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MINING ENGINEERING DEPARTMENT

MINE 416

FINAL REPORT

D | D MINING CO.

EFLANI - KARABUK COAL MINE

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ABSTRACT

In this project, which was designed as the 4th section 1st group in the Mine416 course of the Middle East Technical University, the underground coal mine on the Karabük, Saraycık side was processed. Project details include probable and proven mineral reserves, underground block model constructions, underground method selection and coal production, shaft and transportation designs, ventilation designs, blasting designs and detailed rock mechanics designs. Apart from these, there is detailed information about the selection of equipment used in the design of the mine and the production of coal, and on what basis and how they are selected. Information about the extracted ore and the stages of industrialization that can be used are also given.

1. INTRODUCTION

D | D Mining CO. is a coal mining company which is located in Ankara. Depending on the geologic explorations, coal resource has been found in Saraycık, Karabük. The calorific values of coal are classified as high calorific values.

Coal is a non-renewable energy resource and combustible sedimentary rock. The coal has great significance in electricity production, metallurgical industry, and heating.

The project would be done in a high calorific value hard coal mine. The mining method will be selected with respect to the formation of the deposit, depth, dip of the deposit, material strength, and size of the deposit.

After the mining method is selected, the stability studies are handled. Based on the stability studies, the panel length, pillar width, and height, support of main roads will be decided. The project aims to extract the coal safely and profitably. After the stability studies are carried out, the development studies and production planning are done. Then, the economic analysis is studied to decide whether the project feasible or not.

2. GENERAL INFORMATION

This title includes information such as the general structure of the region, topography information, and climatic conditions.

2.1. Commodity

Coal is the commodity of the project. Coal is a combustible sedimentary rock. It is one of the fossil fuels over the world. It is defined as non-renewable energy resource because once the coal is used, it is not recyclable. The colour of the coal is black or brownish black. It is the deposit composed predominantly of carbon. It consists of more than 50 percent carbon. (U.S Energy Information Administration , 2021).

Coalification is the process of turning dead plants into coal. The earth has extensive forests in low-lying marsh zones at various times in the geologic past. Coalification began in these wetlands when dead plant matter was preserved from biodegradation and oxidation, generally by mud or acidic water, and turned into peat. The carbon was trapped in vast peat bogs, which were subsequently buried by sediments. The heat and pressure of deep burial caused the loss of water, methane, and carbon dioxide over millions of years, and the proportion of carbon rose. The grade of coal produced was determined by the maximum pressure and temperature reached, with lignite being produced under relatively mild conditions and sub-bituminous coal, bituminous coal, or anthracite being produced as the temperature and pressure increased. (Ibarra, Murioz, & Moliner, 1999)

There are different types of coals. The type of the coals is classified into 4 groups according to their carbon content and amount of heat energy that coal can produce.

One of the types of coal is anthracite. It contains between %86 and 97 carbons and generally has the highest heating energy value. The rank of the anthracite coal is the highest. It is a black, brittle, hard coal and generally referred as hard coal.

The other types of coal are bituminous. Bituminous coals are the middle rank coal between anthracite and subbituminous. Bituminous coal generally has high heating values. They are using for electricity generation and steel making. The form of the bituminous coals is blocky and appears shiny.

The third type of coal is subbituminous. Subbituminous coals are black and mostly dull not shiny. It has the low to moderate heating value therefore they are used in electricity generation.

The last type of coal is lignite. Lignite coal is brown coal which has the lowest ranking coal. It has the lowest heating value, carbon content and high moisture content. It is mainly used in electricity generation.

2.4. Climate

To some extent, Karabük has the qualities of the black sea climate. Although it is located in the sea region Karabük cannot benefit enough from the humid air of the Black Sea since it is inland from the coast, and the attributes of the continental climate are offset. While rain is seen in every season with the effect of the black sea climate, Karabük encounters a rainless period in July and August. In Karabük, which receives very little precipitation compared to the Black Sea coast, the annual average precipitation is 605.4 mm. Rainy days are much more in around Eflani, Ovacık, and Yenice regions. (Karabük İl ve Turizm Müdürlüğü, n.d.)

The mainstream is Filyos Stream and the highest elevation is Keltepe (2000 m.). Furthermore, other important streams are Araba, Soğanlı and Eskipazar Streams. Continental climate is seen in the high places, while the climate is moderately gentle in the valley floors. Karabük shows a basin character surrounded by high hills. (T.C. TARIM VE ORMAN BAKANLIĞI KARABÜK İL TARIM VE ORMAN MÜDÜRLÜĞÜ, n.d.)

Table 1. Average Values Measured by Karabük Meteorology Station (climate-data.org, 2022)

Average Values Measure by Karabük Meterology Station	
Annual Average Temperature	13.5°C
The Highest Temperature	42.2°C (19 August)
The Lowest Temperature	-5°C (26 January)
Annual Relative Humidity Average	70.10%
Annual Average Wind Speed, Direction	0.6 m/s, Northeast
Speed and Direction of the Strongest Wind	4m/s, Southwest (29 January)
Annual Precipitation Amount	605.4 mm
Number of Frost Days	44
Number of Rainy Days	136

According to the Table 1, extreme temperatures are not observed in Karabük but the effect of the continental climate can be seen with regard to the months. If open pit mining is considered, the working hour schedule should be arranged according to the temperature observed at certain times of the day. Even though 42.2 °C is not unsighted temperature in the whole cities in Turkey in summer months, this temperature will affect the workers, such as operators, foremen. In terms of occupational health and safety, employees should be given extra time for water and cooling needs during the hottest hours of the day. Furthermore, the machines may overheat, this may prevent the machines from working and the machine efficiency may decrease in monthly/weekly reports. The temperature of the drill holes can be also increased due to environmental conditions and this lead to not charging the holes with the explosives. As a result of this, production can delay, and the company can fall behind the weekly monthly production plan.

If underground mining is considered, this high temperature will affect the ventilation. As we get in deeper and deeper in the ground, the temperature increases gradually. The rise in the temperature due to the environmental conditions and the heat which releases from the working machines create a very challenging working environment for workers and machines. For workers' safety and health issues, the degree of temperature in mine should be monitored regularly for a specific period of time. The mine air should be cooled until it reaches to the acceptable temperature for human work. This means extra effort, extra time and extra money for the company but is compulsory for the safe working environment. As in open pit mine the overheat of the machines should be avoided.

The lowest temperature is observed in January. If the project is considered as open-pit mining, the temperature may not cause a problem in the operation of the machines Since the temperature at which diesel and gasoline starts to freeze is lower than -5 degrees. However, working hours should still be arranged for employees. Due to weather conditions, in January, if the mine is working in 3 shifts, this can be reduced to two. Obviously, the expected production rate should be lowered at this period of the year and production planning will be made accordingly. While the mining action continues in the field, the exploration drill holes can be done with geologists at the same time. These works can be stopped in January, but not in all the winter months, and permits can be given to people who are making exploration drilling operations. The frozen ground can create a problem for the transportation of materials and transportation of the workers who come from the nearby villages. The required explosives

for penetrate the ground at first stages of mining should be kept in a specific temperature range.

In underground mining, this temperature values which can be seen from the Table 1, if the ventilation way is natural ventilation, create less problem than open pit mining. Natural ventilation is strongest in the winter months. It helps the ventilation. If a lot of cracks or joints are existed in the upper surface of the ore or nearby the ore body, these cracks can be loaded with the rainwater and freezes in winter months. This leads to increase the volume of the rainwater and the measure of the cracks or joints can be increased. After the melting of the rainwater in the following days, these cracks may create stability problems. Also melted rainwater continues to go down and when rainwater encounters with the cavity in underground, it can be accumulated there. Some dewatering actions may have to be done to keep these waters out of the mine.

As mentioned before in this report, Karabük shows a basin character surrounded by high hills. Therefore, the wind will not have a great impact in the northeast direction on the mine we will open. Also, 4 m/s speed of the wind may create a problem for dust control in open pit mining. Regularly roads should be watered to avoid occurrence of dust. Otherwise, the dust which comes from the roads used by the mine trucks can go to the nearby villages. It can cause trouble for both the lungs of the people working in the mine and the lungs of the people in the nearby villages. The occurrence of dust in the blasting is unavoidable but with the proper stemming and less usage amount of explosive, its effects can be decreased.

However, in underground mining this speed of wind can be used in an advantage way for the mine ventilation. If the ventilation shaft is located according to the southwest wind, it helps the natural ventilation. On the other hand, the speed of the air should be arranged according to the Turkish Regulations. If natural ventilation is used with the effect of both the fan and this wind, some curtains can be used to reduce the speed of the air inside the mine.

With the information of the climate of Karabük 605,4 mm is expected (climate-data.org, 2022). In an open pit mine this may lead to the occurrence of muddy ground. For excavators and trucks, this muddy ground will create a problem. Pebbles can be spilled on the road. Also, these pebbles can hold the dust on the ground. Fort the production drill holes, accumulation of the rainwater causes a delay of the blasting of drill holes.

In underground mining, this precipitation amount, water filling the cracks may cause stabilization problems or dewatering processes may be required.

Flood disaster is a very common natural phenomenon in Karabük. Stream overflow and insufficient infrastructure can be shown as the cause of the flood disaster. On the other hand, drainage pipeline laying works for 28 neighbourhoods in Karabük continue. The project is 75% complete and 50% operationally active. Filyos stream is almost 180 km away from the mine site. So Filyos stream may not be a problem.

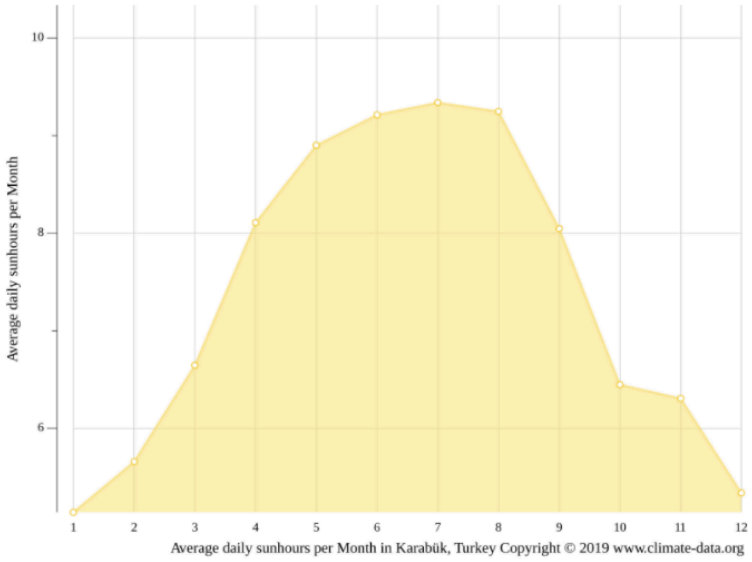


Figure 1. Sunny Hours in Karabük (climate-data.org, 2022)

As seen in Figure 1, the month with the sunniest hours per day in Karabük is July, with an average of 9.34 hours of sunlight. During July, there was a total of 289.47 hours of sunlight. With an average of 5.34 hours of sunlight per day, January is the month with the fewest sunny hours per day in Karabük. In January, there are 165.44 hours of daylight.

2.5. Finance and Taxes

It is obvious that, there are distinctions in operation of different types of mines depending on their location, weather condition, demand, supply etc. Because of those differences, the final financial cost can differ a lot from one project to another. Demand for a specific mineral product is affected not only by its own relative price, but also by the prices of products and labour force that are used during the processing (Hartman etal.,1992). The factors which can have an impact on the final income of the project must be analysed such as transportation and accessibility of the mine. As mentioned in other sections, our site is close

to Karabük is Zonguldak, Kastamonu, Bartın, Çankırı, Bolu and Ankara. It takes 1 to 2 hours to reach the mine location from these cities, which is not a very long period. It also means that transportation of labour workers who are living in the vicinity and fuel consumption of machinery will not cost very much. It is also worth considering that the area itself has a flat surface which decreases the cost of the construction of the worksite and enables more convenient movement of the machinery. As it mentioned before in Climate section the area is rainy. It means that the equipment and machinery can be exposed to corrosion and failures, which will increase the corrective maintenance cases and their costs. In mentioned dry months, additional preventive maintenance must be done to prevent the overheating issues of the components. The working conditions must be also checked for the safe working environment of the employees.

The financial part is not limited with mentioned type of costs, it also includes rent, taxation factors as well. As mentioned in “Permitting, Social and Community Impact” section, some of the permits are taken for a certain fee in Turkey. “Mining Law” by the Energy and Natural Resources Ministry regulates the details of taxation and pricing of fees in the mining plants of Turkey. One of the definitions that is worth considering in this law, is the State’s Right. It mentions that “The portion that falls to the State's share of the income to be obtained through mineral extraction and the payment obligation belongs to the license holder”. This portion depends on a type of the mine and in recent years, among the group IV. mines state right rate for aluminium, copper, zinc, iron, chromium, and lead was increased by 25 percent.

In the scope of Mining Policies of “On Birinci Kalkınma Planı”, financial support and tax exemptions will be applied to the companies which meet certain criterions. Support requirements and types are given in Table 2.

Table 2. Support requirements and types (T.C. Sanayi ve Teknoloji Bakanlığı, 2022)

SUPPORT ELEMENTS	General Incentive Applications	Regional Incentives	Encouragement of Large-Scale Investments	Promotion of Strategic Investments
VAT Exception	✓	✓	✓	✓
Customs Duty Exemption	✓	✓	✓	✓
Tax Discount		✓	✓	✓
Insurance Premium Employer Share Support		✓	✓	✓
Income Tax Withholding Support	✓	✓	✓	✓
Insurance Premium Support		✓	✓	✓
Interest Support		✓		✓
Investment Area Allocation		✓		✓
VAT Refund				✓

2.6. Permitting, Social & Community Impact

Interest in the mining industry is increasing due to the energy crisis, resource depletion, increase in resource use and wars. In recent years, with this increase, social, and environmental effects have started to be discussed and expenditures on these areas have grown considerably. Therefore, the subject of Social License to Operate (SLO) is one of the most popular areas in mining today. The government, community and, if any, company should cooperate. As the stages of exploration, construction and operation have different effects and require different studies, there are also changes in the permissions taken between states. In fact, since every society is different, special social licence operations are required.

