. These are the factors stated by (Martin, 1982) [32].

- Hardness
- Compressive strength
- Abrasiveness
- Toughness

Generally, theses assessments are done on the site, but sometimes these are also done by taking some samples and analyzing it in the laboratory. Occasionally, manufacturers are able to conduct rock drillability tests for the appropriate selection of drill and bits. Analysis of samples in labs gives quite good insight about the drillability of materials.

According to Hustrulid et al. (2013) drill selection process is a three-step process [27]: The first step is the identification of drill size and its type. The following factors should be considered when deciding to purchase the drill rig: Type of material to be drilled, amount of material to be produced and transported, the capacity of loading, haulage and plant equipment, blasting requirements, and the overall costs. Aforementioned points have influence on selection of the size of the drill. In addition, the number of drills to be purchase should be considered.

Typically, it is accepted that for each loading shovel, a drill rig is required except in a situation where the relocation distance of machine is redundant and then an extra drill is required. The Second step is the determination of the potential supplier of drill rig. In the third step, factors get very specific and are mentioned by Martin, (1982) as follows [32]:

"Machine capabilities must exceed formation penetration requirements; Larger machines are more rugged and can drill in harder formations; A machine that can handle drill pipe long enough to permit single pass drilling can significantly improve production; dust control requirements are dictated by regulations and; long-term productivity is dependent on the ruggedness, reliability and maintainability of the design, and few other things."

Besides all of these criterions, following areas should be taken into consideration: mine life, topography of the area, moving distances of drill, skills of operating and maintenance personnel, and service support facilities. Finally yet importantly, the purpose of this assessment and evaluation in the selection process is to determine the proper and cheapest drilling system [32].

3.1.2. Surface Miner

The surface miners primarily were applied for the construction purposes like milling of asphalt. The idea that this technology can be used in the surface mining came into existence in the mid-70s with the development of a cost-effective and new open cast method, called surface mining technology. Wirtgen is the most famous firm that produces and exports the surface miners. The first machine that Wirtgen sold to the market in 1983 was 1900 SM Surface Miner [33].

Currently, about 300 surface miners exist in the world and 150 of these surface miners are working in coal mining industry in India. This machine is not only used in India but also in United States of America, Russia, Bosnia, and Australia. Surface miners can be used in a variety of rocks from soft to medium hard rocks (100-120 MPa) across the world [33]. Figure 3.1-4 shows the applicability of surface miner in different minerals/ores [34].

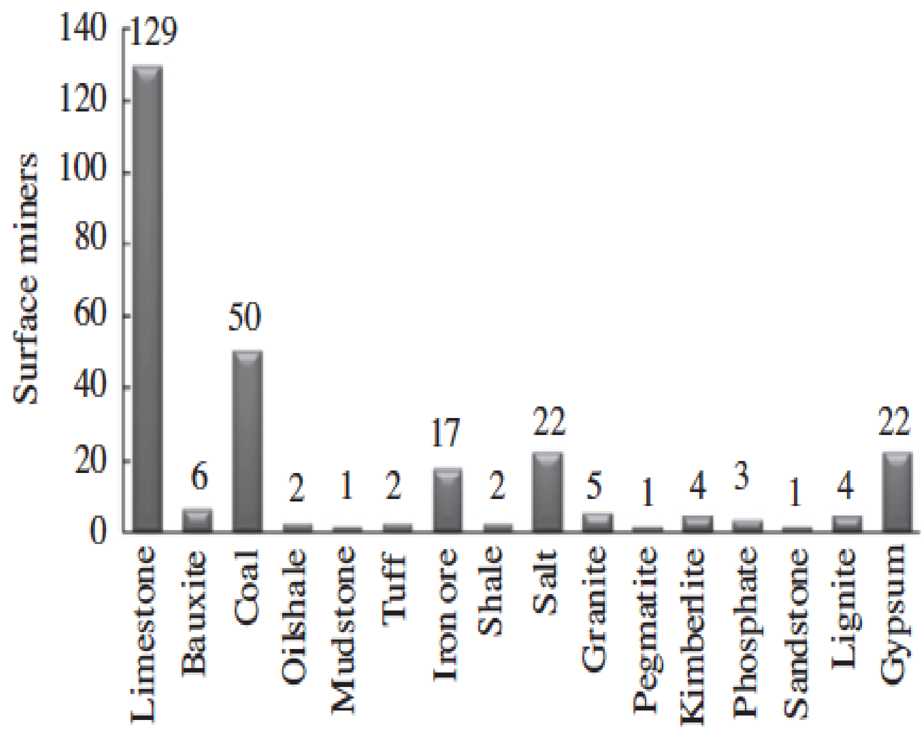


Figure 3.1-4 Application of surface miners in different minerals/ore mining [34]

Application of surface miner in different mineral by Hermann (2016) as follows [33]:"New Hope Group Australia – 4200 SM at New Acland Coal, FMG Australia – 4200 SM Iron ore mining, and The OCP Group in Ben Guerir - 2500 SM at Phosphate Mining."

Working principle of a surface miner is depicted in Figure 3.1-5.

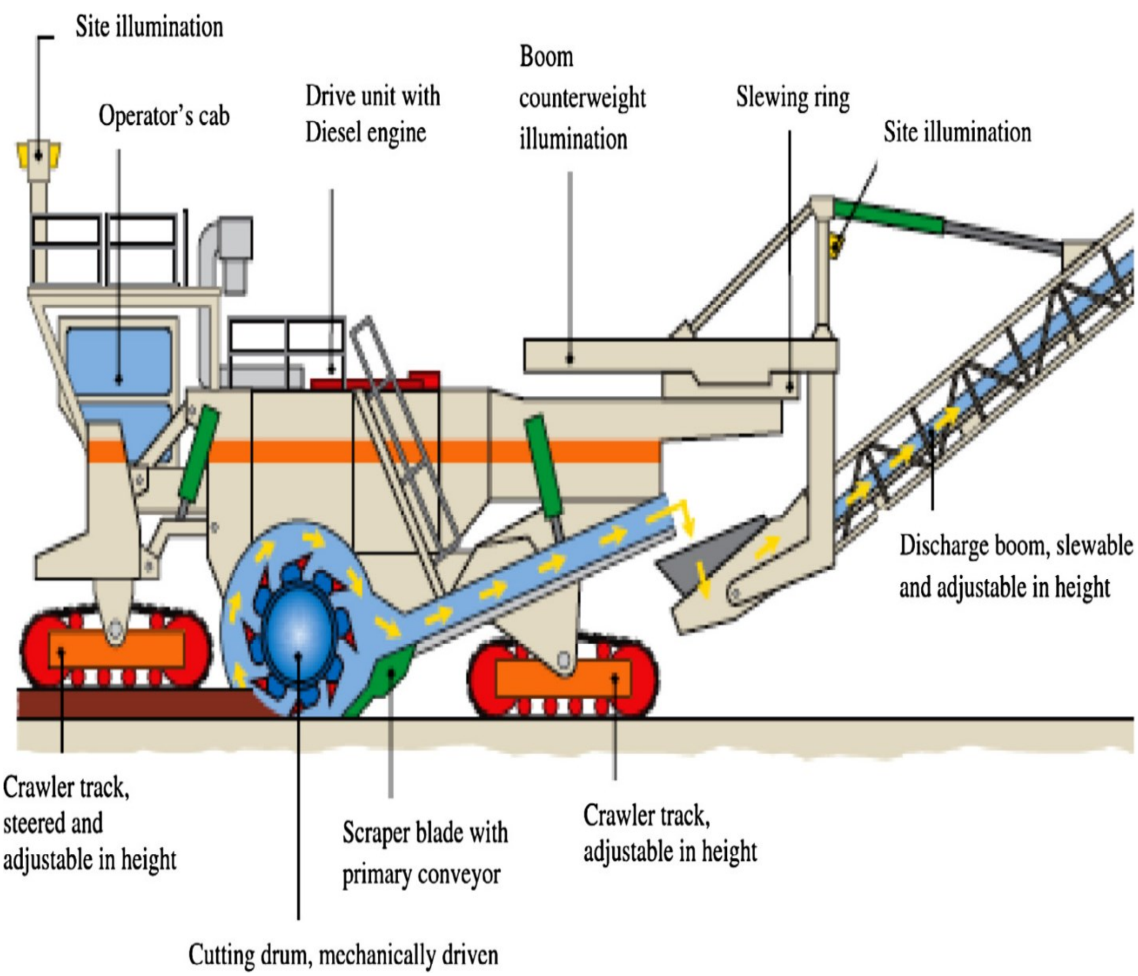


Figure 3.1-5 Principle of Working of Surface Miner [35]

3.1.2.1. <u>Type of Surface Miners</u>

The following three types of surface miners are available in the market [36].

- Machine with middle drum configuration
- Machine with front boom
- Machine with the front cutting wheel.

Among these three type, the machines with the middle drum are the widely used surface miners in the industry. It is applied in different parts of the world and various mineral deposits. Table 3.1-1 shows the technical data of surface miners.

Table 3.1-1 Technical data of SurfAce miners [36]

	Middle drum	Front cutting boom	Front cutting wheel
Cutting width [mm]	250 - 4200	5250	7100
Cutting depth/height [mm]	0 – 800	1000/5500	0 – 2900
Capacity	For all machines output is related to material characteristic		
Weight [t]	40 – 190	135	540
Installed power [kW]	450 – 1200	750	3340
Manufacturer	Wirtgen / Bitelli / Huron	Voestalpine	Krupp Fördertechnik

3.1.2.2. Parameters Influencing Cuttability of Surface Miner

The parameters which are influence the cuttability of Surface Miner are as follows: [37]

- Rock/rock mass Parameters
- Machine configuration, and
- Type of Application

According to Dey et al. (2008), rock parameters which affect the cuttability of Surface Miners are the following: ʻʻmoisture content, density, brittleness, unconfined compressive strength (UCS), point load index, Young's modulus, fracture energy, toughness index, and Brazilian tensile strength.ʼʼ

Dey (2008) has published the following configurations parameters of surface miner in his paper: "clearance angle and tip angle, pick lacing, type of pick (point attack), the number of picks, tip material, drum weight, engine power and nature of coolant for tips" [37].

3.1.2.3. Types of Loading of Surface Miner

Surface miners have three different methods for the loading of materials. First, one is a conveyor loading system. In this type of loading, excavated material, leads to a discharge conveyor via primary conveyor and is loaded directly to a dump truck [35]. The second one is windrowing, and in this type, surface miner excavates the material and a plate located behind the drum heaps the excavated material. Consequently, the cut material is loaded onto a dump truck by a different kind of loading equipment [35]. The third one is side casting, and in this method, surface miner excavates the material and directly dumps the cut material on the either side. Cut material is loaded by front-end-loader to a dump truck [35].

Each surface miner machine can work in all these three methods [33]. until Figure 3.1-8 illustrates different types of loading of surface miners.

Figure 3.1-6 Direct Loading of Surface Miner [38]

Figure 3.1-7 Windrowing of Surface Miner [39]

Figure 3.1-8 Side casting of Surface Miner [40]

3.1.2.4. Pros and Cons of Surface Miner

The most important advantage of surface miner is that this machine makes the process of mining simple by performing four different operations with one individual machine. Selective mining is the second advantage of surface miners that decreases dilution of ore and increases the yield of processing plant. High production rate is the third advantage of surfaces miner and has been achieved in New Acland Coal mine [33].

The only drawback is that the surface miners can be used mostly in the extraction of coal, limestone and other soft to the medium hard materials. Cutting ability of surface miners depend on the unconfined compressive strength of the rock which being cut. By increasing UCS of rock, the ability of surface miners decreases and effectiveness of machine will drop down dramatically.

3.1.3. Ripper

Surface mining is the vital and crucial part of the mineral production. Several methods of rock excavation exist, i.e. drill and blast, mechanical excavation and microwave irradiation, and each of these methods have their advantages and disadvantages. Amongst these methods, drill and blast have more disadvantages as compared to the other two methods. Of the biggest challenge using drill and blast method is the rules and regulations of governmental offices, such as environmental protection agencies, social protection agencies, and a few othes.

In order to avoid this problem, rippers can be used for the excavation of rocks and rock mass. Rippers with dozer can be a reasonable method of excavation for blast free and eco-friendly mining, and to reduce the use of explosives in surface mining [41].

Rippers are generally used for the removal or excavation of over burden and it is much cheaper than drill and blast for the soft rocks/rock masses.

Primarily, rippers were used by Romans for the making the Appian Way and it was mounted on wheels. Consequently, the usage of rippers for the construction of the railway was started between 1860-1880 [42].

After discussing briefly the rippers, it is important to describe the different types of rippers that are available.

- Types of ripper

According to Caterpillar Handbook of Ripping, types of rippers are described in the following: a) Hinge type b) Parallelogram type, and c) Adjustable parallelogram type [42].

a) Hinge-type Ripper

In this type of ripper, the linkage that is carrying the beam and shank pivots is fixed at the rear of the tractor. The shank enters into the rock (ground) and when it is reached the maximum depth, the tooth angel is changed constantly. It counts as drawback of hinge-type ripper [42].Figure 3.1-9 shows the Hinge-Type ripper.