In Turkey, there are many permissions to be obtained from the state and local administrators. Some of these permits are taken for a certain fee, while others are taken in line with the reports submitted. The list of permissions to be obtained is attached. On the other hand, Social License to Operate refers to an intangible and unwritten social contract with a

community or a social group that enables extraction or processing operation to enter a community, initiate and maintain operations (Franks & Cohen, 2012). At this point, we understand that the place where we will open a mine will affect the local people in many areas. Eflani is the closest district to the mine site. The number of people living in the city of this district is around 2110, and the number of people living in the village is around 6500. We see that the source of livelihood is husbandry and agriculture. The fact that the mine is on a forested land and is underground will not prevent the local people from farming. On the contrary, there are examples of fertilizer making with immature coal obtained from the Afşin-Elbistan thermal power plant. This fertilizer is known to eliminate the lack of organic matter in agricultural lands. In this way, fertilizer support can be given to the local people. Regular conferences should be held to inform local people and they should be kept informed of every action to be taken at the mine.

Participation in conferences to be held in the region should be ensured not only by men, but also by women. There will also be new initiatives in this region. Collaboration with these initiatives, training and the creation of related costs will provide the first steps to build trust with local people. Although it varies depending on geography, mining has positive employment and entrepreneurial effects (Komnitsas, 2020). Many jobs will be provided to the local people.

2.7. Population Centre's and Labour Force

The population near the mine site is one of the most crucial parameters for mining. Because the labour force is provided by the population. Moreover, the mining progress affects the population near to the mine. The mine site is in the Karabük city lines. Therefore, the population of the Karabuk has importance for the labour force.

The population of the Karabük city is 249,287. The population consist of 124,524 men and 124,764 women. Eflani is the nearest town to the mine site inside the KARABUK city. The population of Eflani was 8,499 in 2015. The population of the Eflani consist of 4,088 men and 4,411 women. Almost 60 percent of the population of Eflani is built by its villages. (TÜİK, 2015)

To provide the labour force for a mining, the employment status of the field plays a significant role. The young and unemployed population is the potential labour force for mining. The number of unemployed people in the Karabük was 8,908 people in 2012 (İŞKUR, 2012). The distribution of unemployed people with respect to the age is given the chart data table below.

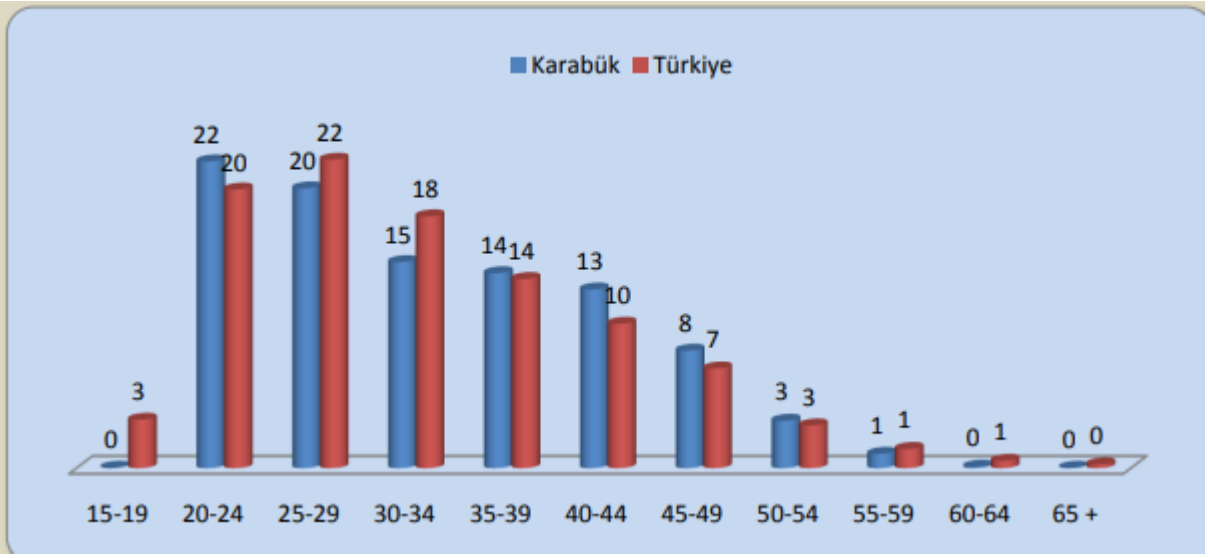


Figure 2. Age distribution of unemployed people in Karabuk (AKTAŞ, 2012)

2.8. Power Supply of Eflani, Karabük

The need for electricity in the formation of current residential areas has increased day by day thanks to the developing technology. Electricity infrastructure is one of the most important factors for the use of natural gas and other energy sources.

Power cuts also adversely affect the basic life in the Eflani region, both for agriculture and for the needs of the local people. It is known that power cuts for long periods caused great problems in the region. It is not a region with a fully established electrical infrastructure. It has been changed many times before, transferred over the forests, and later, extra poles have been erected and the infrastructure has been tried to be improved by moving it even higher. For this reason, long-term blackouts were experienced in the region.

The destruction of these forests and some of the vegetation in the lands has met with backlash from the local people. However, it can be said that there is a need for this to better provide the energy infrastructure.

In the Karabük region, research was carried out in different regions for the installation of wind power plants. For example, after the wind map of the Yenice region was

drawn, it was determined that the region was low or moderate for wind. For this reason, the installation of wind power plants in the region is not preferred (Arca, D., 2020).

The installation of solar power plants in Karabük was found to be reasonable according to a study conducted in 2022. In the general climate of the city, the sunshine duration is high. It was stated that an area of approximately 52 square kilometres within the Karabük area, which is 800 square kilometres on average, is sufficient for solar energy (Arca, D. & Keskin Çıtıroğlu, H., 2022).

The natural gas infrastructure is not yet available in the region, but it is demanded by both the factories and the regional administration. Previously, this issue was transferred to the parliament and a solution to the problem is still being sought.

As a result, there is enough energy in the region to meet the current need, but new resources are required as the energy need will increase with the additional investments to be made.

2.9. Present Ownership and Available History

Owner of the project site is Turkish government, and the mine exploration process done by General Directorate of Mineral Research and Exploration (MTA) which is a governmental organization. After the report of the drilling process is created, D|D Mining Co. applied for the necessary permits to operate the mining project.

2.10. Property Location

Most of the mining area is in the district of Eflani of Karabük, in the Özdere district. There are also some parts of the mine area in Yayla district of Araç district of Kastamonu province. These areas are defined as forested areas by the state. Since there is no private property, there is no need to communicate with the local people about the land.

There is Kızılkelik village in the north of the mining area, Göller and Saraycık village in the west and Demirli village in the east. It is about 1 hour away from the centre of Karabük. It is about 2 hours away from the centre of the Kastamonu. There is a developed industry in the province of Karabük. Operations such as machine spare parts or repairs can be easily handled here.

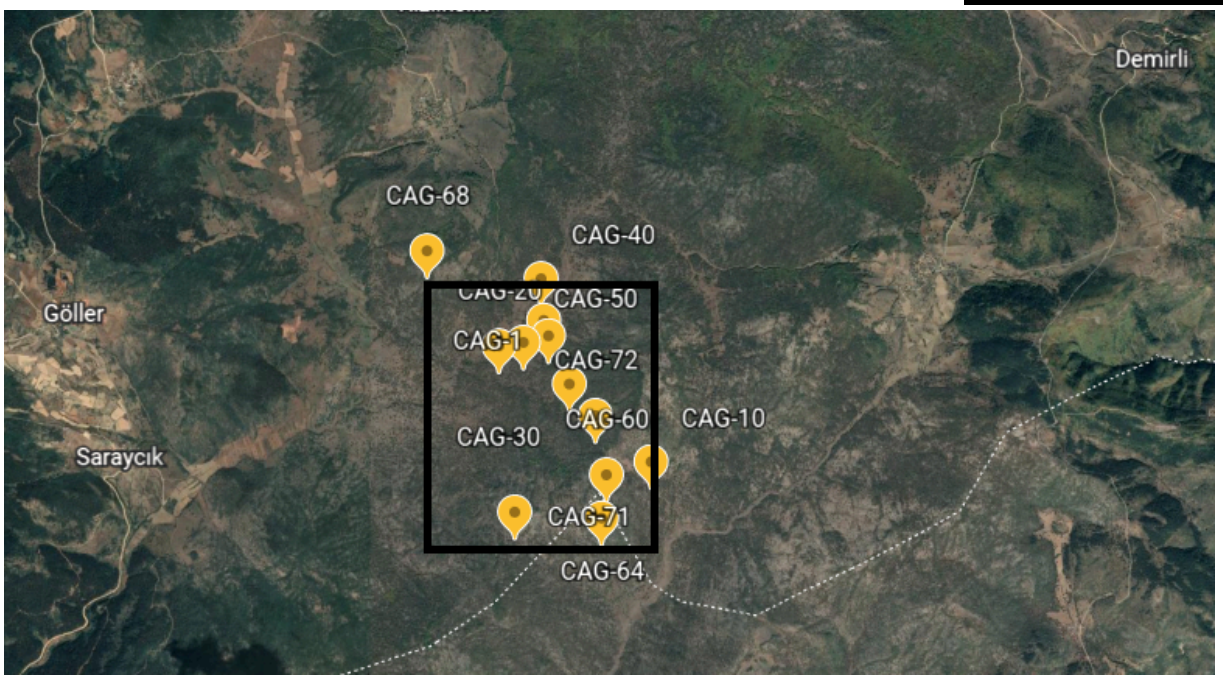


Figure 3. Location of the Mine Area (Google Earth, 2021)

2.11. Stakeholder Commitments and Environmental Concerns

Stakeholder commitments are protected by government and the permits that are required to open a field for mining activities that are explained in the "permitting, social and community impact" part. These commitments also explained in Environmental Impact Assessment (EIA) report as well. According to the EIA report, which is done in 2020 for Karabük, is explained the environmental concerns to the people who are living nearby the mine area. Additionally, EIA report also includes that both fauna, flora disturbance and biodiversity.

2.12. Topography

Area has a flat surface (Figure 4). Average height is approximately 1000 m on the main area. Nearest mountain on the area is the Gevrelik Tepesi with 1241 m elevation. Gevrelik Tepesi located in the middle of the Ovacık, Küpelik, and Demirli villages. (Marked on the Figure 3**Error! Reference source not found.**) Since the project area has a flat surface, it is easier to build worksite, construct road and operate the project. If the project needs to be extended over a wider area, there are some mountainous areas that can be seen on the Figure 4.

There is Cibiş Canyon near the mine area. Cibiş Canyon can be seen on Figure 4 with blue colour. Cibiş Canyon is the lowest area in the mine area. Also, it is the roughest area for operating and the canyon is being used for some touristic travels. Therefore, it is hard to operate a mine near the canyon.

The topographic conditions of the region seem good for both underground and surface mining methods.

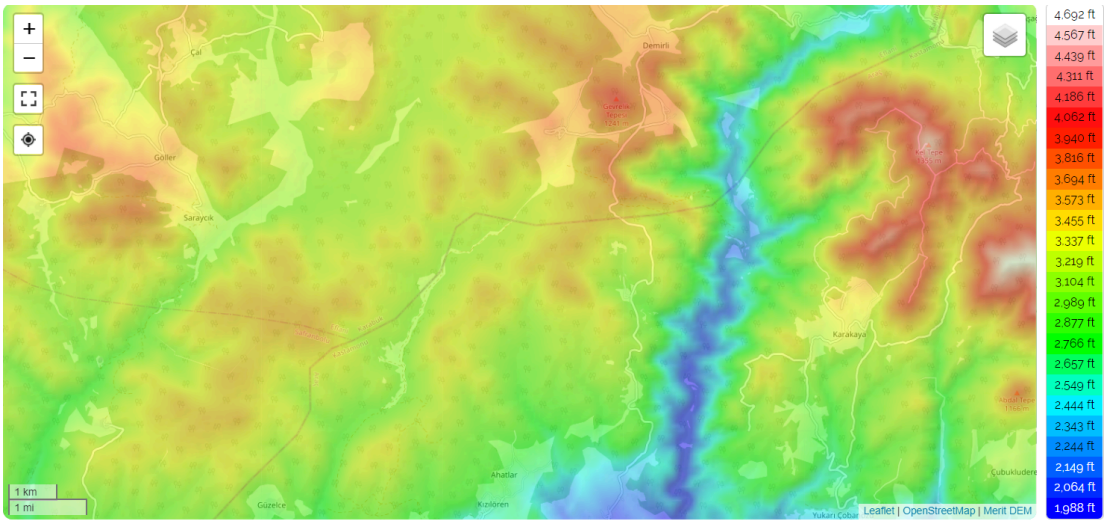


Figure 4. Elevation Map of the Area (topographic-map.com, 2022)

According to Figure 5, there are scrubs on the area and little forests near the project site. Since the area consists of scrubs, the number of trees that need to be cut will be few. Flat surface of the project site, small hills near the area.



Figure 5. 2D View of the Area (Google Earth, 2021)

2.13. Transportation and Accessibility

The neighbourhood cities of Karabük are Zonguldak, Kastamonu, Bartın, Çankırı, Bolu and Ankara.

The distance from Karabük to Zonguldak is 101 km and it takes 1 hours and 32 minutes by car. Between the two cities, railroads exist. This leads to advantage for transportation of material from Karabük to Zonguldak. When the material comes to the Zonguldak with the consideration of the existence of the port there, the material can be transported easily by seaway.

Additionally, Erdemir Company, located in Zonguldak, uses a lot of coal for steel production. By using the railways between Karabük and Zonguldak, the coal need of the Erdemir Company can be met. Therefore, this ease of transportation makes it easier for us to sell our material.

The distance from Karabük to Kastamonu is 112 km and it takes 1 hours and 22 minutes by car. Our mine site is located between the Karabük and Kastamonu bord. This location is very important due to the diversity in the labour force from different places. The fact that the workers in the mine site are from different cities may be important in terms of increasing production efficiency by employing different people in the necessary positions in terms of efficiency. (People in the Ahatlar Village is very suitable for this purpose)

The distance from Karabük to Ankara is 220,9 km and it takes 2 hours and 27 minutes by car. Generally, mining companies have their head offices in the capital Ankara. Being close to Ankara ensures that both the work with the bureaucracy and the work within the company are relatively faster. It can be reached by railway.

The distance from Karabük to Bartın is 89,8 km and it takes 1 hours and 28 minutes by car. Since it is a coastal city, it provides convenience in terms of transportation of the extracted material to overseas regions for commercial purposes.

The nearest airport to Karabuk is Zonguldak Airport (ONQ) 56.19 km, Esenboğa International Airport (ESB) 124.37 km, Hasan Polatkan Airport (AOE) 235.35 km.

Karabuk can be reached from the following airports: Vienna (VIE), Cologne (CGN), Baghdad (BGW), Dortmund (DTM), Frankfurt am Main (FRA), Belgrade (BEG), Düsseldorf (DUS), Izmir (ADB), Erbil (EBL), Brussels (BRU), Stuttgart (STR), Doha (DOH), Moscow (VKO), Munich (MUC), Bodrum (BJV), Istanbul (SAW), Baku (GYD).

2.13.1 Accessibility to Surrounding Villages

The villages around the mine site are Cal (Kizilkelik), Demirli, Göller, Saraycık and Ahatlar. There is existing road connection to these villages, but we will need to plan new routes to reach the mine site according to the mine plan. This planning should also be done by considering the slope and land conditions of the region. The average distance of the villages to the mine site is 15 km.

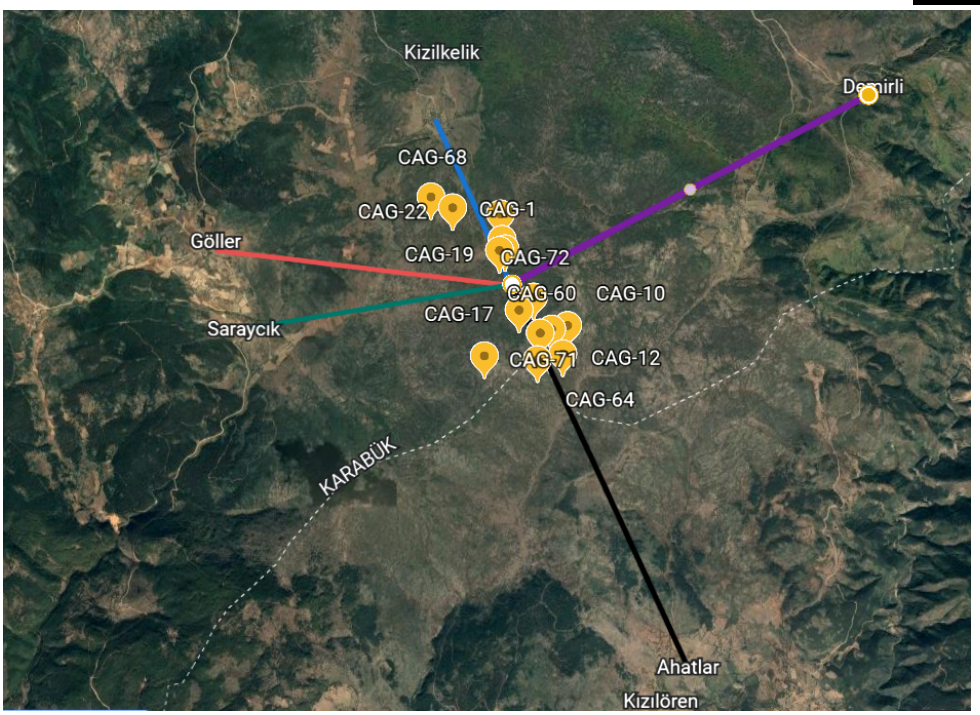


Figure 6. Road to mine site from different places (Google Earth ,2021)

2.14. Water Supply of Eflani

One of the most important factors in providing normal living conditions and establishing civilizations is the water resource.

It can provide water resources and make new probes to nearby villages, which are large and have mineral resources that will affect the local population, if they are not sufficient. Apart from this, water consumption is also high during the extraction and processing of normal mines. To reduce these amounts, the plant waters are used repeatedly. To reduce water costs, natural water resources are used and new probes, ponds, dams can be created.

There is no lake as a natural water source in the Eflani district around Karabük, where the study was conducted. However, in line with the needs of the surrounding people, 3 artificial lakes were built in the region and the water needs were tried to be solved. It is a region that receives precipitation due to its climate and being surrounded by high mountains, valleys, and forests. This advantage has been resolved with ponds. These ponds are Bostancılar Göleti, Kadıköy Göleti and Ortakçiler Göleti.

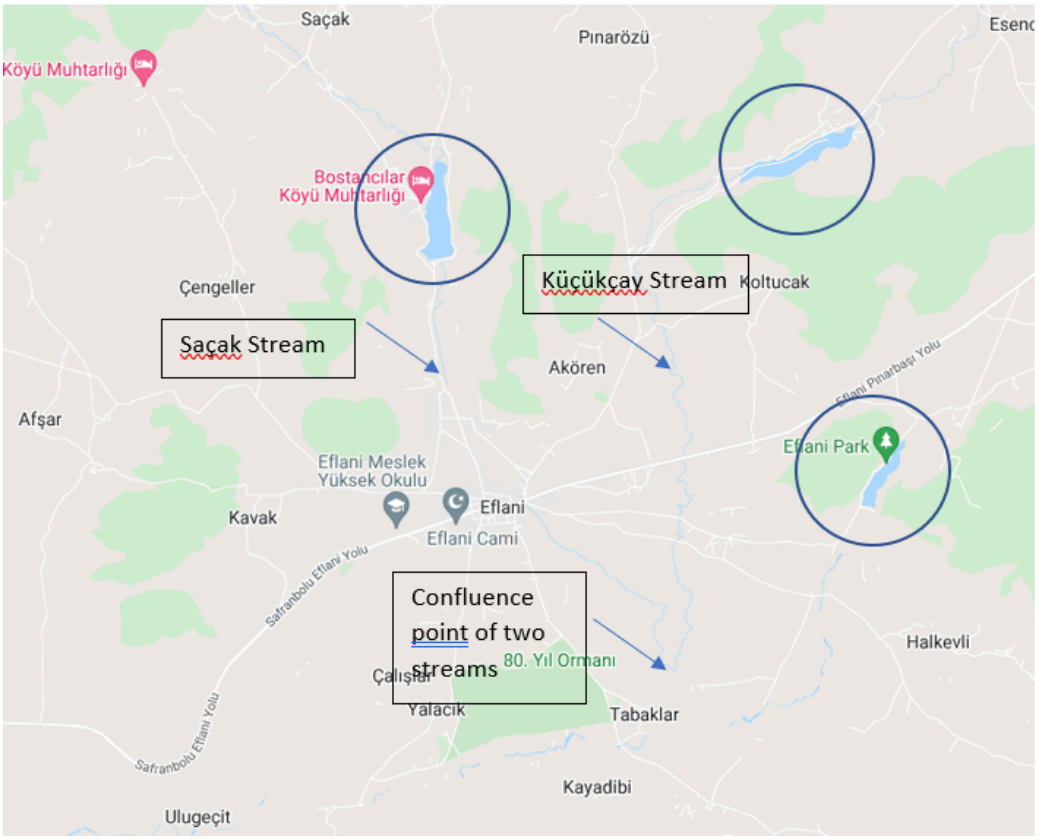


Figure 7. Water Resources in Eflani (Google Maps, 2022)

Apart from these ponds, there are 2 streams in the region. These are Saçak Stream and Küçükçay Streams. There is Karagöl Stream and at a distance from the main centre, and it merges with Küçükçay Stream nearest the Tabaklar Village.

3. GEOLOGICAL, HYDROLOGICAL AND ROCK ENGINEERING STUDIES; CHARACTERIZATION OF THE ROCK MASS

Under this tittle, topographic maps, geological conditions, minor and major discontinuities, and rock engineering studies.

3.1. Topographic Maps

The distance of the region to Ankara, the capital of Turkey, is between 280 km on average from the highway. From a bird's eye view this distance is even shorter. Cide, the northernmost region of Turkey, and Kastamonu region have an average bird's eye view of around 100 km.

When the general elevation of the region is examined, it can be said that it is a flat land since it is behind the mountains. When the figure is examined, the average mining area in the region is formed as in the figure, according to the drills made at the bottom. Drills in the most border regions can be called CAG10 - CAG68 - CAG64. Locations of four drill holes listed in Table 3. In Figure 8, the topography map is shown.

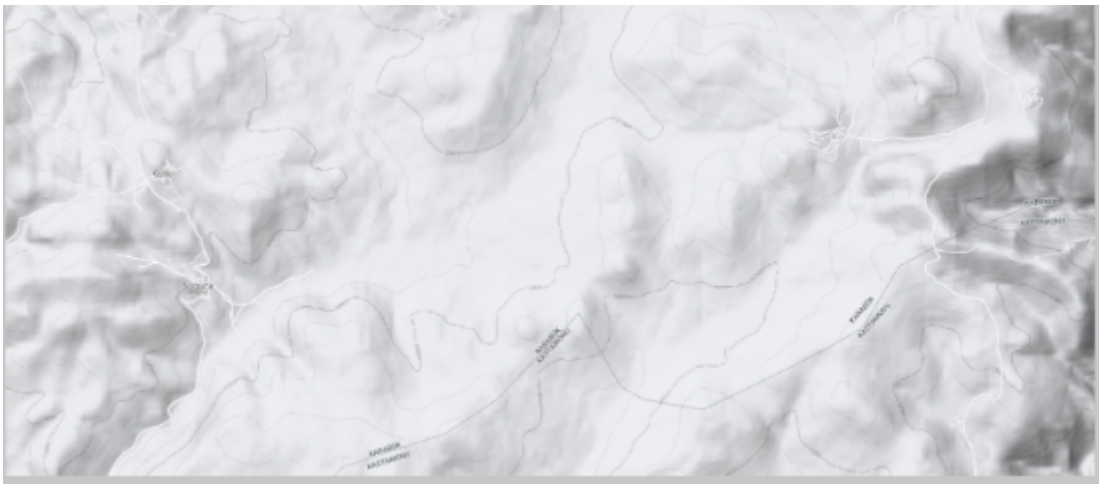


Figure 8. Topographic Map of the Mine Area (Google Earth, 2022)

Table 3. Locations of Four Drill Holes

Drill Holes	Easting	Northing	RL
CAG-10	500118,6	4573562	978,066
CAG-71	498863,8	4573107	1015,805
CAG-40	499107	4575239	965,941
CAG-64	499661,7	4573049	998,906

3.2. Geological Conditions, Minor and Major Discontinuities

According to Karabük Provincial Environmental Report which published by Environmental and Urbanization Ministry in 2011 in Karabük and its surroundings, located in the western part of the Black Sea Region, many lithological units of different ages from the First to the Quaternary are observed. The mine area is limited by the Çağlayan Formation and the Anatolian nappe from the south, and the pre-Jurassic metamorphic and the Çağlayan Formation from the north and northwest. The Paleozoic aged Formations, constituting the basement units in the region, cropped out in limited areas in the west, north and east. Mesozoic aged units overlying the basement units are represented as limestone's of İnaltı Formation and as flysch series in Ulus Formation. The horizontal and near-horizontal layered units at the top are composed of Cenozoic aged formations. Quaternary alluvial deposits form the youngest units in river valleys.

The calcareous (limestone) lands formed in the third geological time in Karabük occupy a large area. It is found in clayey and sandy layers among the limestones. The calcareous land around Safranbolu and Eflani has undergone metamorphism and rich marble beds have been formed. Besides marble, there are also flint and coal deposits in Eflani.

Tectonic movements are observed in the east-west direction in the Cenozoic aged sediments covering large areas. The Karabük fault, which is seen along the contact of Mesozoic and Cenozoic aged formations in the northwest of the Karabük-Safranbolu settlement area, is a reverse fault pushing the Cretaceous aged units onto the Cenozoic aged younger units.

96% of Turkey's lands are in danger of earthquakes. Especially around the North Anatolian Fault (NAF) constitutes the most active region. The North Anatolian Fault Line is 55 km away from the city centre of Karabük.

The reverse fault, known as the Karagöl Fault, which borders the south of the Karabük Tertiary Basin, again pushed the Ulus Formation and small outcrops Tertiary units into the Karabük Basin.

In Figure 9, The distribution of Fault lines of Turkey is shown.

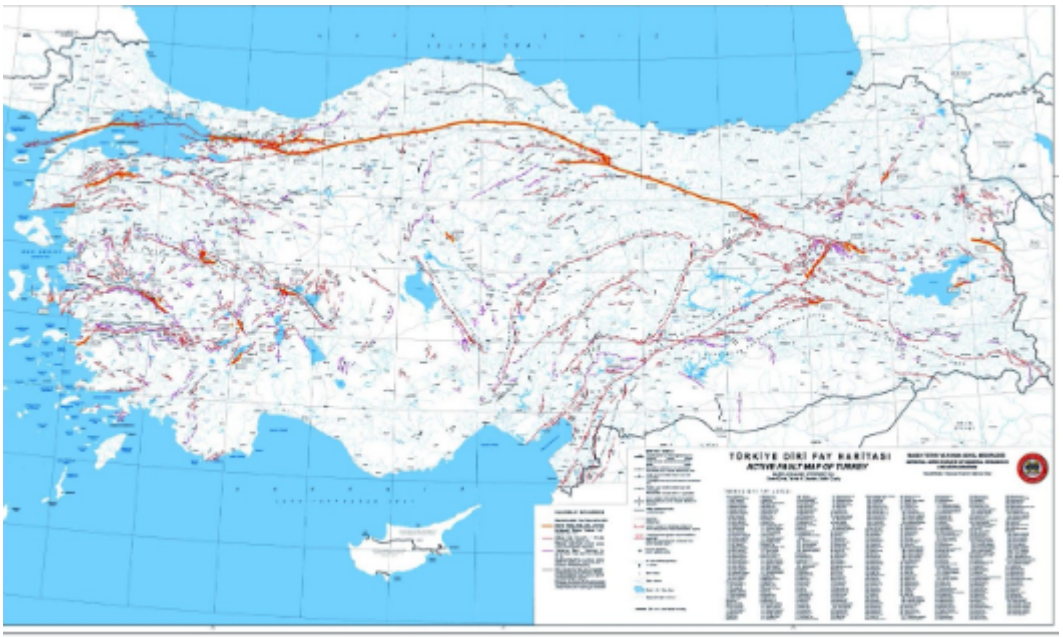


Figure 9. Distribution of Turkey Fault Lines (Jeoloji Mühendisleri Odası, 2012)

3.3. Hydrological Conditions

As described in the title of Water Supply of Eflani, it does not have a natural lake. However, the region is not inadequate in terms of water resources. The region receives a lot of precipitation due to the main features of the surrounding large and tall mountains and additional vegetation, with an average annual precipitation of around 605.44mm. The highest temperatures are usually observed in August at 40 degrees, while the lowest temperatures are usually observed in January at -5 degrees Celsius.