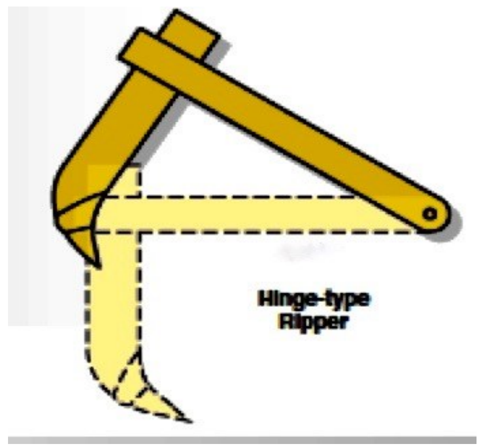


Figure 3.1-9 Hinge-type ripper [42]

b) Parallelogram-type Ripper

Linkage carrying beam and shank in parallelogram-type is allowed to maintain a constant tip-ground angle irrespective of tooth depth. The advantage of parallelogram ripper compared to hinge-type is that the parallelogram ripper can rip more than specified maximum depth. The disadvantage is that they do not have aggressive tooth angle which is necessary for the ripping of hard material [42]. Figure 3.1-10 depicts the Parallelogram-type ripper.

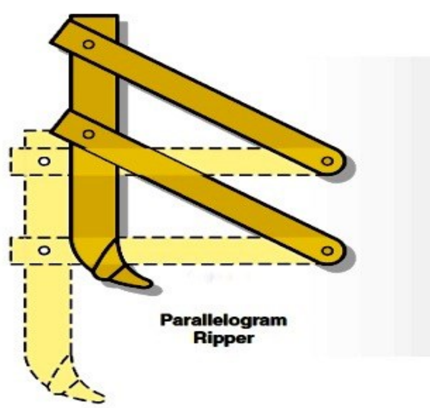


Figure 3.1-10 Parallelogram-type ripper [42]

c) Adjustable Parallelogram Ripper

This type of rippers has the properties of both hinge-type and parallelogram-type rippers. The tip angle can be changed and hydraulically adjusted in order to reach the optimum ripping angle in different materials. Parallelogram rippers are used widely in all caterpillar tractors either single shank or multi shank. Single shank rippers are used in hard rocks while multi shank are used for medium hard rocks [42]. Figure 3.1-11 depicts the adjustable parallelogram ripper.

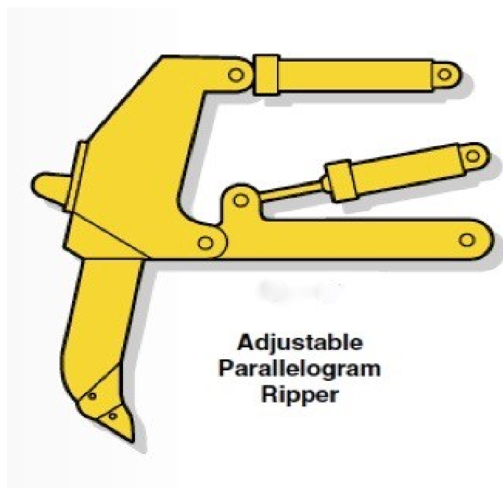


Figure 3.1-11 Adjustable Parallelogram ripper [42]

3.2. Technical Comparison of the Mining Technologies

Reviewing possible technologies for mining in the previous section, it was taught that the decision matrix should be used in order to determine the most appropriate technology for the mining of Aynak copper deposit. The process of decision matrix and the result is shown in Table 3.2-1.

Table 3.2-1 Decision Matrix for Mining Technology

Decision Matrix for Mining Technologies													
Criterion / Technologies	Different Geology of Deposits	Different strength of Rock (UCS)	Safety (transport, handling)	Flexibility	Environmental Impact	Continuity of Process	Different Mining Methods	Selective Mining	Productivity	Automation of Processes	No Requirement of Higher Force (SE)	Permit Application Procedure	Total Points
Drill and Blast	+	+	–	+	–	–	+	–	+	–	+	–	6
Surface Miner	–	–	+	–	+	+	–	+	+	+	–	+	7
Ripper	–	–	+	+	–	–	–	+	–	–	–	+	4
Descriptions: *(+) shows the advantage and effectiveness of the criterion to the listed technology and weighs 1 point.* *(–) shows the disadvantage of the criterion to the listed technology and weighs 0 points.*													

Considering the decision matrix for mining technology, the best technology is the use of Surface miner following by drill and blast and rippers. Different properties of rock formation should be studied in order to determine the fragmentation methods such as strength, hardness, geological strength index (GSI), discontinuities, and other parameters.

Applicability of different methods is being studied in the following:

- Drill and blast

In order to determine that whether drill and blast should be applied or not, the uniaxial compressive strength, hardness and other physical and mechanical properties of rock mass must be studied. UCS of major formations in Aynak Copper is given in Table 3.2-2.

Table 3.2-2 UCS of Aynak Major Cock Formation [10]

Type of Rock	Unconfined compressive strength (UCS) MPa
Marble	120 -150
Quartzite	80 -120
Dolomite	90 – 140

According to M. Chatziangelou et al. (2015), when the UCS of the rock formation is above 15 MPa and Geological Strength Index (GSI) is above 60, the drill and blast is the effective method of fragmentation. [43] The UCS of and GSI (GSI= 38-90) value of Aynak rock formation, favor the use of drill and blast method for fragmentation of rock formation [10].

- Surface Miner

Regarding applicability of Surface Miners for the fragmentation of Aynak deposit, it can be said that Surface Miners could be used but the only problem is the reduced productivity. As per guideline manuals of Wirtgen Gmbh, when the UCS of rock is above 80 MPa, the surface miner productivity decreases dramatically [44].

Applicability of surface miners faces difficulties in deeper part of the pit because down in the bottom of pit the space is confined and surface miner cannot turn, thus some places will be left and need other excavation methods.

- Assessment of Rippability of Rock

In order to determine the rippability of rock, it is important to know the geology of the rocks that constitute the mineral deposit. The physical characteristics of rocks which make the ripping process effective, are frequent planes of weakness, moisture, weathering, a high degree of stratification, brittleness, low strength, and low field seismic velocity. On the contrary, factors like a massive rock formation, lack of weakness planes, crystalline rock, lack of brittleness, high strength, and high field seismic velocity make ripping challenging and difficult [42].

There are different methods for the assessment of excavatability or rippability of rocks. In almost all of these methods, three important factors are taken into account, which are the uniaxial compressive strength, degree of weathering, and spacing of discontinuities. In some of other systems, the seismic velocity is also included [45].

Regarding the excavatability of rocks, Hoek and Karzulovic (2000) used the data which was studied by Abdullatif and Cruden (1983) in order to estimate the Geological Strength Index, and the result of their study is as follows [45]:

- Ripping can be used when the GSI Value of rock is 60 and rock mass strength value about 10 MPa.
- Blasting was the only effective excavation method for the rocks which have the GSI value greater than 60 and rock strength value is more than 15 MPa.

As per the evaluation of data which is used by Tsiambaos and Saroglou (2010) using the classification method of Franklin et al. (1971) and Pettifer and Fookes (1994), they reached the following conclusions [45]:

- Rock masses which have the joint spacing as 0.3-0.5 m and point load strength of intact rock more than 1 MPa should be excavated by blasting or hydraulic breaking.
- Rock masses which has the point load index less than 0.5 MPa can be easily fragmented by ripping or digging.
- When point load strength of intact rock is between 2-5 MPa (mean value 3MPa), it should be excavated by drill and blast.

Most researchers, Bell (2004), Gribble (1985) and Bieniawski (1975) suggest that point load strength index of 3 MPa is equal to UCS of 70 MPa [46] [47] [48]. Taking into consideration the UCS of the rock formations in Aynak (70-150) MPa and the Geological strength index (30 – 90), ripping will not be feasible [10].

The second method, which is used for the rippability assessment of Aynak Deposit rock, is rippability chart of Caterpillar that is illustrated in Figure 3.2-1. [42] The main factor that Caterpillar and Komatsu use in their guidelines for the rippability assessment of rocks is the seismic velocity of minerals.

Aynak Copper deposit comprise mostly from marble, quartzite, and dolomite [10]. Seismic velocity has not been done in Aynak deposit, therefore, the numbers of seismic velocities have been taken from different sources, and a decision was made. Seismic velocities for the marble, quartzite, and dolomite is given in Table 3.2-3 [49] [50].

Table 3.2-3 Seismic Velocities of materials

Material	Velocity(km/s)
Marble	3.75 – 6.94
Granite	4.5 – 6.5
Quartzite	5.0 – 7.0
Dolomite	3.5 – 6.9
Gneiss	3.5 – 7.5

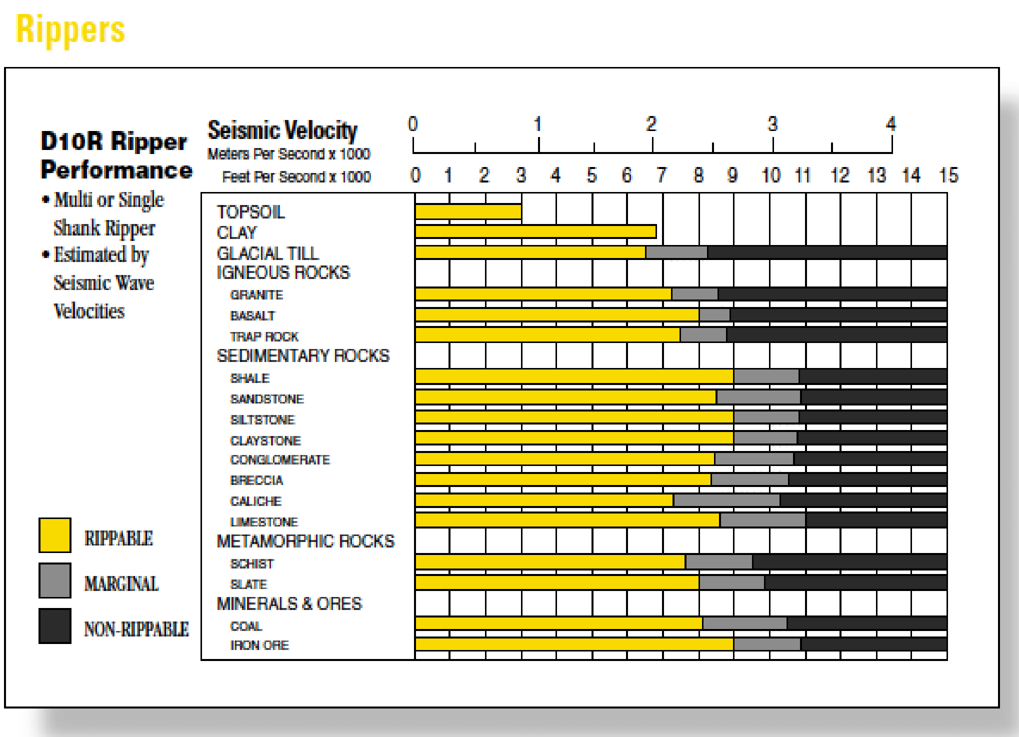


Figure 3.2-1 Rippability investIgation chart of caterpillar [42]

As it is clear from the above figure, seismic velocity above 3000 $^{m}/_{s}$ is located in the range of not rippable material and it is illustrated in Figure 3.2-1 with dark black color. Based on the given seismic velocities and considering the rippability investigation chart of Caterpillar, it shows that the rocks at Aynak copper deposit can be placed in the Not-Rippable zone, the reason for the non-feasibility are its UCS, GSI and Seismic Velocities [42].

In the previous section, different technologies were compared for mining of the deposit. Two of three technologies, which were compared, seemed feasible. The number of equipment were estimated according to planned annual production of the deposit

3.3. Calculation Procedures of Selected Mining Technologies

3.3.1. Drilling Rig

In order to determine the type of drill rig that is needed, the blast holes for the bench should be designed. Figure 3.3-1 illustrates the design of blast hole parameters.