The humidity in the region is sensibly high and around 70%. It can be said that the region is experiencing the windiest times during the low temperature periods, namely January.

While talking about the water resources of the region in detail, Saçak Stream, Küçükçay Stream and Karagöl Stream are the rivers of the region. These then continue by

merging in the Tabaklar region. The water used in agriculture in the region comes from both these streams and abundant precipitation.

3.4. Rock Engineering Studies

Rock engineering studies have great significance for mining activities. The stability of the mine site is carried out in rock engineering studies. Thanks to rock engineering studies, the dimensions of adits, ramps, panels, underground pillars, the overall slope angle of open pits, bench heights, and the overall slope angle of waste dumps are determined. To determine rock mass properties, Rosscience's Rocdata software is used.

3.5. Laboratory Tests

Drill holes provide geologists cores. The cores are used to gain an idea about rock mass properties, grade, joint, and faults through laboratory tests. There are seven kinds of different rock types in the topography. The rock types are Conglomerate, Clay, Limestone, Marl-Limestone, Marl, Shale, and Coal. The chemical and physical properties of these seven different types rock are explained in the subtitle of Properties of Rocks.

The laboratory tests are, Uniaxial Compressive Strength and Deformability Test and Triaxial Compressive Strength Test. Thanks to the laboratory testing, the rock's Unit Weight, Elastic Modulus, mi values, Geologic Strength Index(GSI), Uniaxial Compressive Strengths(UCS), and Poisson Rations are calculated. The laboratory test results are shown in Table 4 below.

Table 4. Laboratory Test Results

Lithology	GSI	UCS(MPa)	Elastic Modulus(GPa)	Poison Ratio	Unit Weight (kN/m^3)	mi values
Conglomerate	55	34	7.5	0.25	22	21
Clay	40	13	3.85	0.19	18	6
Coal	50	51	9.13	0.2	18	10
Marl	60	28	6.87	0.23	19	7
Shale	45	51	17.5	0.28	21	8
Limestone	55	59	14.2	0.26	23	12

The laboratory test results show the intact rock properties. However, in the mining process, there are joints, faults, and discontinuities to decrease the rock's properties. To obtain fractured rock mass properties, Rocscience's Rocdata software is used.

3.6. Properties of Rock Types

Conglomerate

The conglomerate is a clastic sedimentary rock composed of rounded gravel and boulder-sized clasts cemented or supported in a matrix. The clast cement is typically calcite, silica, or iron oxide, but the matrix can also include sand and/or silt-sized clasts cemented together with the various coarser clasts. The conglomerate generally is soft to complex dependent on clast composition and strength of cement (GeologyScience, 2022).

Clay

Clay is a naturally occurring material made up of fine-grained minerals. The minerals are typically less than 2 microns in size and have a plastic water content that solidifies when dried. Clay is the most abundant mineral on the earth's surface, forming shale rocks and serving as the primary component of sedimentary rocks. Clay materials have unique properties due to their small particle size and unique crystal structures. Cation exchange capabilities, plastic behaviour when wet, catalytic abilities, swelling behaviour, and low permeability are among these properties. Clay materials have unique properties due to their small particle size and unique crystal structures. Cation exchange capabilities, plastic behaviour when wet, catalytic abilities, swelling behaviour, and low permeability are among these properties. The specific gravity of most clay minerals ranges between 2 and 3.3. Their hardness is usually less than 2.5. Clay mineral refractive indices generally fall within a relatively narrow range of 1.47 to 1.68 (Mana, 2017).

Limestone

Limestone is a sedimentary rock that contains more than 50% calcium carbonate (calcite – $CaCO3$). There are numerous unique types of limestone formed through a variety of processes. It can be precipitated from water (non-clastic, chemical, or inorganic limestone), secreted by marine organisms such as algae and coral (biochemical limestone), or formed from the shells of lifeless sea creatures (biochemical limestone) (bioplastic limestone). Some limestones (clastic limestone) are formed by the cementation of sand and/or mud with calcite and have the appearance of sandstone or mudstone. Calcite, the primary mineral component of limestone, will fizz in dilute hydrochloric acid. The colour of limestone can be various

such as yellow and white. Limestone is generally hard material. Hardness of limestone is 3 to 4 on Moh's scale. Limestone has quite low porosity (GeologyScience, 2012).

Marl

Marl, also known as Marlstone, is a non-clastic sedimentary rock composed of calcium carbonate. It contains a variety of clays and silt. Calcite, aragonite, dolomite, and siderite are the most common carbonate minerals found in marls. Marls are usually whitish grey or brownish, but they can also be grey, green, red, or variegated. The green mineral glauconitic is found in greensand marls, and iron oxides are found in red marls. Marl is much more difficult to split than shale and tends to break up in blocks. Specimens are frequently nodular, with nodules that are usually better cemented than the surrounding rock. Hardness of marl is 2 to 3 on Moh's scale (GeologyScience, 2022) .

Shale

Shale is a laminated or fissile clastic sedimentary rock composed primarily of silt and clay with minor amounts of other minerals, particularly quartz and calcite. Fragility is a property of shale that refers to breaks along thin laminae or parallel layering or bedding. It is the most common sedimentary rock. The composition (silt and clay) of shale in mudstone, a type of sedimentary rock. The difference between shale and mudstone is that it is fissile and laminated. Along the laminations, shale rock easily fractures into thin pieces.

Shales are fissile clastic sedimentary rocks formed by the transportation, deposition, and compaction of silt and clay detrital materials. The clay's Fragility is its primary distinguishing feature from other sedimentary rocks. Fragility is defined as a rock's ability to split easily along thin, closely spaced (approximately 10mm) parallel layers. This Fragility factor is highlighted in, which depicts the classification of sediments and sedimentary rocks based on fragment size (Comunale, 2021).

3.7. Rock Mass Classification

The classification of rock mass proceeded in three methods. The methods are Geological Strength Index (GSI) Rock Mass Rating (RMR) and Q-System. (Hoek & Marinos, 2000). To have more precise information about rock mass classification, more than one method is used to classify.

The GSI values are already obtained from the laboratory tests by using the surface conditions of discontinuous and the structure of lithology. In Figure 10, the GSI values of each rock are marked with a different colour in different GSI values. The GSI values are informed us about the structure and surface conditions.

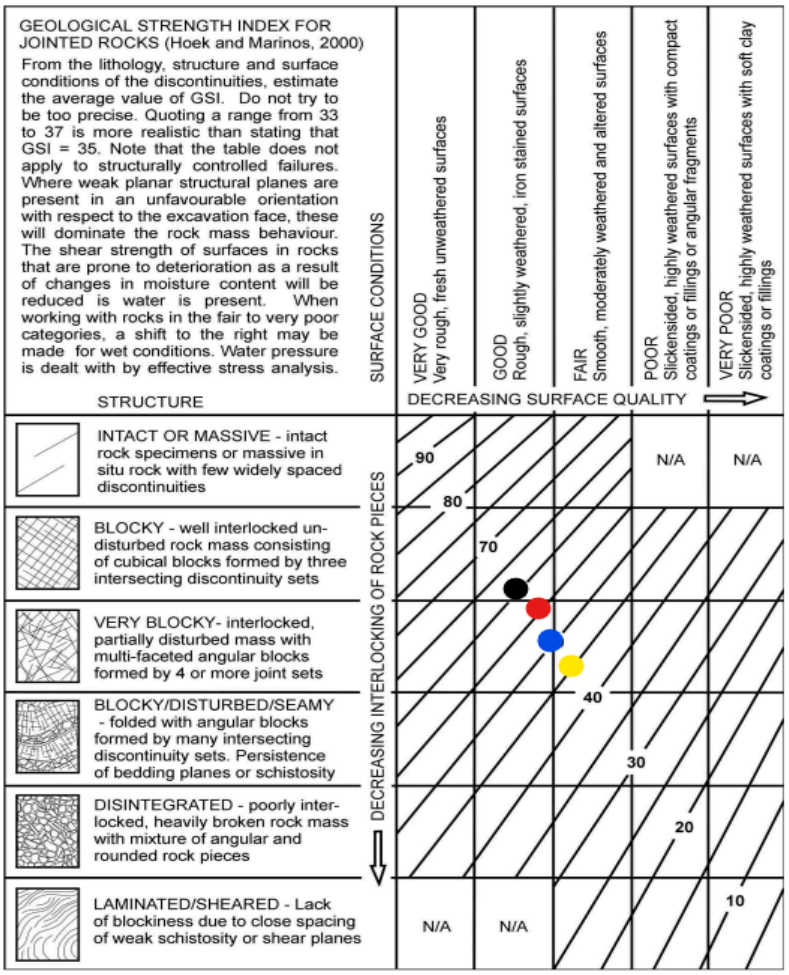


Figure 10. Geologic Strenght Index Chart (Hoek & Marinos, 2000)

Each colour represents a type of rock. In Figure 11, the colour legend is shown.

Color	Lithology	GSI
	Marl	60
	Conglomerate	55
	Coal	50
	Shale	45

Figure 11. Color Legend of GSI

The GSI values can be converted into the RMR and Q values.

$$RMR_{89} = GSI + 5 \text{ (Hoek \& Marinos, 2000)}$$

According to this equation, the RMR value is calculated and listed in Table 5.

Table 5. RMR and GSI values of Rocks

Lithology	GSI	RMR_{89}
Conglomerate	55	60
Clay	40	45
Coal	50	55
Marl	60	65
Shale	45	50
Limestone	55	60

According to RMR values, stand-up times of rocks can be evaluated for specific roof heights.

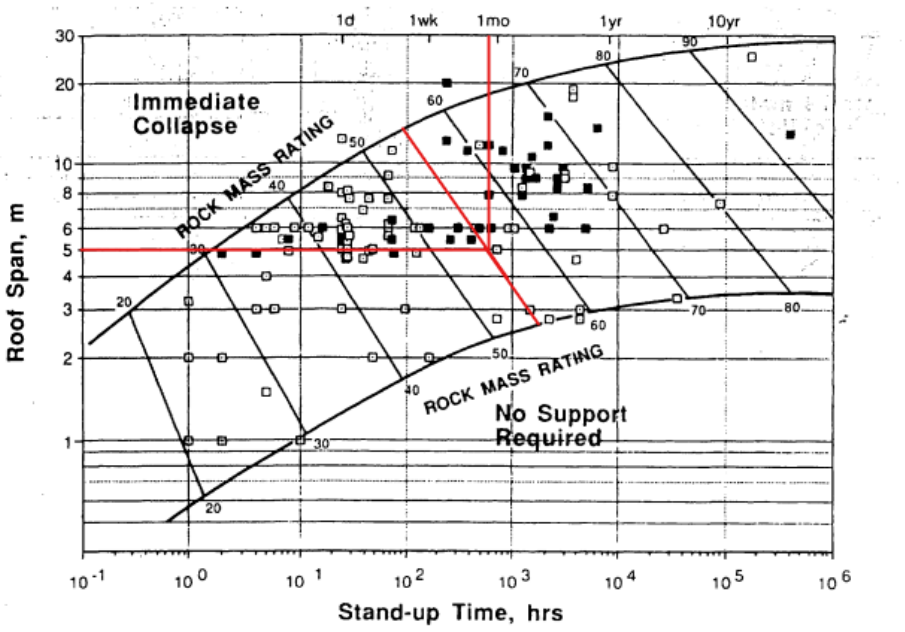


Figure 12. Relationship between Roof Span and Stand-up Time for Coal (Z.T.Bieniawski, 1989)

In Figure 12, the red line represents the RMR value of coal and roof height of roadways. The stand-up time is nearly 1 month for 5-meter roof span for roadway. In Table 6 , the roof span for 5 m and stand-up times are labelled for all rock types.

Table 6. Roof Span and Stand-up Times of Rocks

Lithology	Roof Span (m)	Stand-up Time	RMR_{89}
Conglomerate	5	4 months	60

Clay	5	2 days	45
Coal	5	1 month	55
Marl	5	nearly 1 year	65
Shale	5	1 week	50
Limestone	5	4 months	60

The Q value is also used to classify the rock masses. Each rock has a specific Q value with respect to Q System. The relationship between RMR value and Q value is shown in the correlation below:

$$RMR_{89} = 9lnQ + 44 \text{ (Bieniawski, 1989)}$$

Thanks to this correlation, the RMR values of rocks can be converted into the Q values and can be examined by Q System. In the Q system, besides the classification of rock mass, required support properties such as bolt spacing and lengths can be obtained for Span of height.