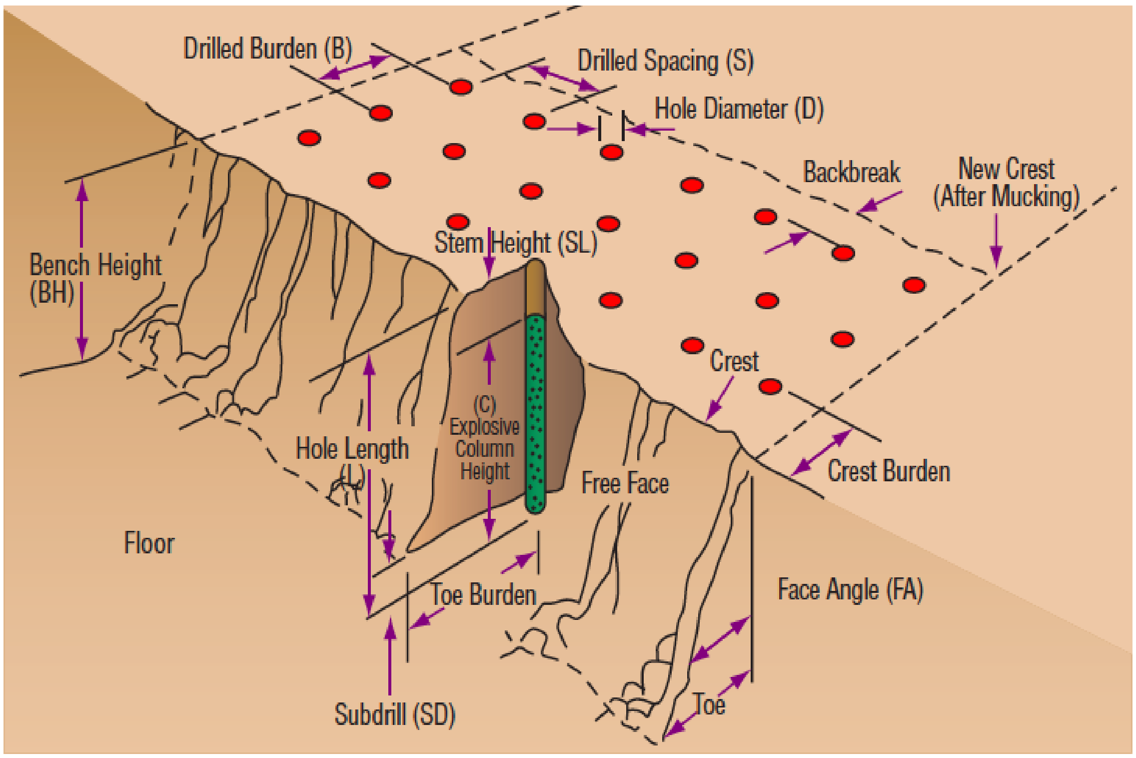


Figure 3.3-1 Blast hole parameters [51]

Drilling diameter is selected based on hourly production and compressive strength of the rocks, and it is taken from Table 3.3-1.

Table 3.3-1 Blasthole diameter based on average production and (UCS) of rock [22]

Blast hole Diameter D (mm)	Average production per hour bm³ (Bank cubic meter) /h	
	Medium-Soft rock <120 MPa	Hard-Very hard rock >120 MPa
65	190	60
89	250	110
150	550	270

The average planned production per hour is more than 250 bm^3 (bank cubic meter), therefore the bigger diameter (150mm) of blast hole is selected. 89 mm diameter was not selected because the average production per hour, in that case would be in the range of (110 – 250) bm^3.

Bench height is selected based on the blast hole diameter and the loading equipment, and it is taken from Table 3.3-2.

Table 3.3-2 Bench height based on blast hole diameter and loading equipment [22]

Bench height H (m)	Blast hole Diameter D (mm)	Recommended Loading Equipment
8-10	65-90	Front end loader
10-15	100-150	Hydraulic rope shovel

Design parameters like burden, spacing, stemming and sub drilling are selected considering the UCS of rock, and they are taken from Table 3.3-3.

Table 3.3-3 Geometric parameters according to UCS of rocks [22]

Design Parameter	Uniaxial Compressive Strength (MPa)			
	Low <70	Medium 70-120	High 120-180	Very High >180
Burden B	39 D*	37 D	35 D	33 D
Spacing S	51 D	47 D	43 D	38 D
Stemming T	35 D	34 D	32 D	30 D
Subdrilling J	10 D	11 D	12 D	12 D
(*) blasthole diameter (mm)				

Bottom charge length can be calculated based on Table 3.3-4.

Table 3.3-4 Bottom charge length based on UCS of rock [22]

Design Parameter	Compressive strength (MPa)			
	Low >70	Medium 70-120	High 120-180	Very High >180
Bottom Charge Length l_f	30 D	35 D	40 D	46 D

- Calculation Procedure

Uniaxial compressive strength: 70-150 MPa [10]

Blasthole diameter: 150 mm (taken from Table 3.3-1)

Bench height: 15 m (taken from Table 3.3-2)

The width of the round is determined based on an assumption to be: 50 m

Burden, spacing, stemming and sub drilling is calculated based on Table 3.3-3.

Burden B= 37D $\rightarrow$ B = 37 150mm $\rightarrow$ B=5,500 mm $\rightarrow$ B= 5.5 M*

*Spacing S= 47D $\rightarrow$ S= 47*150mm $\rightarrow$ S= 7000 mm $\rightarrow$ S= 7 M*

*Stemming T= 34D $\rightarrow$ T= 34*150 $\rightarrow$ T= 5000mm $\rightarrow$ T= 5 M*

*Subdrilling J= 11D $\rightarrow$ J= 11*150mm $\rightarrow$ J= 1500 mm $\rightarrow$ J= 1.5 M*

The length of the blast hole is calculated using Equation (3.3-1).

$$L = H + J\ [m] \qquad \textbf{\textit{(3.3-1)}}$$

Where,

H: bench height in (meter)
J: is sub drilling in (m)

$$L = 15 + 1.5 = 16.5\ meter$$

The volume of rock which will be broken by the blast hole is calculated using Equation(3.3-2).

$$VR = B \times S \times H[m^3] \qquad \textbf{\textit{(3.3-2)}}$$

$$VR = 5.5 \times 7 \times 15 \approx 578\ m^3$$

The length of the bottom charge can be calculated by the help of Table 3.3-4.

l_f = 35*D= 35*150 = 5.3 m

The concentration of bottom charge for ANFO is taken from Table 3.3-5.

Table 3.3-5 Charge concentration for different blasthole (D) and explosives [29]

Charge Concentration for different Diameters of Blasthole							
Blast hole diameter (mm)	51	64	76	89	102	127	152
Emulite (cut and dropped into dry blast hole) (kg/m)	2.3	3.7	5.0	7.1	9.3	-	-
Bulk emulate (kg/m)	2.4	3.9	5.3	7.5	9.9	15.3	21.9
ANFO (kg/m)	1.6	2.6	3.6	5.0	6.5	10.1	14.5

Concentration of bottom charge q_f = 14.5 kg/m

The ANFO is used in dry blast holes. In wet drill holes, the Heavy ANFO will be used that is the ANFO and bulk emulate. This mixture creates a water resistant explosive.

The Bottom charge is calculated using Equation (3.3-3).

$$Q_f = q_f + l_f \,;[kg] \qquad \textbf{\textit{(3.3-3)}}$$

$$Q_f = 5.3 \times 14.5 = 77\ [kg]$$

Length of the column charge is calculated as follows:

l_c=L-(l_f +T+J) = 16.5- (5.3+5+1.5) =16.5-11.8=4.7 M

According to Olofsson (1988) the concentration for column charge is between 40 -60 % of the charge concentration of bottom charge [29].

Concentration for column charge will be as following:

q_c= 0.4 – 0.6 Q_f = 0.6*14.5 = 8.7kg/m

Q_c= q_c * l_c = 8.7*4.7= 41 kg

The Amount of charges (explosives) which will be used in one blast hole is following is calculated using Equation (3.3-4).

$$Q_c = Q_f + Q_c\,; [kg] \tag{3.3-4}$$

$$Q_c = 77 + 41 = 118\ kg$$

Powder factor is calculated using Equation(3.3-5).

$$PF = \frac{Q_b}{VR}; \left[\frac{kg}{m^3}\right] \tag{3.3-5}$$

$$PF = \frac{Q_b}{VR} = \frac{118}{578} = 0.20\ \frac{kg}{m^3}$$

The yield of broken rock can be calculated using Equation(3.3-6).

$$RA = \frac{VR}{L}; \left[\frac{m^3}{m}\right] \tag{3.3-6}$$

$$RA = \frac{VR}{L} = \frac{578}{16.5} = 35\ \frac{m^3}{m}$$

W, Width of the round is adjusted accordingly:

$$\frac{W}{S} = \frac{50}{7} = 7.14 \approx 8$$

The number of the hole in a row is equal to the number of spaces + 1.

Specific drilling is calculated using Equation (3.3-7)

$$b = \frac{n \times L}{B \times H \times W}; \left[\frac{m}{m^3}\right] \tag{3.3-7}$$

b: specific drilling

n: number of boreholes

$$b = \frac{9 \times 16.5}{5.5 \times 15 \times 50} = \frac{148.5}{4125} = 0.036 m/m^3$$

Summary of the important data can be seen in Table 3.3-6.

Table 3.3-6 Summary of Calculated Data

Bench Height M	Hole Depth m	Burden m	Hole Spacing M	Bottom Charge Kg	Column Charge		Specific Charge (Powder Factor) Kg/m³	Specific Drilling m/m³
					Kg	Kg/m		
15	16.5	5.5	7	77	41	8.7	0,20	0.036

Based on properties of rocks, drilling methods should be selected. Different drilling methods of drilling can be applied for the situation of Aynak Copper deposit has. Aynak copper Rock properties was indicated in section (3.2), (Table 3.2-2.)

According to Hartman (2002), there are different drilling methods that are applicable for different types of rock, as shown in Table 3.3-7 [14].

Table 3.3-7 Application of drilling methods to different type of rocks [52]

Drilling Method	Rock Type/ Drillability			
	1 Soft (shale, weathered limestone, coal)	2 Medium-Hard (Limestone, Weathered Sandstone)	3 Hard (Granite, Chert)	4 Very Hard (Taconite, Quartzite)
Hydraulic Jet	×	×		
Rotary, drag-bit	×	×		
Toray, roller-bit	×	×	×	
Rotary Percussion		×	×	
Percussion		×	×	×
Thermal jet piercing			×	×

Considering the above-mentioned situations, it is decided that the Rotary drilling rig for the drilling of blast holes is appropriate and possible choice.

Annual production of equipment is calculated with the help of the Figure 3.3-2 [53].

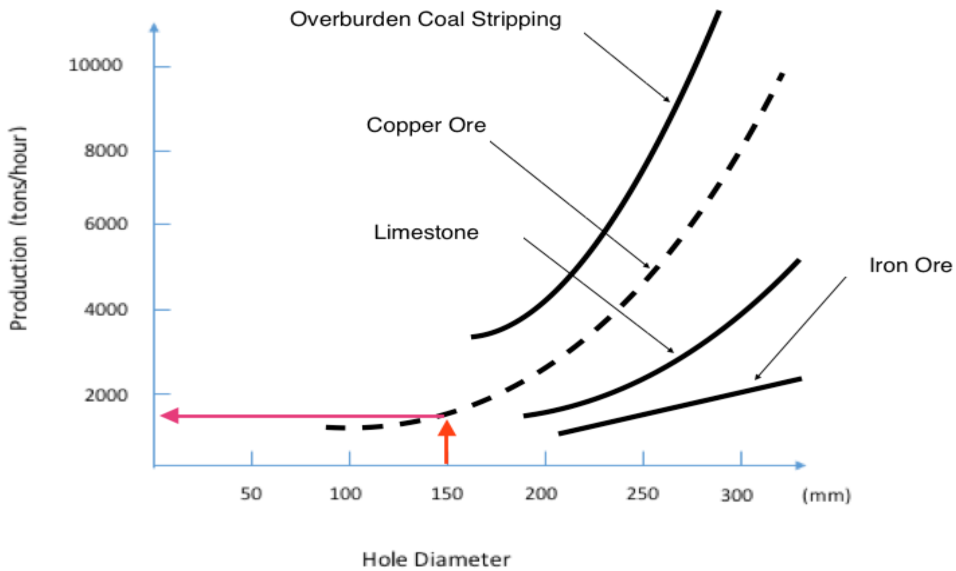


Figure 3.3-2 Tons Drilled/hr. in different material by drilling rigs [53]

Considering the hole diameter that was designed for the blast pattern, and the material which is being drilled, the hourly production of the machine is 1,500 t.

Annual production of machine = 1500*19.6*355=10.43*10^6 t=10.43* $10^6 \approx$ 10*10^6 t

By dividing the above amount to the annual production plan, the number of drills for the ore and waste is calculated accordingly:

$$\text{Number of drills for ore} = \frac{\text{Designed annual production of ore}}{\text{Machine annual production}}$$

$$Numbr\ of\ drills\ for\ ore = \frac{7 * 10^6}{10 * 10^6} = 0.70 \cong 1\ drill$$

$$Number\ of\ drills\ for\ waste = \frac{designed\ annula\ production\ of\ waste}{Machine\ annual\ production}$$

$$Number\ of\ drills\ for\ waste = \frac{52 * 10^6}{10 * 10^6} = 5.2 \cong 6\ drill$$

In previous chapter, different methods of drilling were discussed and an appropriate method selected based on the rock properties of Aynak deposit. Design parameters of blast hole have been calculated, subsequently the number of drills for ore and waste production was calculated.

3.3.2. Surface Miner and Trucks

According to Dey and Ghose (2011), production rate of a Surface miner can be calculated using Equation (3.3-8) [54].

$$L^* = \left(1 - \frac{CI}{100}\right) kM_c; \left[\frac{bcm}{h}\right] \tag{3.3-8}$$

Where,

L^*= production of cutting performance per hour (bcm/h)

M_c= Rated Capacity of the machine (bcm/h)

CI = Cuttability Index

K = a factor for taking into account the influence of specific cutting conditions and it is a function of pick lacing (array), pick shape, atmospheric condition etc., and it varies between (0.5-1.0).

Rated capacity for the 4200 SM is 3000 $t/_h$ and it is converted to cubic meter per hour accordingly [44].

$$M_{c4200SM} = \frac{3000\,\frac{t}{h}}{2.8\,\frac{t}{m^3}} = 1071\,\frac{m^3}{h}$$

The density of rock is taken from Table 3.3-9.

Table 3.3-8 Bank and Loose Density of Rock

Copper ore	
Bank density (t/m³)	Loose density (t/m³)
2.3 - 2.8	1.7

CI cuttability index is taken from Table 3.3-9 [54]:

Table 3.3-9 Assessment of excavatability of surface miners based on CI [54]

Cuttability index	**Possibility of cutting**
50 > CI	*Very easy excavation*
50 < CI <60	*Easy excavation*
60 < CI < 70	*Economic excavation*
70 < CI < 80	*Difficult Excavation, maybe not economic*
CI > 80	*Surface miner should not be used*

In this case, it is assumed that the excavation is economical.

CI = 70

K = 0.75

$$L^* = \left(1 - \frac{CI}{100}\right) kM_c = \left(1 - \frac{70}{100}\right) \times 0.75 \times 1071 = 240\ \frac{m^3}{h}$$

It is known that each machine has an efficiency factor and it also should be considered for surface miners. In this study the efficiency factor is calculated 82% meaning that in every 8 hours' shift, 6.5 hours the machine will be efficiently used.

Therefore, the annual production of a 4200 SM Surface miner is calculated accordingly:

$$\begin{aligned} &Annula\ Production\ of\ 4200\ SM \\ &\quad = Hourly\ production\ of\ SM\ \times\ Actual\ hours\ per\ day \\ &\quad \times\ days\ per\ year = 240\ \times\ 19.6\ \times\ 355 = 1{,}669{,}072\ m^3 \\ &\quad \approx 1{,}700{,}000\ m^3 \end{aligned}$$

The annual production equivalent in tons:

$$1{,}700{,}000\ m^3\ \times\ 2.8\ \frac{t}{m^3} = 4.76 * 10^6\ t$$

Consequently, the number of Surface miners for the production of ore and waste is calculated as follows:

$$Number\ of\ Surface\ Miners\ for\ ore = \frac{Planned\ annula\ production\ of\ ore}{Surface\ Miner\ annual\ production}$$
$$= \frac{7 * 10^6}{4.76 * 10^6} = 1.47 \cong 2\ Surface\ Miner$$

$$Number\ of\ Surface\ Miners\ for\ waste = \frac{Planned\ annula\ production\ of\ waste}{Surface\ Minr\ annual\ production}$$
$$= \frac{52 * 10^6}{4.76 * 10^6} = 10.9 \quad \approx 11\ Surface\ Miners$$

After determining the number of surface miner, the type and number of haulage trucks are determined respectively. The 90 ton 777D Cat haulage truck is selected based on the cycle time and loading capacity of Surface Miners SM 4200. The loading cycle time of Surface Miner is between 4-6 minutes, and here it is considered 6 minutes [40]. It is assumed that the distance to the waste dump from loading point is 5 km.

In order to determine the number of required trucks, the theoretical cycle time of truck, transporting the waste to the waste dump can be calculated using Equation (3.3-9) [55].

$$\mathrm{T_{ch}} = \mathrm{T_l} + \mathrm{TT_0} + \mathrm{T_{dp}} + \mathrm{TT_R}\ ;\ [min] \qquad \textbf{(3.3-9)}$$

Where,

T_{ch}= Theoretical cycle time (min)

T_l = Equipment load time (min)

TT_0 = Travel time to dump point" in case of waste" or Processing Plant "in case of Ore" (min)

T_{dp} = Dump or spread time (min)

TT_R = Return time (min)

The actual or corrected cycle time includes waiting and expected delays, and is calculated using Equation(3.3-10). Waiting and expected delays also takes into account bunching and queuing of trucks.

$$\mathrm{TC_{ch}} = T_{ch} + \mathrm{T_w} + \mathrm{T_d};\ [min] \qquad \textbf{(3.3-10)}$$

Where,

TC_{ch} = Corrected Haulage Cycle time (min)

T_w = Wait Time (min)

T_d = Delay Time (min)

Therefore,

T_l = 6 min

For the calculation of TT_0 and TT_R a hypothetical haul road segment was created (Figure 3.3-3) and calculation process is performed in the following:

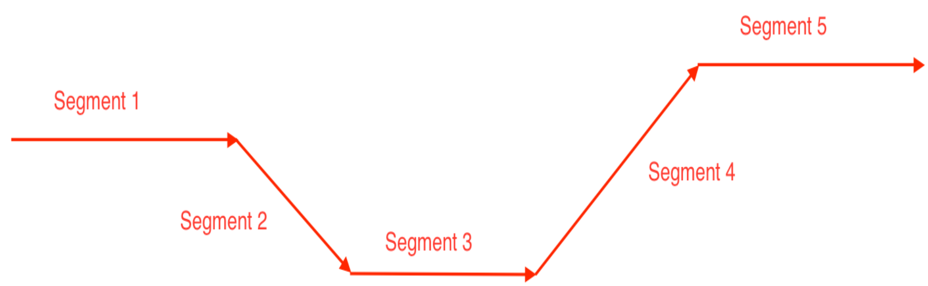


Figure 3.3-3 Idealized Segments of Road to Waste Dump

Characteristics of Haul road to waste dump are given in Table 3.3-10.

Table 3.3-10 Characteristics of haul road to the waste dump

	Segment Number	Length (meter)	Grade Resistance (%)	Rolling Resistance (%)taken from (reference SME)	Total Resistance (%)
Haul	1	500	0	3	3
	2	900	-6	3	-3
	3	1200	0	5	5
	4	1200	10	2	14
	5	1200	0	7	7
Return	5	1200	0	7	7
	4	1200	-10	2	-8
	3	1200	0	5	5
	2	900	6	3	9
	1	500	0	3	3

The rolling resistance has been taken from SME Mining Reference Handbook and grade resistance has been assumed [55].

E — Empty 64 359 kg (141,889 lb)

L — Max GMW 163 360 kg (360,143 lb)

Knowing the weight of the vehicle and total resistance, the maximum speed that a vehicle can achieve can be derived directly from the Performance Curve (Figure 3.3-4) and Retardation curves (Figure 3.3-5), accordingly:

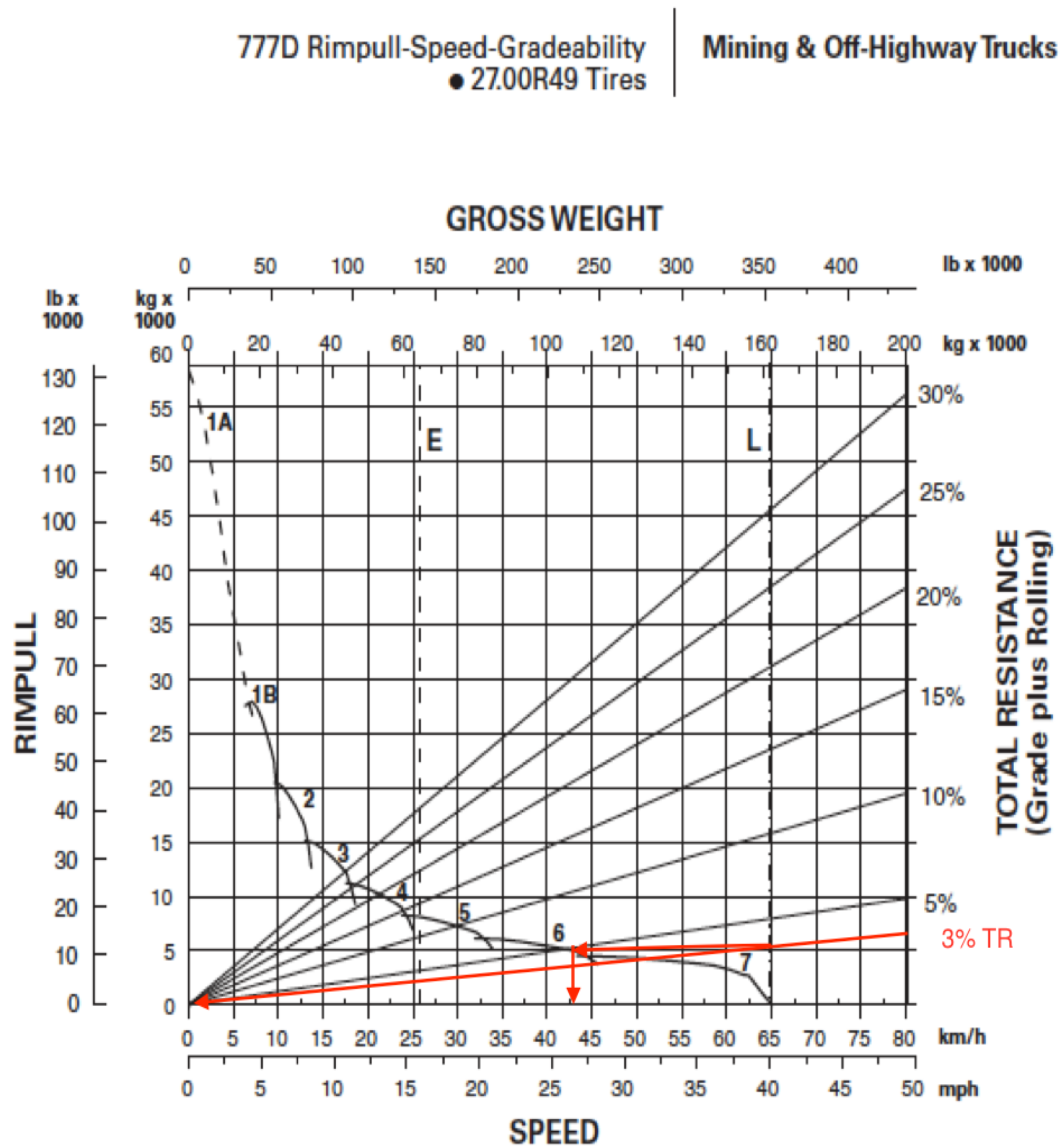


Figure 3.3-4 Performance Chart of Caterpillar

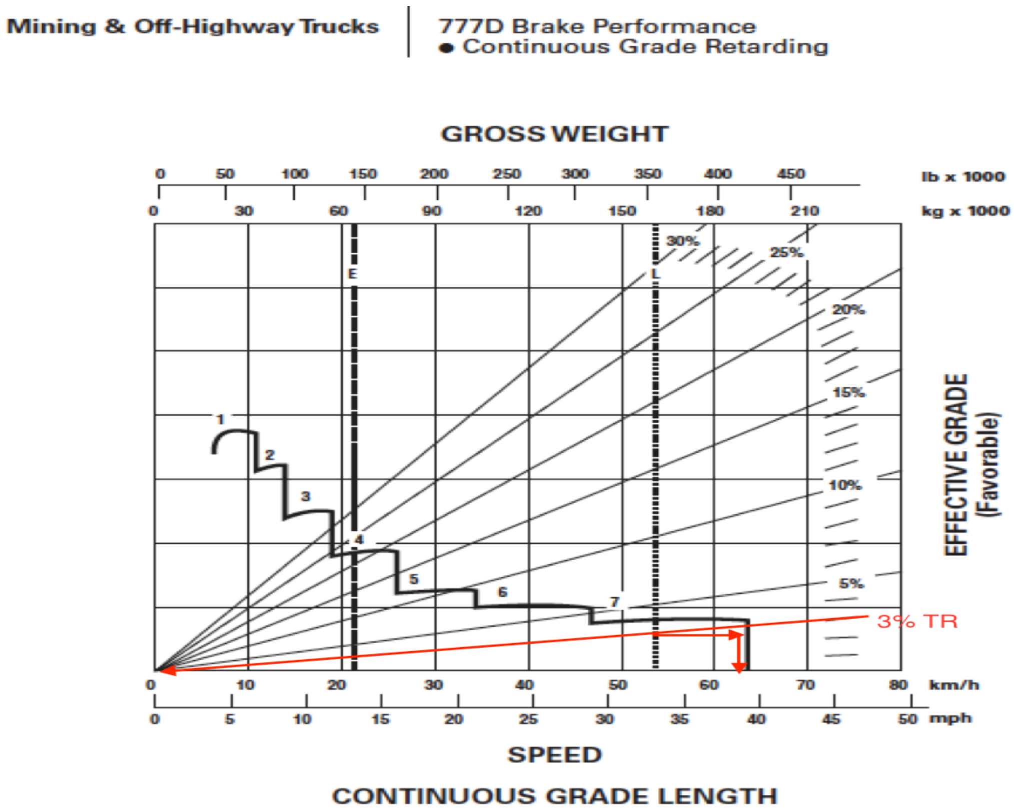


Figure 3.3-5 Retarding Curve of Caterpillar

The same procedure was repeated for all segments of the roads. The maximum speed the Trucks can achieve in each segment is given in Table 3.3-11.

Table 3.3-11 Speeds Taken from Performance and Retardation Curves

Segment	**Speeds from Performance(p)/Retardation(R) Curves (km/h)**
Haul	
1	43
2	63
3	30
4	12
5	16
Return	
5	45
4	63
3	58
2	40
1	63

In order to change the maximum speed derived from Performance/Retarding curves to average speed, the speed factor method is used. In this method, the maximum speed is multiplied by some speed factors given in Table 3.3-12 to convert it to an average speed.