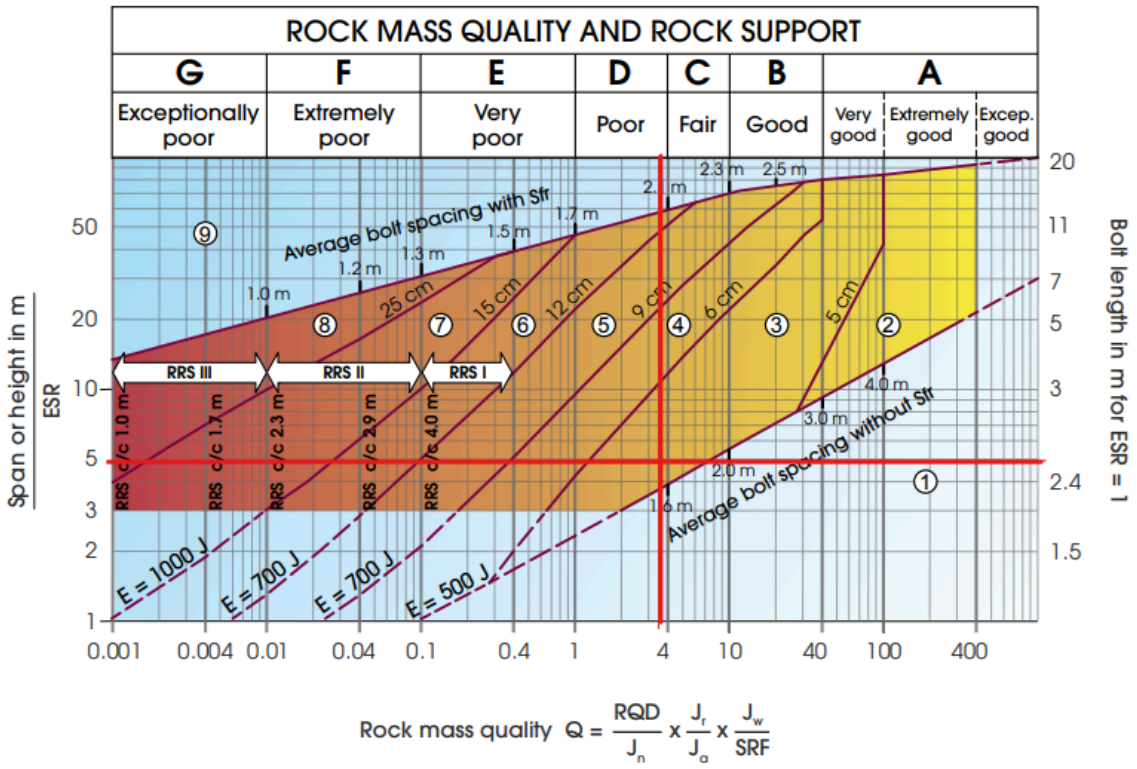


Figure 13. Rock Mass Quality and Rock Support of Coal (Z.T.Bieniawski, 1989)

The rock mass quality and rock bolting, lining and bolt spacing of coal is shown in the Figure 13. Rock mass quality for coal is poor. For 5 m span height, the bolt length is 2.5 meter. Moreover, the bolt spacing is differing if shotcrete is applied or not. When the shotcrete is applied for 6 cm, the bolt spacing is 2.7 m. Otherwise, the bolt spacing is 1.9 meter without shotcrete.

According to the RMR, GSI and Q values, the rock mass can be classified. The rock mass classification is listed in Table 7.

Table 7. Rock Mass Classification

Lithology	GSI	RMR	Q Value	Classification with respect to Q value
Conglomerate	55	60	5.92	Fair
Clay	40	45	1.12	Poor
Coal	50	55	3.39	Poor
Marl	60	65	10.31	Good
Shale	45	50	1.95	Poor
Limestone	55	60	5.92	Fair

3.8. Rock Mass Properties

In the laboratory tests, the intact rock properties are obtained. However, in the mine, rocks are exposed to fractures, joints, and faults. Therefore, the rock mass properties are weakened. To obtain the rock mass properties, the Rocscience's RocData software is used. By using the Elastic Modulus, mi values, GSI, Intact Uniaxial Compressive Strength, and Unit Weight of rocks as a result of laboratory tests, cohesion, friction angle, uniaxial compressive strength, tensile strength, and modulus of deformation can be found for the rock mass. Rock mass parameters are shown in Table 8. Because of the geological events, there are two different marl and limestone layers at different depths. Therefore, they show different rock mass parameters as deep and shallow rock types. To obtain this difference, the rocks are evaluated at different depths. In Table 8 , rock mass properties are listed.

Table 8. Rock Mass Properties

	Hoek and Brown Classification					Hoek and Brown Criterion			Mohr Coulomb Criterion		Rock Mass Parameters			
Lithology	**Intact UCS (MPa)**	**GSI**	**mi**	**D**	**Intact E (MPa)**	**mb**	**s**	**a**	**c (MPa)**	**phi (°)**	**sigt (MPa)**	**sigc(MPa)**	**sigm(MPa)**	**Modulus of Deformation(MPa)**
Conglomerate	34	55	21	0.7	7502	1.77	0.0015	0.5	1.7	31	0.16	1.27	6	1100
Clay	13	45	6	0	3846	0.84	0.0022	0.5	0.5	25	0.08	0.58	1.6	860
Coal	51	50	10	0.2	9132	1.37	0.0026	0.5	1.76	33	0.32	2.5	8	2111
Limestone	59	55	12	0.4	14170	1.6	0.003	0.5	2.9	30	0.4	3.22	10	3332
Marl	27.9	60	7	0.2	6870	1.43	0.008	0.5	1.36	29	0.32	2.54	4.76	2809
Shale	51	45	8	0	17532	1.12	0.0022	0.5	1.62	31	0.29	2.3	7.15	3921
Deep Marl	27.9	60	7	0.2	6870	1.43	0.008	0.5	1.5	28	0.33	2.54	4.76	2809
Deep Limestone	59	55	12	0.4	14170	1.6	0.003	0.5	2.44	32	0.42	3.22	10	3332

4. RESOURCE & RESERVE EVALUATION

In this section, the reserve and resource estimation of the ore will be made using drilling data. Drilling data was first used to identify the source. After the resource was determined, optimization was carried out and the parts where the resource would be reserved were determined.

4.1. Evaluation of Drill Hole Logs and Descriptive Statistics

After visualizing 71 drilling data on Micromine Software, the impact area of the drilling data was determined as 4.5 km^2. Besides the different coloring of the rock types and coal, the coal was named in 3 different groups (coal1, coal2 and coal3). These 3 coal groups were made according to their heights and the proximity of the drilling data to each other. In this way, 3 different coal seams were created while making the reserve model. Below is a visualization of boreholes on the Micromine Software.

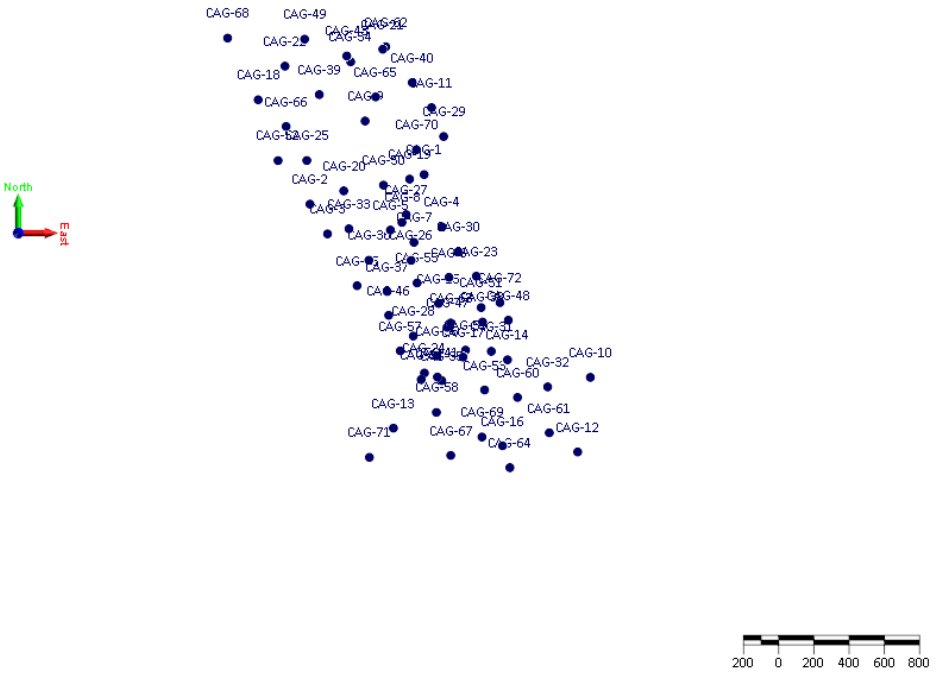


Figure 14. View from Top View

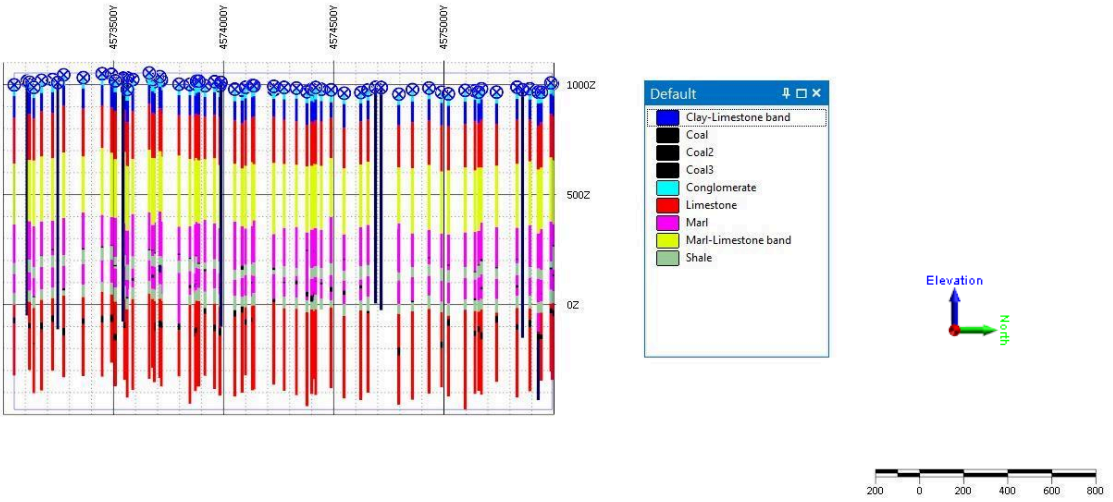


Figure 15. West View of Drill holes

For a good resource evaluation and grade distribution, it is first to create the search ellipsoid. Statistical studies are needed for the ellipsoid. To begin, Micromine software does a standard static analysis, which provides some broad information such as whether the presented data fits a normal or natural log distribution. The operations and outcomes are listed below.

With the quick statistical overview method, values such as mode, median, maximum calorific value, minimum calorific value, standard deviation, and coefficient of variance were analysed with the inputs in the project. The Table 9 shows a summary of the data.

The summary of the test file gives us preliminary information and hints for the block model. The most important part of this simple statistical explanation is the coefficient of variance. The coefficient of variance is the ratio of the standard deviation to the mean. As a result, the larger this coefficient is, the more variable the data is.

Table 9. Statistical Parameters of Ore Bodies

	Calorific Value(kcal/kg)
Maximum	7,457
Minimum	5,004
Sum	5,312,543
Variance	242,716.095
Mean	6,347.12
Standard Deviation	492.66
Coeff. Of Variation	0.078
Median	6,398
No Of Points	837
Geometric Mean	6,327.19
Harmonic Mean	6,306.36

The 25 and 75 percentiles, which are the first and third quartiles, may be easily read from the preceding graph, as can the mean and median. The above-mentioned statistical parameters are shown in Table 9.

The histogram, probability and cumulative frequency plots are also very useful to see if the data in the assay file conforms to a normal or log-normal distribution. Although the cumulative frequency plot shows sum of all frequencies in a calorific value, the histogram shows frequency value of data of various calorific value ranges as percentage. Here, as a result, the graphs show that the most appropriate distribution for the given data is the log normal distribution.

In order not to affect the average grade much, the balance cutting procedure was applied. According to the Rule of thumb, the calorific value corresponding to 97.5 percent on the histogram graph is approximately 7090. With the code written on the test file, values above 7090 calorific values were printed as 7090. On the other hand, bottom-cutting procedure was not required.

The next step of the ellipsoid required to create a block model is to create a composite file. For the composite file, the structure of the assay file was changed, and a new column called thickness was added. This method is also known as the method of generating individual sample values. The result from minus to is printed in the thickness column. Here

again, the process of writing code on Micromine software was applied. In the figure below show the photo of some of the printouts in the thickness column. According to descriptive statistics, the mean value is 56.95, the median is 3.55, and the standard deviation is 90.31.

The composite file is created to obtain the semi-variograms. Semi-variogram is one of the operations required for ellipsoid information. Here, besides the effect spectrum, the sill and nugget effect are also obtained. The nugget effect refers to measurement inaccuracies caused by data sampling, processing, and calculation. Within a field of influence, other points have less influence on measurements. Downhole, omnidirectional, and directional are the variograms used for this article. The importance of the nugget effect and sill values can be seen right here while making these variograms, and they are also used. These three variograms were made in the same order as written above, but for different purposes.

The first variogram, the downhole semi variogram, was created to achieve the nugget effect. In order for this process to be successful, that is, the number of nugget effects, the length of the longest lag interval borehole was taken into consideration. Accordingly, the intervals were set as 25 and the number of intervals as 55. These values give us approximately 1350 m of borehole. Nugget effect is recorded as 0.0015.

The second variogram is the omnidirectional semi-variogram. Here, the intervals were determined according to the state of the log variance line, and the most appropriate interval was found to be 40, while the number of intervals was found to be 55.

The last variogram is the directional semi-variogram. The fact that the search ellipsoid will give its data also indicates that the last variogram is the most important variogram. Currently, the search ellipsoid's three-axis influences are regarded as mean variables, with azimuth and plunge values assigned by instructions. The tolerance and bandwidth choices in Figure 16 enable for enough points to be covered in one direction while preventing improper study. To put it another way, these characteristics limit the search field. As a result, the azimuth distance is 200 m, and the resistance is 22.5 degrees. For a bandwidth of 40 m, plunge tolerance is commonly employed at 22.5 degrees.