Table 3.3-12 Speed Factors from Terex (1984)

Under 300 lbs/hp (136 kg/kW) Haul Road Length in Feet (meter)	**Level Haul Unit Starting from 0 MPH(km/h)**	**Unit in Motion When Entering Haul Road Section**		
		Level	**Downhill Grade**	**Uphill Grade Factor**
0-200 (0-60)	0-.40	0-.065	0-0.67	1 (Entrance Speed greater than maximum attainable speed on section)
201-400 (61-122)	0.40-0.51	0.65-0.70	0.67-0.72	
401-600 (123-182)	0.51-0.56	0.70-0.75	0.72-0.77	
601-1000 (183-305)	0.56-0.67	0.75-0.81	0.77-0.83	
1001-1500 (306- 457)	0.67-0.75	0.81-0.88	0.83-0.90	
1501-2000 (458 – 610)	0.75-0.80	0.88-0.91	0.90-0.93	
2001-2500 (611- 762)	0.80-0.84	0.91-0.93	0.93-0.95	
2501-3000 (763- 914)	0.84-0.87	0.93-0.95	0.95-0.97	
3501 and up (1067 & up)	0.87-0.94	0.95	0.97	
300-380 lbs/hp (136-172 kg/kW)				
0-200 (0-60)	0-0.39	0-.62	0-.064	1 (Entrance Speed greater than maximum attainable speed on section)
201-400 (61-122)	0.39-0.48	0.62-0.67	0.64-.068	
401-600 (123-182)	0.48-0.54	0.67-0.70	0.68-.074	
601-1000 (183-305)	0.54-0.61	0.70-0.75	0.74-0.83	
1001-1500 (306- 457)	0.61-0.68	0.75-0.79	0.83-0.88	
1501-2000 (458 – 610)	0.68-0.74	0.79-0.84	0.88-0.91	
2001-2500 (611- 762)	0.74-0.78	0.84-0.87	0.91-0.93	
2501-3000 (763- 914)	0.78-0.84	0.87-0.90	0.93-0.95	
3501 and up (1067 & up)	0.84-0.92	0.90-0.93	0.95-0.97	
380 & up lbs/hp (172 and up kg/kW)				
0-200 (0-60)	0-0.33	0-.55	0.56	1 (Entrance Speed greater than maximum attainable speed on section)
201-400 (61-122)	0.33-0.41	0.55-0.58	0.56-0.64	
401-600 (123-182)	0.41-0.46	0.58-0.65	0.64-0.70	
601-1000 (183-305)	0.46-0.53	0.65-0.75	0.70-0.78	
1001-1500 (306- 457)	0.53-0.59	0.5-0.77	0.78-0.84	
1501-2000 (458 – 610)	0.59-0.62	0.77-0.83	0.84-0.88	
2001-2500 (611- 762)	0.62-0.65	0.83-0.86	0.88-0.90	
2501-3000 (763- 914)	0.65-0.70	0.86-0.90	0.90-0.92	
3501 and up (1067 & up)	0.70-0.75	0.90-0.93	0.92-0.95	

In order to determine the loading condition of trucks, whether it is empty, loaded, or partially loaded, ratio of weight to power (WPR) can be determined using Equation (3.3-11).

$$\text{WPR} = \frac{\text{Vehicle Weight (lbs} - \text{kg)}}{\text{Flywheel (brake) horsepower} - \text{kW}} \qquad \textbf{\textit{(3.3-11)}}$$

By finding WPR, the part of the table to be used can be determined.

For loaded truck

Weight = 163 360 kg

Net Power = 699 kW

$$WPR = \frac{163,360\ \text{kg}}{699\ kW} = 233\ kg/kW$$

For empty truck

Weight = 64 359 kg

Net Power = 699 kW

$$WPR = \frac{64{,}359\ \text{kg}}{699\ kW} = 92\ kg/kW$$

By calculating this ratio, it can be determined which horizontal part of the table to be used. The second part is deciding which vertical part of the table should be used, and for this purpose the table is divided into two parts: *"Level Haul Unit Starting from 0 MPH"* and "*Unit in Motion When Entering Haul Road Section*".

The second column of table, **"***Level Haul Unit Starting from 0 MPH***"** is used for the vehicles which are starting from rest along leveled road (grade resistance=0). When a vehicle is in motion, *"Unit in Motion When Entering Haul Road Section***"** column of the table has been used. This column has been divided into three parts, level, downhill grade, and uphill grade.

level column which is existed in "Unit in Motion When Entering Haul Road Section" has been used for the vehicles which are already in motion and enter to a new segment of road (rolling resistance exist). Consequently, for the segments which have downhill or uphill grade, the "Downhill Grade" and "Uphill Grade" columns of the table is used.

After determining the average speed with the help of speed factors for different segments, the time which it takes the vehicle to travel the segment is calculated using Equation (3.3-12).

$$\mathrm{TT} = \frac{\mathrm{D}}{\mathrm{V_{ave}} \times \mathrm{UCF}};\ [min] \qquad \textbf{\textit{(3.3-12)}}$$

Where,

TT = Travel Time (min)

D = Distance (m)

V_{ave} = Average Velocity (km/h or mi/h)

UCF = unit conversion factor = 16.7 (m-h)/(km-min) or 88 (ft-h)/(mi-min)

Travel time for segment 1 is calculated as follows:

$$TT = \frac{500}{22 \times 16.7} = 1.36\, min$$

Same procedure is used for each individual segments, and the calculated times are given in Table 3.3-13.

Table 3.3-13 Calculated Travel and Return Time

Segment	Segment Length (m)	Performance/ Retardation Curve Speed (km/h)	Speed Factor	Average Speed (Km/h)	Segment Time (min)
Haul					
1	500	43	0.51	22	1.36
2	900	63	0.85	54	0.99
3	1200	30	0.86	26	2.76
4	1200	12	1.0	12	5.98
5	1200	18	0.92	17	4.22
Total (*TT_0*)					15.31
Return					
1	1200	45	0.88	40	1.79
2	1200	63	0.97	61	1.17
3	1200	58	0.87	50	1.43
4	900	40	1.0	40	1.34
5	500	63	0.81	51	0.58
Total (*TT_R*)					6.31

TT_o = 15.33 min

$TT_R = 6.31\ min$

$T_{dp} = 1\ min$ (Taken from SME Reference Handbook)

$T_w = 0.6\ min$ (Taken from SME Reference Handbook)

$T_d = 0.5\ min$

Therefore, actual or corrected cycle time can be figured out as follows.

$T_{ch} = T_l + TT_0 + T_{dp} + TT_R + T_w + T_d = 6 + 15.33 + 1 + 6.31 + 0.6 + 0.5 = 29.7 \approx 30\ min$

Number of tucks for each Surface Miner which is producing waste, is calculated by Equation (3.3-13) :

$$N_h = T_{ch}/T_l \qquad \textbf{(3.3-13)}$$

where,

N_h = *Number of Tucks*

T_{ch} = *Theoretical truck cycle time (min)*

T_l = *Truck load time (min)*

Therefore, the number of tucks can be calculated as follows:

$$N_h = \frac{T_{ch}}{T_l} = \frac{30}{6} = 5 = 5 + 1 = 6\ trucks$$

It is obvious from the calculations, that 5 trucks are needed for each surface miner. One more truck should be added to the number of trucks because Surface Miner never stops working, thus 6 trucks will be needed for each Surface Miner.

The same procedure is performed for the calculation of the number of trucks for ore transportation to the processing plant, the distance from pit to the processing plant is assumed to be 3.6 km.

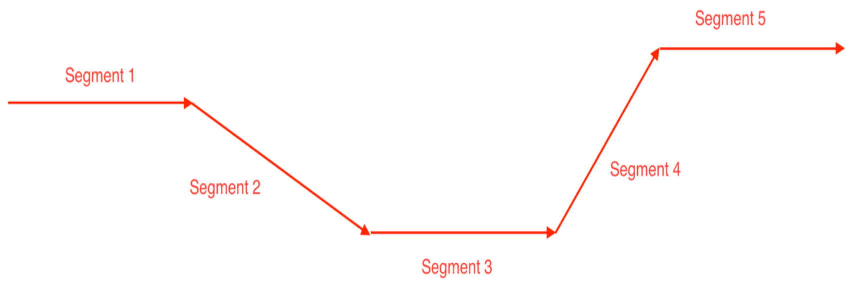


Figure 3.3-6 Idealized Segments of Road to Processing Plant

Characteristic of the haul road to the processing plant is given in Table 3.3-14.

Table 3.3-14 Characteristics of haul road to the processing plant

	Segment Number	Length (meter)	Grade Resistance (%)	Rolling Resistance (%)	Total Resistance (%)
Haul	1	300	0	3	3
	2	500	-6	3	-3
	3	1000	0	5	5
	4	2000	10	2	14
	5	1200	0	7	7
Return	5	1200	0	7	7
	4	2000	-10	2	-8
	3	1000	0	5	5
	2	500	6	3	9
	1	300	0	3	3

The cycle time for the trucks transporting ore to the processing plant is calculated as in below.

$T_l = 6$ *min*

TT_o and TT_R are calculated the same as it was calculated in previous section and it is given in Table 3.3-15.

Table 3.3-15 Calculated and Return Time

Segment	Segment Length (meter)	Performance/ Retardation Curve Speed (km/h)	Speed Factor	Average Speed (Km/h)	Segment Time (min)
Haul					
1	300	43	0.51	22	0.81
2	500	63	0.85	54	0.55
3	800	30	0.86	26	1.84
4	1000	10	1.0	12	4.99
5	1000	18	0.92	17	3.55
Total (*TT_0*)					11.74
Return					
1	1000	45	0.88	40	1.49
2	1000	63	0.97	61	0.98
3	800	58	0.87	50	0.95
4	500	40	1.0	40	0.74
5	300	63	0.81	51	0.35
Total (*TT_R*)					4.48

TT_o = 11.74 min

TT_{R} = 4.48 min

T_{dp} = 1 min (Taken from SME Reference Handbook)

T_w = 0.6 min (Taken from SME Reference Handbook)

T_d = 0.5 min

$$T_{ch} = T_l + TT_0 + T_{dp} + TT_R + T_w + T_d = 6 + 11.74 + 1 + 4.48 + 0.5 + 0.6$$
$$= 24.32\ min$$

The number of trucks for each surface miner which is transporting ore to the processing plant is calculated as follows:

$$N_h = \frac{T_{ch}}{T_l} = \frac{24.32}{6} = 4.05\ truks + 1 = 5\ trucks$$

Finally, the number of Surface miners and the trucks for the excavation and materials handling are listed in Table 3.3-16.

Table 3.3-16 Overall number of Surface Miner and Dump trucks

Name of Equipment	Total Number of Equipment
Surface Miner SM 4200	13
Cat 777D Dump Truck	76

3.4. Possible Loading and Transport Technologies

3.4.1. Truck and Shovel

Generally, there are two types of haulage trucks or haulers.

On-Highway Trucks and off-highway trucks: On-highway trucks are used for the haulage of the aggregates in road construction and buildings, in some cases, these are also used to haul the coal, ore, overburden, and topsoil [56].

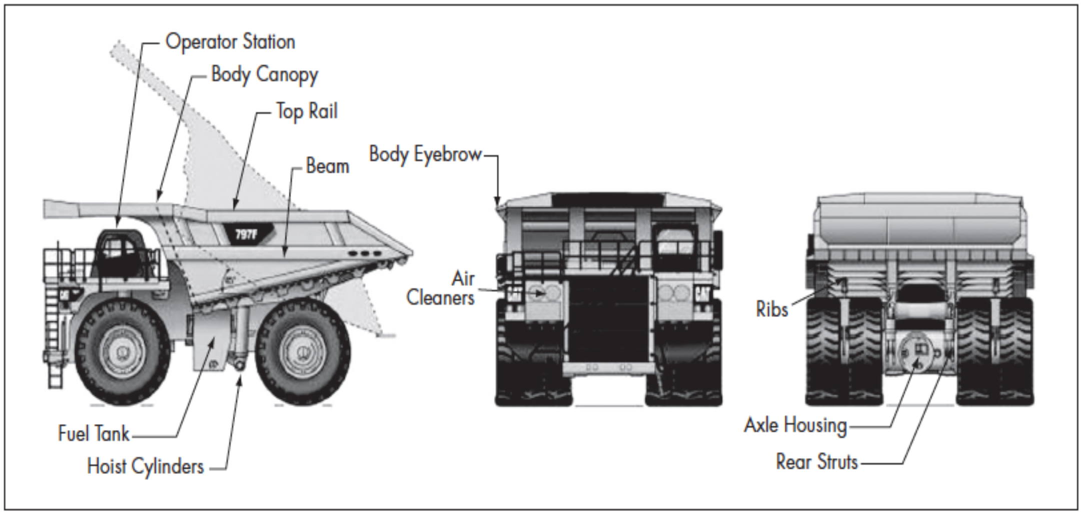
As the capacity of the on-highway truck is not very high (6-12 m^3), this type of truck is not discussed here further [56].

- Off-Highway Trucks

There are two type of Off-Highway truck: Off-Highway articulated Trucks and Off-Highway Rigid-Frame Trucks.

The first type of articulated dump truck is usually used for purposes of transporting aggregates for preparation of ground. When it is combined with a hydraulic excavator or loader, they are very effective for excavation of big masses [56].

Rigid-Frame Dump Trucks (Figure 3.4-1) have been used in the mining industry since the 1950s and 1960s. The trend was to replace a locomotive and small capacity dump truck to a better haulage system. This kind of dump truck is the cost effective and appropriate option for surface mining [56].



Courtesy of Caterpillar, Inc.

Figure 3.4-1 Off-highway rigid-frame truck [56]

The most important and determining fact about the trucks is their capacity because the capacity of the trucks has effect on mining operations. The trend in the industry was to make larger trucks in order to achieve high productivity and lesser operating costs. Different payload capacities existed for the surface mining trucks ranging from 90 -360 t [56].

- 90 class
- 135 class
- 180 class
- 220 class
- > 290 class

3.4.1.1. <u>Description of Shovel:</u>

There are three types of loading equipment like front end loaders, hydraulic shovels, and mining shovels [56]. Among these, each individual loading equipment has its own advantages and disadvantages. For loading of the material in this Project it was decided to use shovels.

Shovels are either Electric (power shovels) or Hydraulic. As known from the name, an electrical shovel uses the electrical power for the movement of the its bucket to the top and bottom, but in hydraulic shovels it is the hydraulics that cause the movement of bucket. Both types of shovels have a different range of bucket capacity starting from 20 to 100 m^3, for both electric and hydraulic. P&H Mining Equipment Inc. and Caterpillar are the two companies which manufacture electrical shovels. Caterpillar global mining, Hitachi construction machinery and some other companies manufacture the hydraulic shovels [57].

The electric shovel is mostly used in the coal mining for the removal of overburden. In some cases it is also used for the loading of other minerals like bauxite, gypsum and phosphate [58].

In electric shovels, there are a lot of mechanical components which make the process of loading a bit more complex. It requires electrical networking in the pit, substations, and cables for the distribution of electricity [56].

All of these characteristics which are listed above require special handling methods. Hydraulic shovels make these complexities less and simplify the process. It is worth mentioning that the hydraulic shovel has some advantages compared to other loading equipment mentioned below.

Compared to front-end loaders, they have low operating costs and greater digging force. With the above - mentioned criterion of hydraulic shovels, it was decided to use the hydraulic shovels (Figure 3.4-2) for the loading of haulage dump trucks [59].

Figure 3.4-2 Hydraulic Mining shovel [60]

3.4.2. In pit Crushing and Conveying

Materials handling has a vital role in mining operations whether surface mining or underground mining, when estimating both capital and operating costs. As stated previously, up to 50%of overall mining costs related to material handling costs [61].

The mining industry, especially companies which are working on surface mines, show their interest to in pit crushing and conveying system, because of fuel price fluctuation and rising operating costs of haulage dump trucks. Other factors which caused the interest of mining companies to the IPCC, was the increasing depth of open pit mines and a decrease of ore grade.

This challenge is compensated with the application of in pit crushing and conveying systems which were created previously in 1956 in Honver, Germany [61].

In pit crushing and conveying systems are divided into three systems based on their mobility:

- Mobile
- Semi-mobile
- Fixed

The mobile crushers are fed by shovel or excavators directly from the face and the mobile crasher must follow the digging shovel or excavator. Semi-mobile crushers are relocatable and depending on the deepening period of mine, these relocated in periods of 2 to 3 years. Fixed or stationary crushers are the crushers which are installed near to processing plants and it is designed for the entire life of mine.

In pit crushing and conveying system is the most reasonable option for materials handling when the open pit is extremely large and deep. Applying IPCC, the following two criterions should be achieved:

- Having the physical ability to excavate and transport material to some form of out of pit system at requested capacity. [1]
- Should be acceptable and cost effective in the capital and operating costs phases of the project. [1]

The drawback with IPCC systems, is high investment costs and possibility of damaging the conveyors by the blasting operations which will be done in the pit.

3.5. Technical Comparison of Loading and Transport Technologies

The same procedure was followed for the Mining technology is repeated here again for Loading and transport technology in order to determine the most appropriate technology for the transportation of fragmented material to a specific destination (was dump or Processing plant). The decision matrix is shown in Table 3.5-1.

Table 3.5-1 Decision Matrix for Loading and Transport Technologies

Decision Matrix for Loading and Transport Technologies											
Criterion / Technologies	Operation reliability	Flexibility	Capital Cost	No Need for Material Sizing	Environmental Impact	Adaptability to existing operation	No Equipment interface	No Influence on Mine planning	Maintenance not causing System Shut down	Higher Operating Costs	Total Points
Shovel and Truck	+	+	+	+	–	+	+	+	+	–	8
IPCC (in pit crushing and Conveying System)	–	–	–	–	+	–	–	–	–	+	2
Description: *(+) shows the positive impact of the criterion to the listed technology and weighs 1 point.* *(–) shows the negative impact of the criterion to the listed technology and weighs 0 points.*											

Considering decision matrix table, it can be seen that shovel –Truck technology outweighs IPCC system. Thus, detailed calculations were performed for the shovel-truck technology which is recommended for loading and transport of the material.

A detailed comparison performed between these two technologies considering different aspects of loading and transport technology and it is described hereinafter.

The first criterion to compare the hauling and loading technology is the flexibility of the technologies. Shove and truck system is more flexible than transporting bulk material via conveyors. Flexibility makes the truck haulage system a widely used technology in surface mining. It does not have any impact on mine planning, while using conveyors strictly affects the mine planning, because it requires long and straight pits. The second criterion is the production rate of the mine, shovel and truck system is used in operations with low to medium production rates, but in pit crushing and conveying (IPCC) is used when the production rate of the mine is too high. The third criterion is depth of the open pit. Shovel and truck system is used in shallow or moderate depths, but IPCC is used in open pits which are very deep and have a low grade and long distances [62].

IPCC systems confines the use of blasting in open pits, because if the blasting is performed near conveyors it may damage the foundations on which the conveyors stand. The large capital cost of IPCC can be compensated if the production is more than 100,000 t/d, haulage distance exceeds 5 km, and the system will be in operation for at least 7-8 years [63]. The use of IPCC system could be an option when the depth of the pit increases. This issue requires a detail study that will be covered in a detailed feasibility study of the deposit.

Based on the above mentioned reasons, the conventional operating system of shovel-truck seems suitable for the loading and transportation of materials from pit to the stockpile (ore) or the dump site (waste).

3.6. Calculation Procedure of the Selected Loading and Transport Technology

In this section Truck and Shovel system is discussed. Capacity of shovel bucket and a matching dump truck, and the total number of the shovel and dump trucks for the transportation of ore and waste was determined.

3.6.1. Shovel and Truck

The selection procedure of loading shovel is taken from Hartman et al. (2002). The first step is the determination of idealized output. The rate of shovels is determined generally in yd^3/hr. or $m^{3/}$hr. In shovel selection process the bank (solid) units are used.

Bank density of rock is taken from MoMP Reports and it is = 2.3 t/m^3

Required Production: 6,600 ton/shift

$$Q = \frac{6{,}600\frac{t}{shift}}{2.3\frac{t}{m^3}\times 6.5\frac{h}{shift}} = \frac{6{,}600}{15} = 440\,\frac{m^3}{h}$$

It is imperative that, the size of shovel should be estimated with the help of Table 3.6-1 in order to get the value of theoretical cycle time (t_c) of loading shovel for the calculation process. Both of these numbers are taken from Table 3.6-1. The cycles per hours is calculated using Equation (3.6-1) [64].

Table 3.6-1 Loading Shovel cycle time [17]

Loading Shovel Cycle Times (sec)					
B_c		**Digging conditions**			
yd^3	m^3	Easy	Medium	Medium-Hard	Hard
15	11.5	26	30	33	38

$$C = \frac{3600}{t_c} \tag{3.6-1}$$

t_c = Loading Shovel cycle time (sec)

$$C = \frac{3600}{t_c} = \frac{3600}{33} = 109\ cycles/hour$$

The Second step is the determination of operating factors. In order to modify the production rate which is calculated using ideal data with real-world situations, following operating factors should be considered:

- working time
- swing angle, and
- working and condition

Working time: the real-time for which an operator of the shovel works in a shift is affected by the operator change out system, number of shovel moves, meal breaks, human limitations, and other issues. It means that shovel will not be productive for 8 hours in a shift instead may be for 7 hours per shift. Therefore, the production factor (P) which is a unitless factor, is defined by excavation experts, and is calculated using Equation (3.6-2).

$$P = \frac{Actual\ min/hr}{60\ min/h} \tag{3.6-2}$$

Actual minute 50 min/hr. and is taken from Table 2.5-3.

$$P = \frac{50}{60} = 0.83$$

Swing Angle: Generally, the geometry of bench is selected as the swing angle of shovel becomes 90°. If the swing angle exceeds 90°, then according to Atkinson (1992a), following correction factors are used. Swing factor is given in below. [17].

Angle of Swing Degree	45	60	75	90	120	150	180
Swing Factor	1.2	1.1	1.05	1.00	0.91	0.84	0.87

Swing angle is assumed to be 120° for Aynak deposit, thus the value of S would be 0.91.

Working Condition: the bench condition, excavated material and degree of fragmentation are related to working condition. Based on Hartman et al. (2002), "The fill factor which is a portion of the bucket volume actually utilized during normal operation" is taken from Table 3.6-2 based on the properties of the material [17].

Table 3.6-2 Fillability of different materials [17]

Material	Factor
Rock-soil mixtures	1.1
Easy digging material (sand, small gravel)	1.00
Medium digging material	0.90
Medium hard digging material (iron ore, phosphate, copper ore, hard limestone)	0.85
Hard digging material (blocky iron, sandstone, basalt, heavy clay)	0.80

In the case of Aynak, the mineral is copper, thus the value of F= would be 0.85.

The dipper factor is also needed for the calculation of the size of the shovel, and it is the ratio of solid material to the material loaded in the bucket. Dipper factor B_f is calculated using Equation (3.6-3).

$$\mathrm{B_f} = \frac{\mathrm{F} \times (\text{weight } \div \text{loose unit volume})}{\text{weight } \div \text{bank unit volume}} \tag{3.6-3}$$

$$B_f = \frac{0.85\ (1.7)}{(2.5)} = 0.57$$

The densities are taken from Table 3.3-8 [10].

Different methods exist to calculate the size of mining equipment. Atkinson (1992) and Hartman (2002) [17] use different formulas to determine bucket capacity. Hartman' formula is explained by Equation (3.6-4) [64].

$$\boldsymbol{B_c = \frac{Q}{\left[C \times P \times S \times B_f\right]}} \tag{3.6-4}$$

where,

B_c = dipper (bucket capacity) in (m^3)

Q= production required in bank (m^3/h)

C= Shovel cycles time per hour

P= production factor

S = swing factor

B_f = dipper factor

The calculated variables are listed below for the calculation of the size of the shovel are listed below:

Q = 440 m^3/hr.

C = 109 cycles/hr.

P = 0.83

S = 0.91

B_f = 0.57

$$B_c = \frac{Q}{C \times P \times S \times B_f} = \frac{440}{109 \times 0.83 \times 0.91 \times 0.57} = 9.38 \cong 10\ m^3$$

The calculated value (10 m^3) of the bucket capacity of the shovel is close to the value of the shovel that was assumed (11.5 m^3). Thus, bucket capacity of shove (11.5 m^3) Is used in calculation procedures hereinafter.

The tonnage moved in each bucket should be calculated to determine the capacity of truck. When dealing with haulage equipment. It important to consider the loose density of material. The calculation as follows:

$$\frac{Weight}{bucket} = F \times \frac{Volum}{bucket} \times \ densityof\ loose\ material = 0.85 \times \frac{11.5\ m^3}{bucket} \times 1.7 = 16.67 \approx 17\ t$$

Generally, three to seven passes are needed for loading of a truck, in this case seven passes have been selected.

$$Capacity\ of\ Truck = number\ of\ passes \times tonnage\ of\ each\ bucket$$

$$Cpacity\ of\ Truck = 7 \times 17 = 119\ t$$

Keeping in mind the calculated capacity, the 777G dump truck with 120-t capacity from caterpillar was selected [65].

As stated by performance handbook of Caterpillar the annual production of the selected shovel reaches to approximately 11 million metric tons. One hydraulic shovel will be needed for the loading of the annual planned ore production and 5 shovels for waste rocks.

The cycle of haulage truck is calculated using equations (3.3-9) and (3.3-10) used also for the calculation of the haulage truck for surface miners in section (3.3).

$$T_{ch} = T_l + TT_0 + T_{dp} + TT_R + ;[min]$$

$$TC_{ch} = T_{ch} + T_w + T_d;\ [min]$$

T_{ch} = Theoretical haulage cycle time (min)

T_l = Equipment load time (min)

$$T_l = 7\ Shovels' swing \times 33 \frac{sec}{swing} = 3.85\ min$$

TT_0 = Travel time to processing plant (min)

TT_R = Return time to loading area (min)

Both TT_0 and TT_R are calculated following the same procedure as in section (3.3) for calculating the haulage cycle time for Surface Miners. Idealized Segments of the haul road to the processing plant is illustrated in Figure 3.3-6.