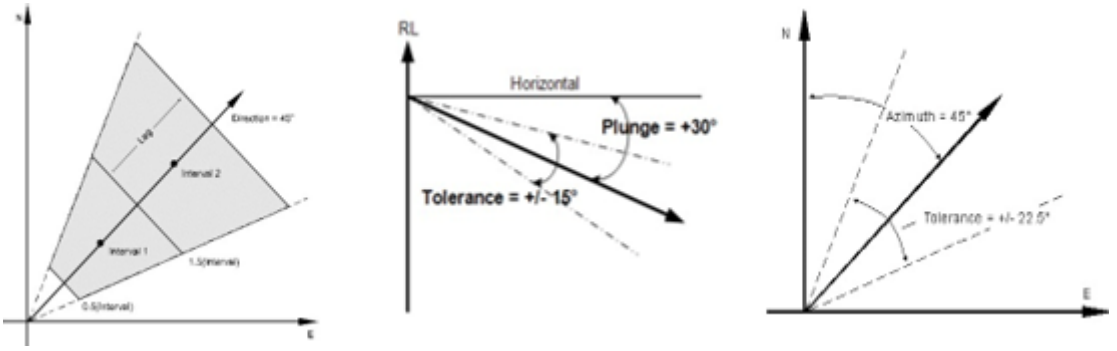


Figure 16. Parameters For Directional Semi Variogram (Micromine Manuel,2022)

The main task of the search ellipsoid is to assist in assigning a value according to the ellipsoid parameters to the empty blocks to be created using the IDW and Kriging methods. The ellipsoid obtained after the completion of the geo-statistical studies is shown in Table 10. Table 10, belong to the parameters of the ellipsoid. Although the distance between the boreholes varies, the average is 200 m, and the ellipsoid radius is set as 220 m. It will assign sufficient calorific value for IDW and Kriging measurements.

Table 10. Parameters of Ellipsoid

Axis	Factor	Azimuth	Plunge	Rotation
1	1.000	160.000	0.000	0.000
2	0.170	250.000	0.000	
3	0.125	250.000	90.000	

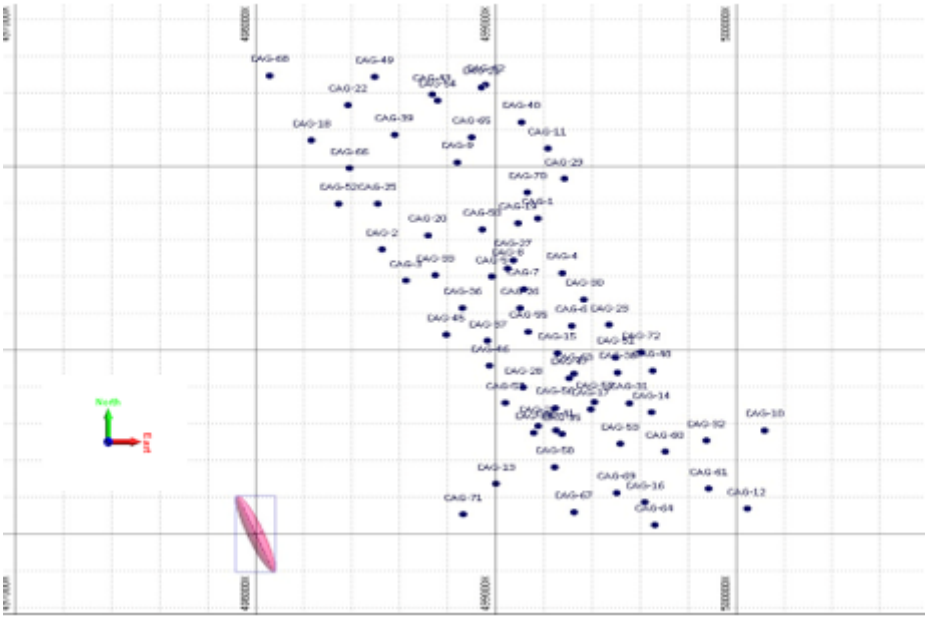


Figure 17.Search Ellipsoid Visualization

4.2. Selection of Grade Estimation Method

There are two widely used strategies for estimating grades. These are the IDW and kriging techniques. The geological data composition, which may be obtained using statistical analysis and variogram modelling, determines the grade estimation approach.

To assign grade scores to blocks that are weighted with $1/d^2$, the inverse distance weighting method is employed. The distance between a block with a known grade and a block with no grade is denoted by "d." The "d" value's power can be modified from 2 to 3 or any other number. The ore kind determines this. This power can be calculated as 2 in the case of coal. The data set does not need to follow a conventional distribution to utilize this form (Demirel & Güyagüler, 2008).

Kriging is a local estimate approach that approximates values from sample data using weighting factors. This weight is determined by the geometrical relationship between the sample and the panel, as well as the component's spatial features. The data set must have a regular distribution to apply the standard kriging technique (Demirel & Güyagüler, 2008).

The data exhibits a nearly normal distribution, according to statistical analysis and variogram analysis. As a result, both the inverse distance weighting method and the standard kriging method are sufficient for our project. Following the creation of the block model, both strategies will be evaluated, and the optimum method for estimating the grade and other properties will be chosen. After the model has been developed, both strategies will be applied

in the next stages, and a decision will be reached as to which strategy is superior for estimating the grade and other features.

4.3. Resource Modelling

The resource and reserve values of the project given to us will be calculated over Micromine application. The first job is to draw cross section strings to include all of the drilling data. Since coal groups are grouped in different regions and altitudes in our project, each subsequent operation will be done for 3 coal ores (north, south and middle). The purpose of creating string files is to obtain a solid model by keeping the block modeling within a certain limit. After the string files, the upper and lower layers of the coal ore were formed as seams, and the solid between these two layers was modeled. In Figure 18 refers to the some photos of the ore deposit from the top and the side.

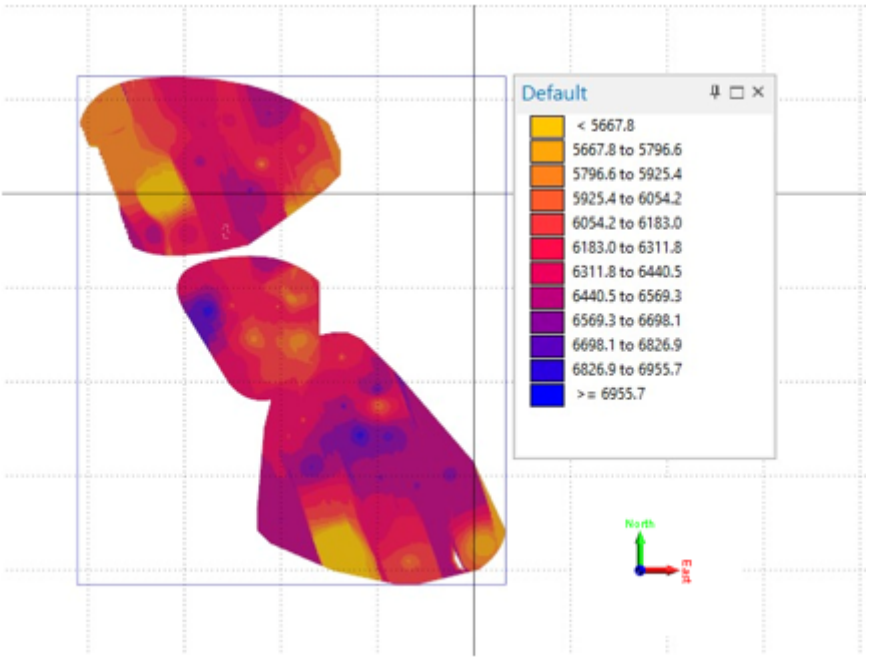


Figure 18. Top View of Coal Seams

The total resource of three seams is 81,846,858 m^3. The density of the coal is 1.3 tonne/m^3. The total resource is equals to 106,400,915 tonnes.

Block sizes are taken as 10x10x10 because very small block lengths take a long time on the software and in some cases give errors. In addition, these block sizes are quite sufficient in coal ore.

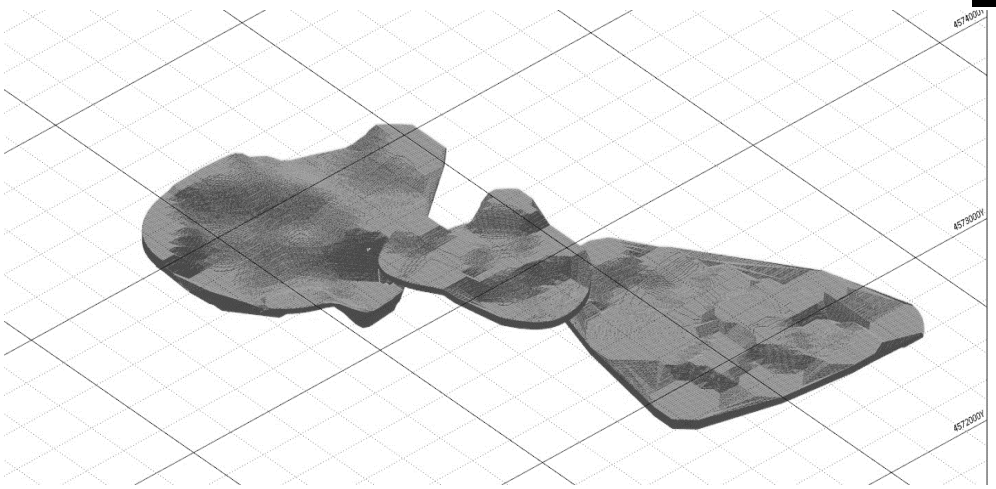


Figure 19. Block Mode of Seams

As explained above, the ellipsoid obtained by geo-statistical study was developed at this stage for an accurate calorific value assignment. Variograms are used to discover the isotropic behaviour of data and to explain the impact area of drill holes. The lengths of the major and minor axes, as well as their directions, were calculated using the variograms in order to achieve an optimal search ellipsoid for the existing drill hole assay values and the resulting block model, so that the variograms along those directions would best reflect assay value variance.

After passing solid modelling and blank modelling, the necessary procedure for assigning grade to the blocks was followed on Micromine. Compositing data, model files, seam block and search ellipsoid are used. Both kriging and IDW methods were used in this project. For the inverse distance weighting method, inverse power is set to 2 in the inverse distance control section in the modelling parameters section. For the element area, the calorific value was chosen. Since there is no block without calorific assignment, the calculation has been completed and the result has been reached.

The amount of resource and reserve is listed in Table 11.

Table 11.Resource ans Reserve Amount

	Tonnage (tonne)	Volume (m^3)
Resource	106,400,915	81,846,858

Reserve	14,966,917	11,513,013

Range of Calorie is between 5000 and 7457 kcal.

4.4. Determination of Reserve Evaluation and Operation Cost Factors

To optimize the mine using Micromine software, first detect some inputs such as dilution percentage, recovery percentage, selling cost, price, and some administrative costs. In Table 12, operating cost factors are listed.

Table 12. Operating Cost Factors

Parameters	**Value**
Mining Cost $/tonne	15
Waste Extraction Cost $/tonne	30
Dilution %	10
Rehabilitation Cost $/tonne	0.1
Price $/tonne	155
Selling Cost $/tonne	5
General & Administration Cost $/tonne	2

4.5. Cut-off Calorific Value Estimation

Because the primary goal of this project is to supply hard coal to iron plant industry in the mining area, the minimum required calorific value is 3230 kcal, as shown in Mine Life, Yearly Production, and Demand. In the mine area, extracted coal has minimum 5000 kcal. Therefore, there are not any problem of cut-off grade value.

4.6. Underground versus Surface Mine Selection

There are multiple factors for the design of a mine as underground or above ground. However, the most obvious ones that are actively used today are the comparison of the stripping ratio and the maximum allowable stripping ratio. The amount of stripping to be taken from the surface, the amount of the mine underground, costs and slope angles are important. Since the amount of stripping to be taken will change according to the selected angle, the design is one of the main determining factors. Thanks to the information obtained and compared from different angles by Micromine, it can be decided whether the mine is underground or above ground.

$$Stripping\ Ratio = \frac{Overburden\ or\ Waste\ (m^3)}{Ore\ (kg)}$$

Units may vary within different companies or designs, but the most prominent units are as in the formula. What this literally means is the amount of stripping that needs to be done to get one tonne of ore.

Maximum stripping ratio, also known as break-even stripping ratio, is the maximum overburden that can be taken for an economical pit limit. It is an important calculation for the ore and mining economy.

$$MASR = \frac{(UG\ Mining\ Cost(tonne\ of\ ore) - Surface\ Mining\ Cost\ (tonne\ of\ ore))}{Stripping\ Cost\ (m^3\ waste)}$$

As a result of the calculations, if the total cost is lower than the underground cost, open pit is preferred, and if it is higher, underground mining is preferred.

The stripping ratios of different open cast models are listed Table 13 below.

Table 13. Stripping Ratio for Different Open Cast Models

Overall Slope Angle	Tonnage of Ore (tonne)	Volume of Overburden (m^3)	Stripping Ratio (m^3/t)	Total Volume (m^3)
10°	106,400,915	129703789386	985:1	129835442228
20°	106,400,915	49010955774	372:1	49142608616
30°	106,400,915	29483681746	224:1	29615334588
40°	106,400,915	21896597495	166:1	22028250337

The plot of overall slope angle vs stripping ratio is shown in Figure 20

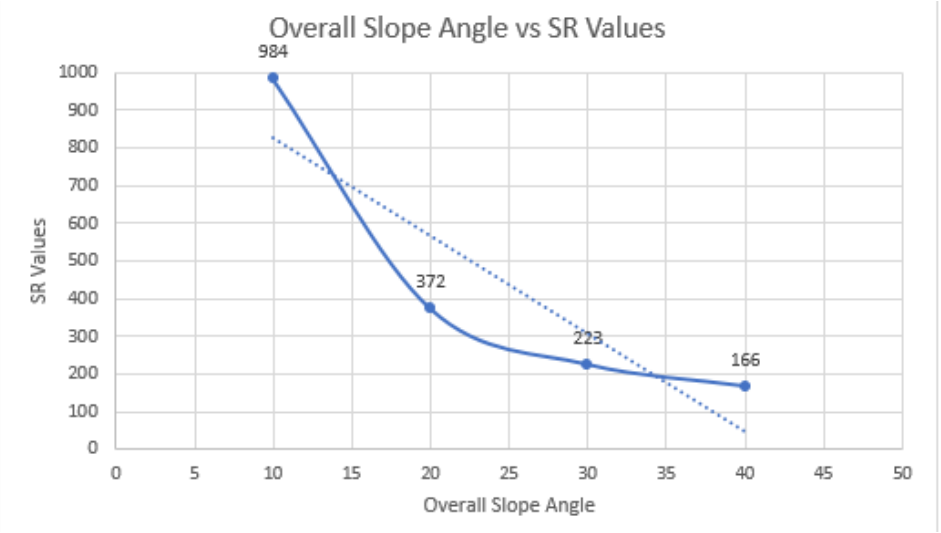


Figure 20.Overall Slope Angle vs SR Values

The calculations are shown in the Appendix A.