The maximum speed read from the Performance and Retardation Curves is changed to the average speed by multiplying it with Speed factors given in Table 3.3-12 and are given in Table A-1 and Table A-2 in Appendix section.

TT_0 = 7.79

$TT_R = 3.66$

T_{dp} = *Dump or spread time (min)*

T_{dp} = *1 min*

Therefore,

T_l = *3.85 min*

TT_0 = *7.79 min*

T_{dp} = *1 min*

TT_R = *3.66 min*

T_w = *0.6 min*

T_d = *0.5 min*

$$T = T_l + TT_0 + T_{dp} + TT_R + T_w + T_d = 3.85 + 7.79 + 1 + 3.66 + 0.6\ + 0.5 \\ = 17.4\ min$$

The number of tucks for the selected shovel is determined using the following equation (3.6-5).

$$T \leq n(T_l + T_s) \tag{3.6-5}$$

where, T_l is the loading time and calculated according to the number of swings which load the truck (3.85 min). T_s is spot time and is taken from SME Mining Reference Handbook between 0.3-0.6 minutes.

$$23.5 \leq n(3.85 + 0.5) \rightarrow 17.4 \leq 4.35\ n \rightarrow n = \frac{17.4}{4.35} = 4\ trucks$$

Referring to the calculations, 4 haulage dump trucks are required for the transportation of ore to the processing plant.

In an effort to determine, how many haulage dump trucks will be needed for the transportation of waste to the waste dump, the cycle time of Haulage dump trucks should be calculated using equation (3.3-9) and (3.3-10).

$$T_{ch} = T_l + TT_0 + T_{dp} + TT_R;\ [min]$$

$$TC_{ch} = T_{ch} + T_w + T_d;\ [min]$$

Therefore,

T_l = 3.85 min

TT_0 = 10.21 min

T_{dp} = 1 min

TT_R = 5.26 min

T_w = 0.6 min

T_d = 0.5 min

$$T = T_l + TT_0 + T_{dp} + TT_R + T_w + T_d = 3.85 + 10.21 + 1 + 5.26 + 0.6 + 0.5 = 21.42\ min$$

The number of tucks for the transportation of waste rock to the waste dump is calculated using the equation (3.6-5) in this way.

$$29.31 \leq n(3.85 + 0.5) \rightarrow 29.31 \leq 4.35n \rightarrow n = \frac{21.42}{4.35} = 4.9 \cong 5\ trucks$$

It should be pointed out, that two shot loader trucks are needed for the mixing and loading of emulsion explosive when there is water inside the balstholes.

The overall number of hydraulic shovels, haulage dump trucks and shot loader trucks needed for materials handling is listed in Table 3.6-3.

Table 3.6-3 Overall number of haulage dump trucks and hydraulic shovels

Name of Equipment	Total Number of Equipment
Cat 777G Rear Dump Truck (120 t)	29
Hydraulic Shovel (11.5 m^3)	6
Shot Loader Truck	2

3.7. Final Selection of Two Feasible Technologies

After conducting technical comparisons between the possible technologies the following two technologies are selected for the development of Aynak Copper deposit:

- Technology I

In Technology I, drill and blast is proposed to be used for the excavation (Fragmentation) of the rock and shovel and truck is proposed to be used for material handling.

- Technology II

In Technology II, the Surface miner is recommended to be used for the excavation, crushing and loading. Rear dump trucks should be used for the transportation of material to the destination points.

3.8. Cost Evaluation of Feasible Technologies

It is evident that equipment manufacturers do not publish their products cost publicly. In order to estimate the current price of mining equipment, usage of previous publication is one possible option. Different methods are used to escalate older costs and update it for estimation purposes in mine planning phase. The easiest and a handy method is using producer price index which is published monthly in their websites.

In this case, the costs of the mining machinery including drill rig, shovels, surface miners shot loader trucks and dump trucks are taken from the Mine and Mill Equipment Costs (2010) and are updated to the 2017 year by Producer Price index as follows [66]:

$$Cost\ in\ 2017 = Cost\ 2010 \times \frac{Mining\ Machinary\ and\ Manufacturing\ Price\ Index\ 2017}{Mining\ Machinary\ and\ Manufacturing\ Price\ Index\ 2010}$$

The mining machinery and the manufacturing price index for both years 2010 and 2017 years, are taken from Figure 3.8-1.

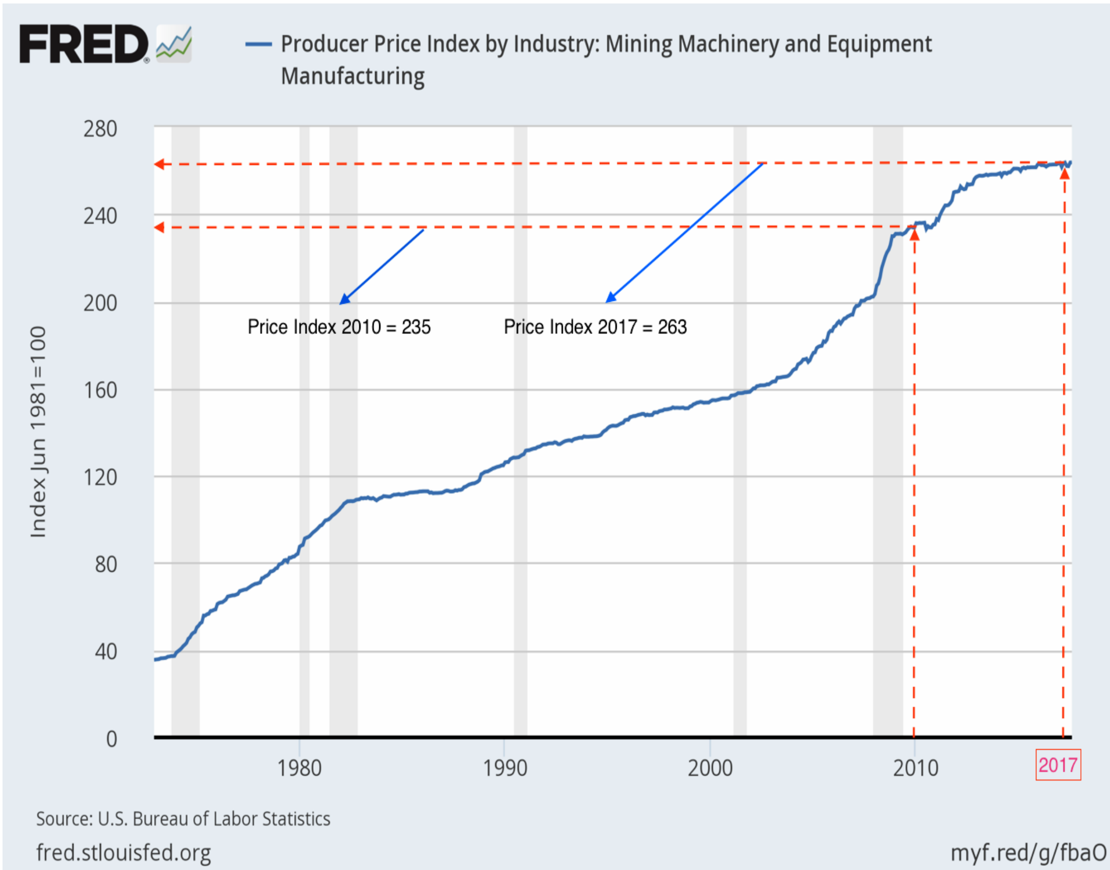


Figure 3.8-1 Producer Price Index for mining equipment manufacturing [67]

The equipment costs, taken from Mine and Mill Equipment Costs (2010) are given in Table 3.8-1.

Table 3.8-1 Costs taken from Mine and Mill Equipment Costs (2010)

Name of Equipment	Description	Unit	Quantity	Unit Price (USD)
Drill Rig	15.2-25.1 cm Hole, Diesel, (7.6 m) Rod Length	Set	1	1,010,000
Hydraulic Shovel	12 m^3 Bucket Capacity	Set	1	2,950,000
Cat Rear Dump Truck	Cat 777G Rear Dump Truck (120 t) Mechanical drive	Set	1	2,000,000
Cat Rear Dump Truck	777D (90.7 t) Mechanical Drive	Set	1	1,288,000
Shot Loader Truck	7.5 – 16.8 MT	Set	1	81,400
Surface Miner SM 4200	79.2 cm Deep × 4.2 m Wide Cut	Set	1	4,515,000

Based on the costs that are given in Table 3.8-1 the cost of a drill rig for this year is estimated as:

$$Cost\ of\ drill\ rig\ in\ 2017 = 1{,}010{,}000 \times \frac{263}{235} = 1{,}010{,}000 \times 1.11$$
$$= 1{,}121{,}000\ USD$$

It is apparent from the above calculation process that the factor (1.11) which is derived by dividing the price indexes of years 2017 and 2010 is the same and not going to be changed for other machineries. Therefore, this procedure is applied for all equipment in order to estimate the current costs of equipment that are needed for the development of Aynak Copper deposit.

The updated and estimated prices of equipment for Technology I are listed in Table 3.8-2.

Table 3.8-2 List of equipment in Technology 1

Name of Equipment	Description	Unit	Quantity	Unit Price (USD)	Total Price (USD)
Drill Rig	15.2-25.1 cm hole, Diesel, (7.6 m) Rod Length	Set	7	1,121,000	7,847,000
Hydraulic Shovel	12 m^3 Bucket Capacity	Set	6	3,274,000	19,644,000
Shot Loader Truck	7.5 – 16.8 MT	Set	2	81,400	180,000
Rear Dump Truck	Cat 777G Rear Dump Truck (120 t) Mechanical Drive	Set	29	2,220,000	64,380,000
Total Costs					92,051,000

The updated and estimated prices of equipment needed for Technology II are listed in Table 3.8-3.

Table 3.8-3 Equipment for Technology II

Name of Equipment	Description	Unit	Quantity	Unit Price (USD)	Total Price (USD)
Surface Miner SM 4200	79.2 cm Deep × 4.2 m Wide cut	Set	13	5,011,000	65,143,000
Cat Rear Dump Truck	777D (90.7 t) Mechanical Drive	Set	76	1,429,000	108,600,000
Total Costs					173,743,000

It can be seen from the costs table that the Technology I is cost effective and cheaper as compared to Technology II. Thus, the Technology I is recommended for the mining of Aynak Deposit.

Chapter 4. Financial Consideration for the Selected Technology

Comparing the investment costs of the selected technologies from Chapter 3 and distinguishing the economic technology was set as a main task in this chapter.

4.1. Investment Costs

The number and size of equipment have an influence on the capital and operating costs of open pit mines. Mining companies try to bring the investment costs into lower possible amount because in the mining industry the risks are big and there should be a financial backup in case some natural disaster happens. This eminent point is considered in this thesis. The investment cost for the selected technology is given in Table 4.1-1.

Table 4.1-1 Investment cost for selected technology

Name of Equipment	Description	Unit	Quantity	Unit Price (USD)	Total Price (USD)
Drill Rig	15.2-25.1 cm Hole, Diesel, (7.6 m) Rod Length	Set	7	1,121,000	7,847,000
Hydraulic Shovel	12 m^3 Bucket Capacity	Set	6	3,274,000	19,644,000
Shot Loader Truck	7.5 – 16.8 mt	Set	2	81,400	180,000
Cat 777G Rear Dump Truck	120 t Mechanical Drive	Set	29	2,220,000	64,380,000
Total Costs					92,051,000

4.2. Operating Cost Estimates

Operating costs mostly in pre-feasibility studies are estimated when the quantity of reserves are determined. Due to unavailable enough knowledge, costs cannot be estimated precisely. Because of that, the operating costs are estimated based on the calculated average costs of other mining ventures considering the site conditions of the current deposit [68].

In reference to O'HARA et al. (1992), the cost which are estimated in pre-feasibility studies are not accurate. There would be difference around $\pm$ 20 % compared to actual prices. The accurate and actual costs are calculated in detailed feasibility studies [27].

Prior to estimating of the costs, the mining conditions which affects the costs should be identified. Various factors have effect on estimation of capital and operating costs, but the essential factors are the size of mine and the processing plant. When the discovery of deposit is finished, the rate of mining and milling must be selected in the manner that, operating benefits increases the return on funds which are invested in development stage. Besides the above-mentioned factors, the production rate, personnel required, electrical power demand, and clearing of mine site also have effect on the costs [68].

The operating costs of an open pit mine are also affected by the size and number of equipment used during development of mine. The operating cost is expressed in tons of ore or waste mined per day. The cost per ton can be derived by dividing costs per day to tons mined per day. The cost of mining a ton of ore is assumed to be the same as that of waste in almost all of mining projects.

The operating cost of mine is calculated for different operations which will be performed during the day, using Equation (4.2-1) [27].

$$Drilling\ cost\ per\ day = \$\ 1.09\ \ T_p^{0.7} \tag{4.2-1}$$

$$Blasting\ cost\ per\ day = \ \$\ 3.17\ \ T_p^{0.7} \tag{4.2-2}$$

$$Loading\ cost\ per\ day = \ \$\ 2.67\ \ T_p^{0.7} \tag{4.2-3}$$

$$Haulage\ cost\ per\ day = \ \$\ 18.07\ \ T_p^{0.7} \tag{4.2-4}$$

$$General\ service\ cost\ per\ day = \ \$\ 6.65\ \ T_p^{0.7} \tag{4.2-5}$$

Where,

$T_p = Total\ ore\ and\ waste\ tonnage\ mined\ daily$

The total amount of ore and waste which can be mined daily is taken from section 2.5) and is given as $T_p = 166{,}300\ tons$.