4.7. Mine Life, Yearly Production and Demand

In the calculations based on Micromine data and the sample drills, stability, faults the reserve amount is 14.9 million tonnes. To calculate the mine life Taylor's Mine Life Rule is used.

The calculations shown in the Appendix B.

The mine life is decided as 13 years. Assuming that the mining site works 330 days a year, it is designed with a production capacity of 1.15 million tonnes of coal per year and an average production capacity of 3,500 thousand tonnes per day.

5. MINING METHOD SELECTION

Mining method selection is the main process for life of mine. There are many parameters to make them maximize such as profit of company, recovery of the resources, conditions of the environment. At the beginning of all steps, after the data calculations, located of mineral resources, proved mineral quantities mining engineers decided for the best mineral extraction method. In the past, there are a lot of mining selection technique was found and attempted.

5.1. Room and Pillar

The logic of room and pillars, which is one of the most common methods used in underground mines, is to open rooms in parallel and in certain arrangements to extract the mine and to leave behind generally rectangular shaped pillars which made of ore. Pillars are left to support the overlying rock. Sometimes, after the workings on the seam is finished, these pillars can be taken out of the mine and this process called as “retreat mining” in which the roof of the mine will collapse.

The room and pillar method is applicable to the flat coal seams with average thickness, located at shallow depths. Coal seams with a thickness of up to 3 m are the most applicable thickness of this method. On the other hand, the thickness of the coal can stretch up to a maximum of 5 m, but it can cause prevent the efficient implementation of this method. In this project, the thickness of the coal seam is reached to 25 m. This is proof that the project cannot be implemented with this method.

In the underground deep coal mining, can be used two mining method for the conditions and the properties and dangerous situations of coal. The first is partial extraction without caving of the roof and the strata pressures are distributed uniformly, vertical mining is more effective and remain pillars in the mining area. This method called as ‘Room and Pillar’. The other method is generally preferable method ‘Longwall Mining Method’. After the caving of the roof strata behind the face, transfer the strata coals by the automatic systems.

Applicability Conditions for Room and Pillar Coal Mining

	Condition	Situation
Deposit Dip	**Max. 30° Inclination**	✓
Thickness of Seam	**Not Thick (low than 5m)**	X
Deposit Shape	**Tabular**	✓
Deposit Size	**Large areal extent**	✓
Coal Strength	**Comparatively Strong**	✓
Deposit Uniformity	**Fairly Uniform**	✓
Depth	**Shallow to moderate**	X
Strength of the Roof	**Medium to Strong**	✓

Generally, %40 - %50 coal reserve may be taken by room and pillar method if pillars are not extracted after the production stage. When the pillars are recovered additional %20 - %30 reserve can be produced as extra.

5.2. Longwall Mining Method

The other method is Longwall mining is that mostly common method for coal production. This method applied generally in flat, thin, tabular deposit panels.

The machines cannot cut whole height of the coal seam, so controllable caving method uses in this type of mining. After the cutting process, thanks to hydraulic supports, cave behind the roof. The caved area called as GOB or GOAF. Also, after the cutting process, filling with the waste to cut areas provides mining support and also safety work places for all the workers and machineries.

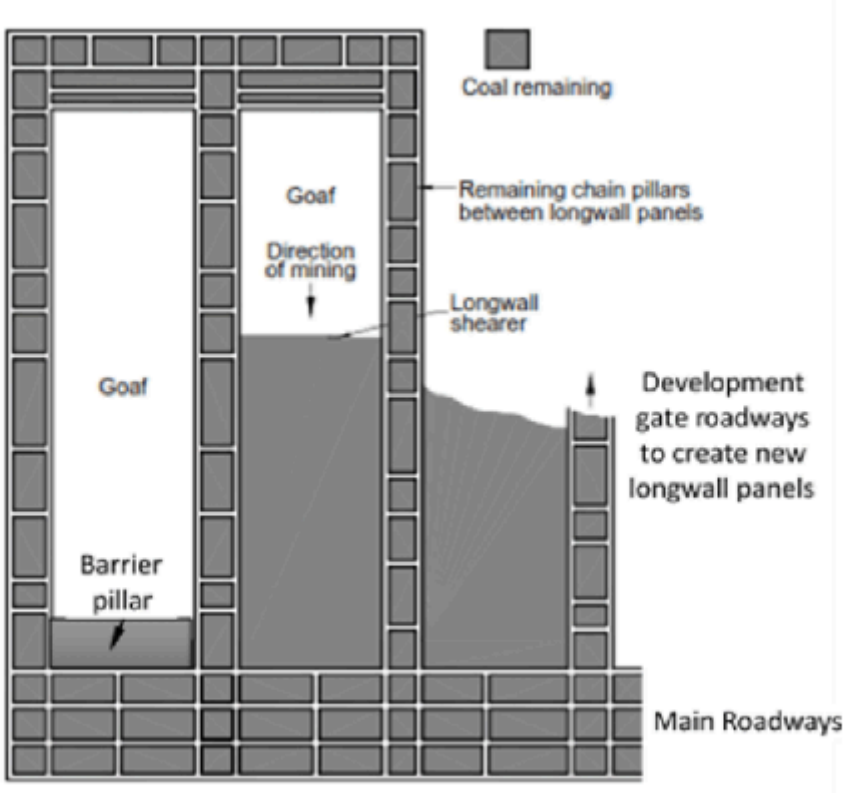


Figure 21.Retreat Longwall Mining Method (Öztürk, 2018)

Applicability conditions for Longwall Mining Method

	Condition	Situation
Deposit Dip	**Max. 15° Inclination**	✓
Deposit Thickness	**Single-lift up to 4.5m, multi 9m**	✓
Deposit Shape	**Tabular**	✓
Deposit Size	**Large areal extent**	✓
Ore Strength	**Preferable weak**	✓
Rock Strength	**Immediate roof cave easily, main roof weak to moderate, floor roof firm**	✓
Depth	**Usually moderate (200-1100m)**	✓

In addition to these suitable situations, longwall mining can preferable for highest underground productivity not to left pillars, thanks to mechanization low mining cost, continuous production process, low labour requirement and remote control, recovery about %75-95, low dilution <%10, applicability for deep seams under bad roof conditions and also highest safety for supports and filling with waste to the cut areas.

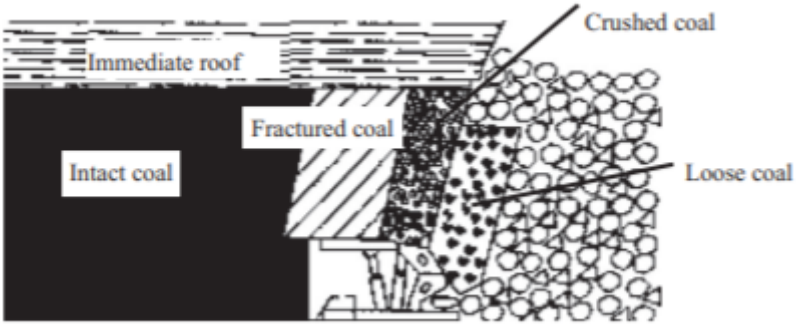


Figure 22.Top Coal Caving Method (Wang, Yang, Li, Wei, & Liu, 2014)

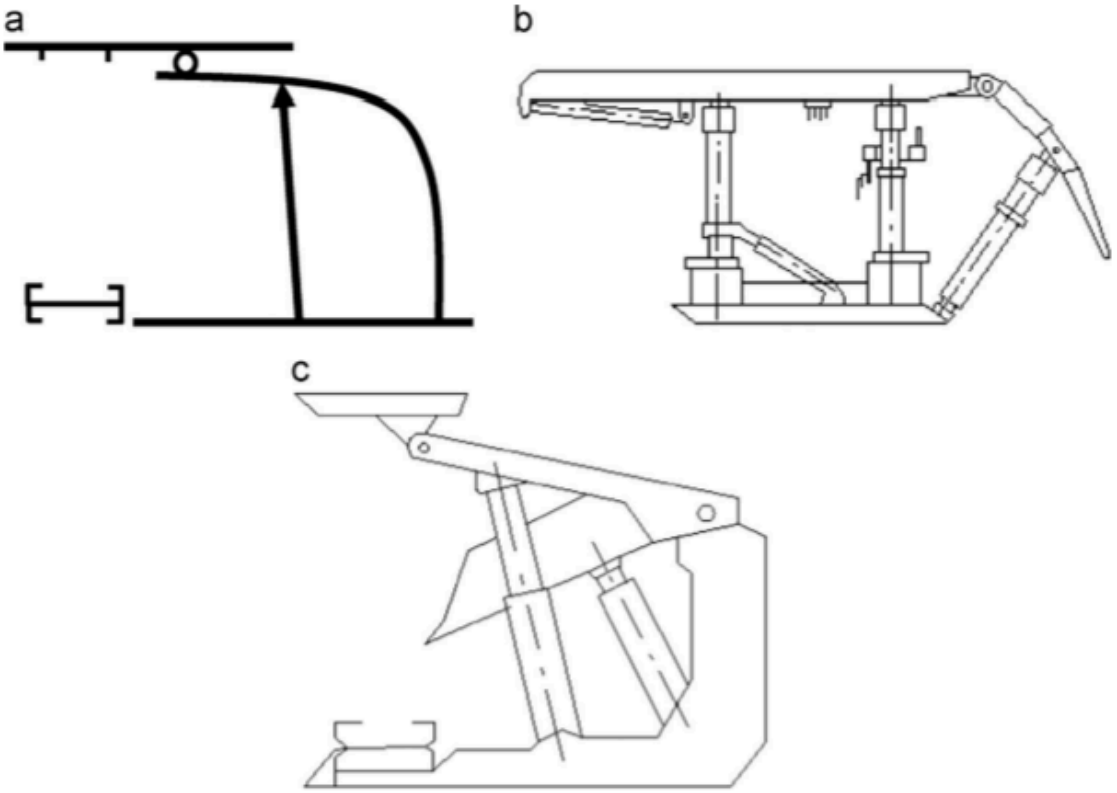


Figure 23.Various top coal caving method hydraulic supports (Wang, Yang, Li, Wei, & Liu, 2014)

In Longwall mining, in the removal of coal seams larger than 4.5 m, due to both stability and technological costs, the production face cannot be taken beyond a certain height.

At this stage, the ceiling supported by hydraulic fortifications is collapsed in a controlled manner and maximum efficiency is tried to be obtained from the thick veins. In general, the efficiency of coal mining is low. Compared to the room and pillar method, the top coal caving method is higher in efficiency.

Longwall top coal caving (LTCC) is a common coal mining method generally used by China. However, it is an extra method used in Australia, Russia, and Turkey. Generally, it is divided into two as Bottom coal seam and Top coal seam. The bottom coal seam is the lower part of the shearer cut and the coals cut by the Armored Face Conveyor are carried. After mining in the Undercut, capillary cracks form in the Top part and the fortifications are poured from the back side by the force of gravity. No extra mining method or technique is applied. In some cases, empty holes are made up from the undercut area as an extra. This is decided according to the durability of the coal.

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Techniques of Longwall Mining

1-) Advance Longwall Mining (Face away from roadway)

2-) Retreat Longwall Mining (Face to the roadway)

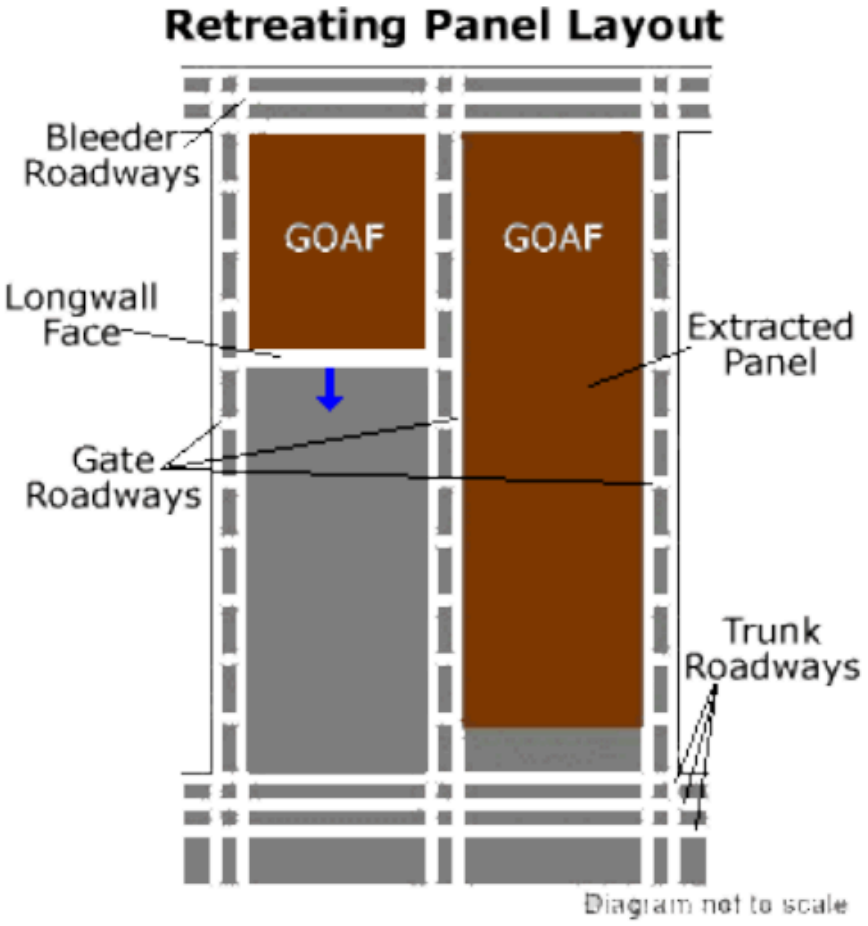


Figure 24. Retreating Panel Layout (Öztürk, 2018)

In these two different techniques, there is a difference in the propagation directions of the face taken. While the sedimentation process is carried out by moving away from the main road in Advance Longwall, in the Retreat Longwall technique, a face is formed from the tunnels opened from the sides and precipitated towards the main transport roads. Both methods have different advantages. Retreat method will be applied in our mining application.