In order to find the per ton operating cost, the calculated operating cost for one day is divided by the overall tons which will be mined the same day.

$$Operatingt\ cost\ per\ ton = \frac{daily\ operating\ cost}{T_p}$$

The calculation procedure as following:

$$Drilling\ cost\ per\ day = 1.09\ \times\ 166{,}300^{0.7} = 1.90\ \times\ 4{,}514 = 8{,}576\ ^{\$}/_{day}$$

$$Blasting\ cost\ per\ day = 3.17\ \times\ 166{,}300^{0.7} = 3.17\ \times\ 4{,}514 = 14{,}309\ ^{\$}/_{day}$$

$$Loading\ cost\ per\ day = 2.67\ \times\ 166{,}300^{0.7} = 2.67\ \times\ 4{,}514 = 12{,}052\ ^{\$}/_{day}$$

$$Haulage\ cost\ per\ day = 18.07\ \times\ 1166{,}300^{0.7} = 18.07\ \times\ 4{,}514 = 81{,}567\ ^{\$}/_{day}$$

$$General\ serive\ cost\ per\ day = 6.65\ \times\ 166{,}300^{0.7} = 6.65\ \times\ 4{,}514 = 30{,}081\ ^{\$}/_{day}$$

The operating cost per ton is calculated accordingly:

$$Drilling\ operatingt\ cost\ per\ ton = \frac{8{,}576}{166{,}300} = 0.051\ ^{\$}/_{ton}$$

$$Blasting\ operatingt\ cost\ per\ ton = \frac{14{,}309}{166{,}300} = 0.086\ ^{\$}/_{ton}$$

$$Loading\ operatingt\ cost\ per\ ton = \frac{12{,}052}{166{,}300} = 0.072\ ^{\$}/_{ton}$$

$$Haulafe\ operatingt\ cost\ per\ ton = \frac{81{,}567}{166{,}300} = 0.490\ ^{\$}/_{ton}$$

$$Haulafe\ operatingt\ cost\ per\ ton = \frac{30{,}081}{166{,}300} = 0.180\ ^{\$}/_{ton}$$

The calculated mining cost (ore and waste) are given in Table 4.2-1.

Table 4.2-1 Calculated operating costs per ton of ore and waste

Mining costs per ton of ore and waste		
No	Name of Activity	Unit ($/ton)
1	Drilling	0.051
2	Blasting	0.086
3	Loading	0.072
4	Hauling	0.490
5	General Service	0.180
	Total	**0.879**

4.3. Sensitivities

There is a need to analyze the existing sensitivities in every mining project in order to consider that in early stages of mine planning. Whereas, Aynak Copper deposit possesses susceptible geo-politic location., it has more vulnerability compared to other deposits. Aynak deposit is located in areas where insurgent forces have immense influence on that areas. The Aynak copper deposit vulnerabilities include security and landsliding. The big security issue which threatens the Aynak Copper deposit is the existence of an insurgent group termed "Taliban", which may hold control over considerable part of area, and it encompasses approximately 30% of local communities. Although the area of Aynak deposit is protected by fences, but sometimes Taliban fire RPG Rockets from far distances at the site and can harm the personnel and other staffs.

There is a possibility that all properties of mine like equipment, machinery, explosive magazines, ware houses etc. are seized by Taliban troops. Loss of these assets will add massively to the project financial costs. In case that explosive magazines, are seized by Taliban, it will equip the insurgents with a huge number of explosives against elected government. Though, there is a special force hired by Ministry of Interior Affairs (MoIA) of Afghanistan for the security of the area, the Taliban hold stronger than these government forces till yet.

The second issue is the probability of landsliding in region. Aynak area lacks vegetation and also there is no forest like other parts of the country. Because same accidents happened in Khahan district of Badakhshan province in 28th of April, 2015 [69]. The reason of landslide was reported heavy snowfall, although the area of Aynak copper is located in hot climate compared to Badakhshan province, but still there is a chance of happening the land slide in that area. Therefore, these sensitivities should be taken into consideration.

Conclusion

In the final analysis, it seems that there are two viable technologies for the development of Aynak Copper deposit. The firs reasonable technology is drill and blast, following by Surface Miners as the second feasible technology. The cost difference between the two technologies is about 40 %.

If the flexibility is considered in mine planning and mine development stages, the drill and blast outweighs surface miner. Numerous articles have been written and researches have been done about the application of blast free mining operations, but still drill and blast is the most effective and widely used technology in mining of medium, hard and extreme hard rocks. In application of drill and blast, the Taliban's gaining accessibility to explosives, should be taken into consideration.

If selective mining is the objective, focusing separation of different ore grades then, surface miner can be used. While applying Surface Miner, the deep part of pit, and bending radii of surface miner should be taken into account. Surface mining technology is the reasonable option in materials like coal, bauxite, gypsum, phosphate and while it's application in hard and extreme hard rock formations brings down the productivity of surface miner.

Development of Aynak copper deposit has considerable effects on employment, attraction of international mining companies, security stabilization in the area, living condition of the people, and overall economy of Afghanistan.

Further detailed study should be for the application of in pit crushing and conveying for the next mining phase.

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List of Abbreviations

AGS. Afghan Geological Survey

ANFO Ammonium Nitrate Fuel Oil

B. specific drilling [m/m^3];, burden [m];

B_c. shovel bucket capaciy [m^3]

B_f. dipper factor;

BGS. British Geological Survey

$bm^{3.}$. bank cubic meter

C. shovel cycle time per hour [min];

CI. cuttability index ;

D. distance [m];, blast hole diameter [mm];

E. empty weight [kg or lbs];

F. fill factor;

GoA. Government of Afghanistan

H. bench height [meter];

IPCC. in pit crushing and conveying system

J. subdrilling [m];

K. influence factor

L. loaded or maximum weight [kg or lbs];, *See* length of blasthole [m];

L^*.	production of surface miner per hour [bcm/h];
l_c.	length of column charge [m];
M_c.	rated capacity of the machine [bcm/h];
MCC.	Metallurgical Corporation of China
MoIA.	Ministry of Interior Affairs
MoMP.	Ministry of Mines and Petroleum of Afghanistan
MPa.	Mega Pascal ;
n.	number of boreholes;
N_h.	number of trucks;
P.	prouction factor;
PETN.	Pentaerythritol Tetranitrate
PF.	powder factor [kg/m^3]
Q.	production required [bcm];
Q_f.	amount of bottom charge [kg];
q_c.	amount of column charge [kg];, concentration of column charge [kg/m];
q_f.	concentration of bottom charge [kg/m];
RA.	yield of broken rock [m^3/m];
S.	swing factor;, spacing [m];
T.	stemming [m];
t_c.	loading shovel cycle time; [sec]

TC_{ch}.	corrected haulage cycle time [min];
T_{ch}.	theoretical cycle time [min];
T_d.	delay time [min];
T_{dp}.	dump or spread time [min];
T_l.	equipment load time [min];
T_s.	spotting time [min];
TT.	travel time [min];
TT_0.	travel time to dump [min];
TT_R.	return time [min];
T_w.	wait time [min];
UCF.	unit conversion factor
UCS.	Unconfined Compressive Strength
USGS.	United States Geological Survey
V_{ave}.	average velocity [km/h];
VOD.	Velocity of Detonation
VR.	volume broken by blasthole
W.	width of the round [m];
WPR.	weight to power ratio

Appendix A Characteristics of Haul Roads

Table A-1 Characteristics of Haul Road to the Processing Plant

	Segment Number	Length (meter)	Grade Resistance (%)	Rolling Resistance (%)	Total Resistance (%)
Haul	1	300	0	3	3
	2	500	-6	3	-3
	3	800	0	5	5
	4	1000	10	2	14
	5	1000	0	7	7
Return	5	1000	0	7	7
	4	1000	-10	2	-8
	3	800	0	5	5
	2	500	6	3	9
	1	300	0	3	3

Table A-2 Characteristics of Haul Road to the Waste Dump

	Segment Number	Length (meter)	Grade Resistance (%)	Rolling Resistance (%)taken from (reference SME)	Total Resistance (%)
Haul	1	500	0	3	3
	2	900	-6	3	-3
	3	1200	0	5	5
	4	1200	10	2	14
	5	1200	0	7	7
Return	5	1200	0	7	7
	4	1200	-10	2	-8
	3	1200	0	5	5
	2	900	6	3	9
	1	500	0	3	3

Appendix B Maximum Speed of Dump Truck

Table B-1 Speed Taken from Performance and Retardation Curve for Processing Plant Road

Segment	Speeds from Performance(p)/Retardation(R) Curves (km/h)
Haul	
1	67
2	68
3	49
4	15
5	37
Return	
5	68
4	67
3	65
2	50
1	66

Table B-2 Speed Taken from Performance and Retardation Curve for Waste Dump Road

Segment	Speeds from Performance(p)/Retardation(R) Curves (km/h
Haul	
1	67
2	68
3	49
4	15
5	37
Return	
5	68
4	67
3	65
2	50
1	66

Appendix C Calculated Average Speed of Dump Truck

Table C-1 Calculated Average Speed of Dump Trucks in Haul Road to the Processing Plant

Segment	Segment Length (meter)	Performance/ Retardation Curve Speed (km/h)	Speed Factor	Average Speed (km/h)	Segment Time (min)
Haul					
1	300	67	0.52	34	0.52
2	500	68	0.88	59	0.50
3	800	49	0.91	44	1.0
4	1000	15	1.0	15	3.99
5	1000	37	0.92	34	1.78
Total (TT_0)					7.79
Return					
1	1000	68	0.95	64	0.93
2	1000	67	0.97	63	0.95
3	800	65	0.87	56	0.85
4	500	50	1.0	50	0.59
5	300	66	0.80	52	0.34
Total (TT_R)					3.66

Table C-2 Calculated Average Speed of Dump Trucks in Haul Road to the Waste Dumps

Segment	Segment Length (meter)	Performance/ Retardation Curve Speed (km/h)	Speed Factor	Average Speed (Km/h)	Segment Time (min)
Haul					
1	500	67	0.60	40	0.74
2	900	68	0.90	61	0.88
3	1200	49	0.91	44	1.63
4	1200	15	1.0	15	4.79
5	1200	37	0.91	33	2.17
Total (TT_0)					10.21
Return					
1	1200	68	0.95	64	1.12
2	1200	67	0.97	65	1.10
3	1200	65	0.86	56	1.28
4	900	50	1.0	50	1.07
5	500	66	0.88	43	0.69
Total (TT_R)					5.26

Appendix D Idealized Segments of Road

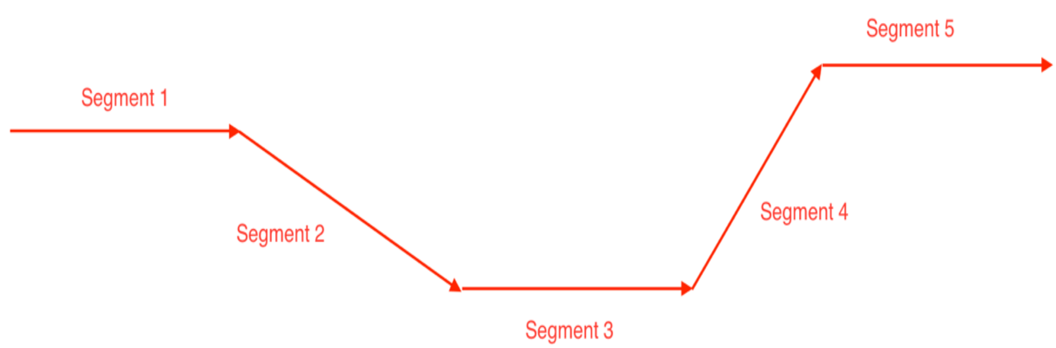

Figure D-1 Idealized Segments of Road to the Processing Plant

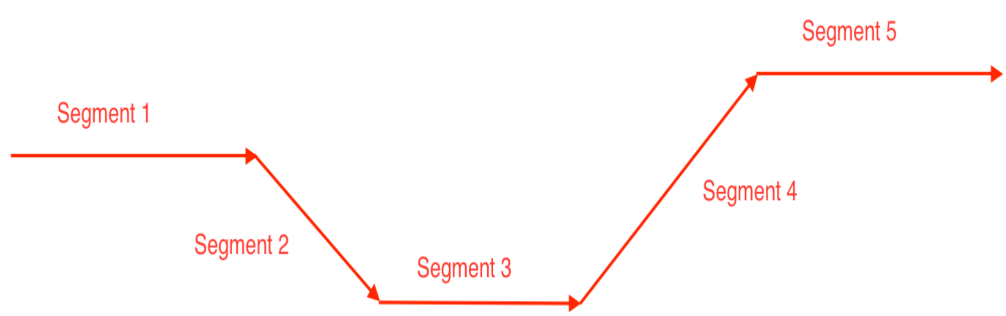

Figure D-2 Idealized Segments of Road to the Waste Dump