Since the Top Coal Caving method will be applied, taking the coal in the deposition area, and leaving it behind reduces the risk and ensures a healthier mining application. It is a more preferred method especially in applications without metal ores. The risk of spontaneous

combustion is significantly reduced as the production area is abandoned. The face, which is produced as an extra, can be abandoned quickly, and mining can continue faster and more regularly. This provides higher efficiency and more time savings.

5.3. Interpreting Other Applicable Method

To decide the method of coal extraction, we need to interpret the Shortwall Mining method. The shortwall mining is comparable to longwall mining, but with shorter face lengths ranging from 40 to 90m, with the goal of managing the caving character of the overlying higher strata, the strain on support, and the general operation of the supports used at the face.

When we evaluate the information, we have and the applicability of the methods, the method that suits the characteristics of the mining area we want to apply and the high recovery it provides and at the same time the low dilution rate is the Longwall Mining Method with Top Coal Caving and Retreat techniques. In terms of worker safety and health, it is the most logical underground coal extraction technique that can be chosen due to the advantage of working with minimum personnel, the advantage of remote control and its applicability to thick veins.

6. STABILITY STUDIES

Determining Panel and Pillar Dimensions

Single panel height and width is decided by opening a panel from the long side of the coal seam by using Rocscience RS2 software. RS2 is a 2D Geotechnical Finite Element

Analysis software provided by Rocscience. Rock mass properties are taken from RocData software. Numeric modelling is made according to the largest coal seam, based on the average heights of various rocks such as limestone, shale, marl, clay, and conglomerate in the project area. Colour legend is shown at Figure 26**Error! Reference source not found.**. Coal seam has about 25 m thickness and 1100 m width. This seam is located about 1000 m below the ground. Main model is shown at Figure 36. Also, Figure 37 shows the colour legend of the materials that used in the numeric model. The ideal panel size is achieved by trying panels of different widths and heights. Numeric modelling has started with 200 m width and 4 m height. Also, hydraulic support of the shearer is taken as 1 MPa. Figure 27 shows the displacement and Figure 38 shows the major principal strain of this model.

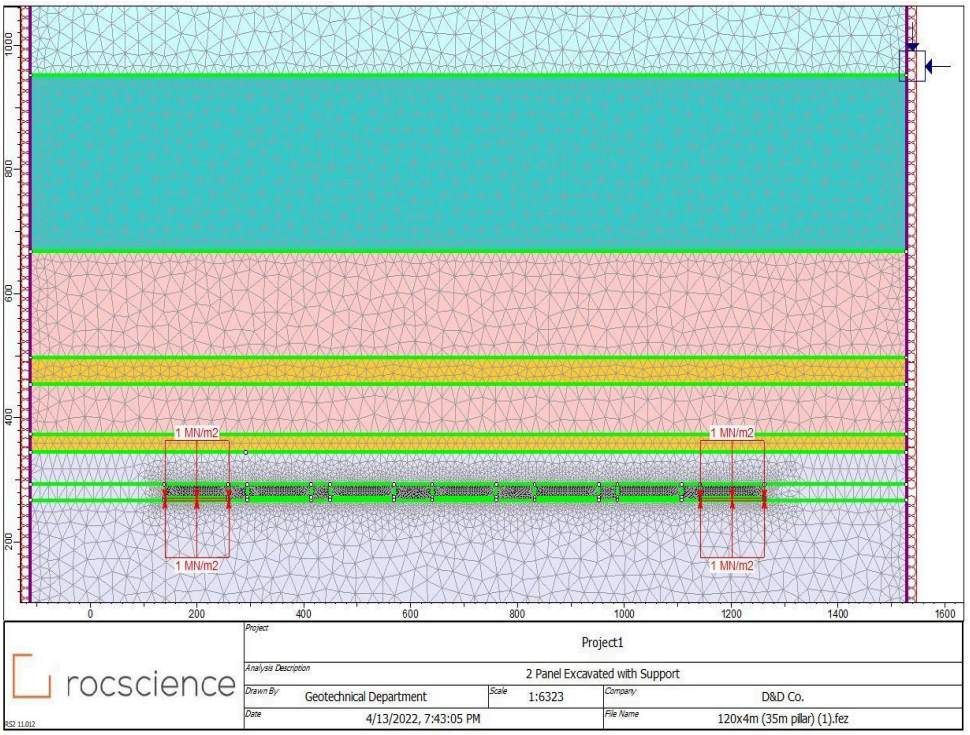


Figure 25. Main Model

Material Name	Material Color
Limestone Plastic (Deep)	
Coal Plastic	
Limestone Shallow	
Shale Plastic	
Marl Plastic	
Clay Plastic	
Conglomerate Plastic	
Limestone-Marl Plastic	
Limestone Elastic (Deep)	
Coal Elastic	
Limestone Shallow Elastic	
Shale Elastic	
Marl Elastic	
Clay Elastic	
Conglomerate Elastic	
Limestone-Marl Elastic	

Figure 26. Colour Legend of Materials

6.1. Determining Pillar and Panel Dimensions

According to Figure 2, the largest displacement is 1.5 m. The mining method is Longwall Top Coal Caving method (LTCC). Where the panel height is 4 meter and thickness of seam is about 25 m, the left part of seam is extracted by caving. To cave the material, there should be displacement. However, if this displacement is large, it can be difficult to control and can lead to disaster. It may be difficult to control the 1.5-meter displacement, but if other conditions are met, it may be considered to work at this displacement. In this type of coal mines, major principal strain is expected to be at most 7%. However, Figure 28 shows that maximum principal strain is about 12%. Also, this principal strain concentrates on the abutment points. This may cause the rock burst, so it is not feasible to work with these conditions.

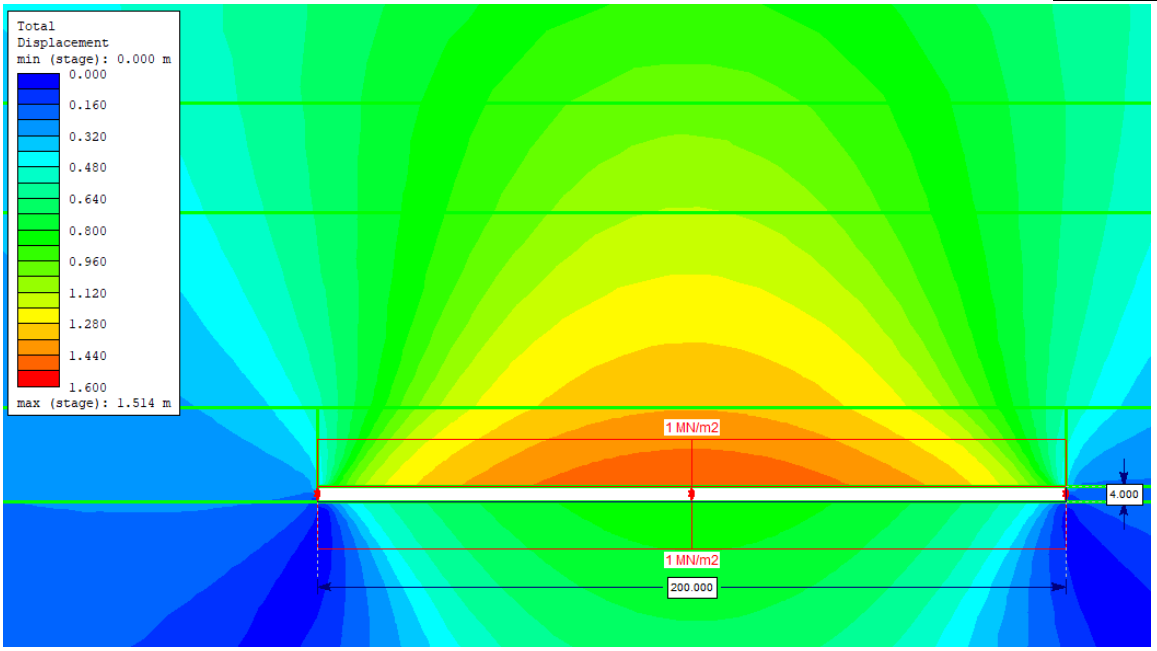


Figure 27.Total Displacement of 200 m Wide Panel in Plastic Model

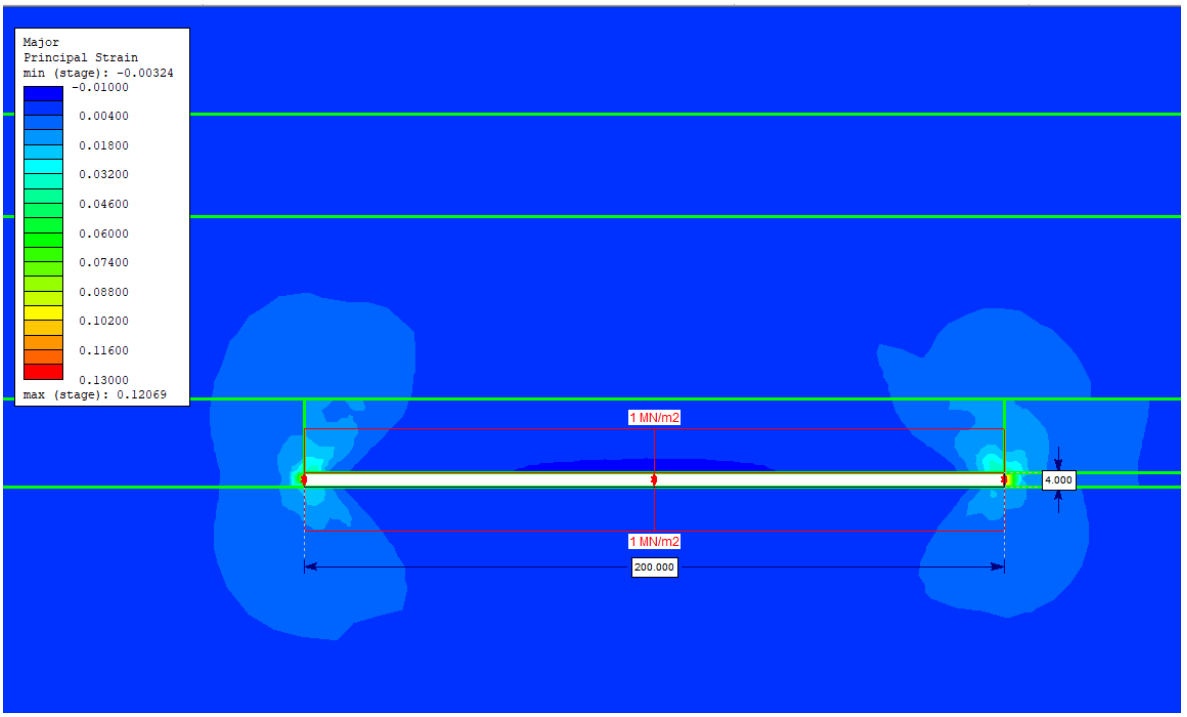


Figure 28.Major Principal Strain of 200 m Wide Panel in Plastic Model

After trying different types of panel models, it is decided that to work with 120 m width and 4 m height. Figure 29 shows the major principal strain of this panel. According to the Figure 28, major principal strain is inacceptable level. Under these conditions the rock burst problem can be controlled by destress blasting when it is necessary.

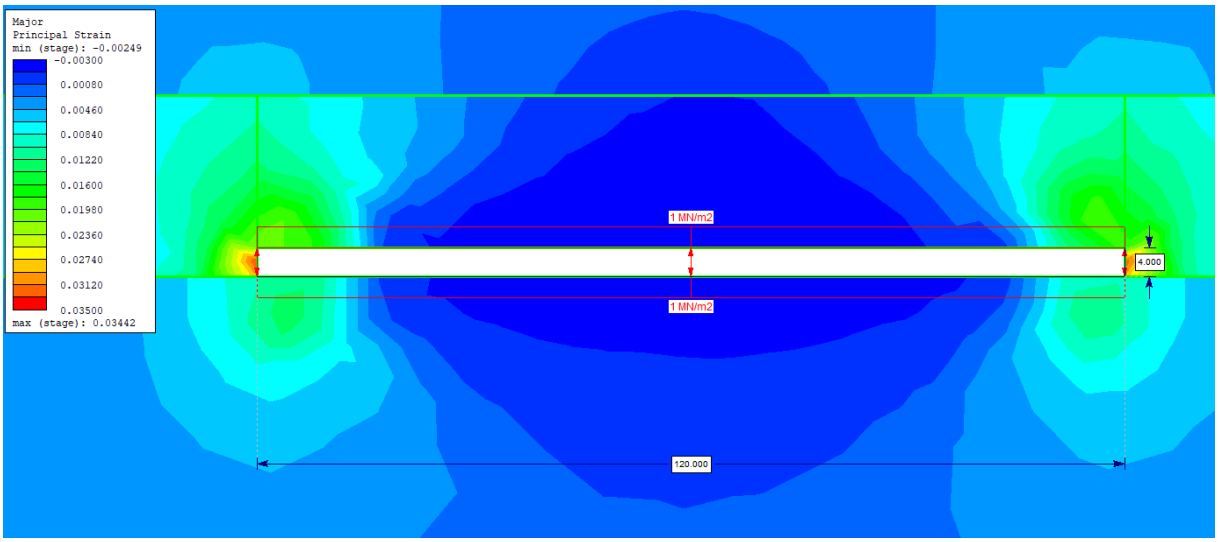


Figure 29.Major Principal Strain of 120 m Wide Panel in Plastic Model

After determining the panel width, the panels are placed in sequence in the coal seam to find the pillar width. Also, the excavated areas are filled in the software with caved loose material rock mass properties. Rock mass properties of loose material is taken as 20% of the coal properties. After the numeric models are designed, it is decided to start the excavation on both sides of the coal seam and move towards to the middle and dimensions of the pillars are determined. In order to get the maximum recovery from the middle panel with a stable structure, dimensions of the barrier pillar are determined. Figure 30, Figure 31,Figure 32, and Figure 33show the excavation planning respectively.

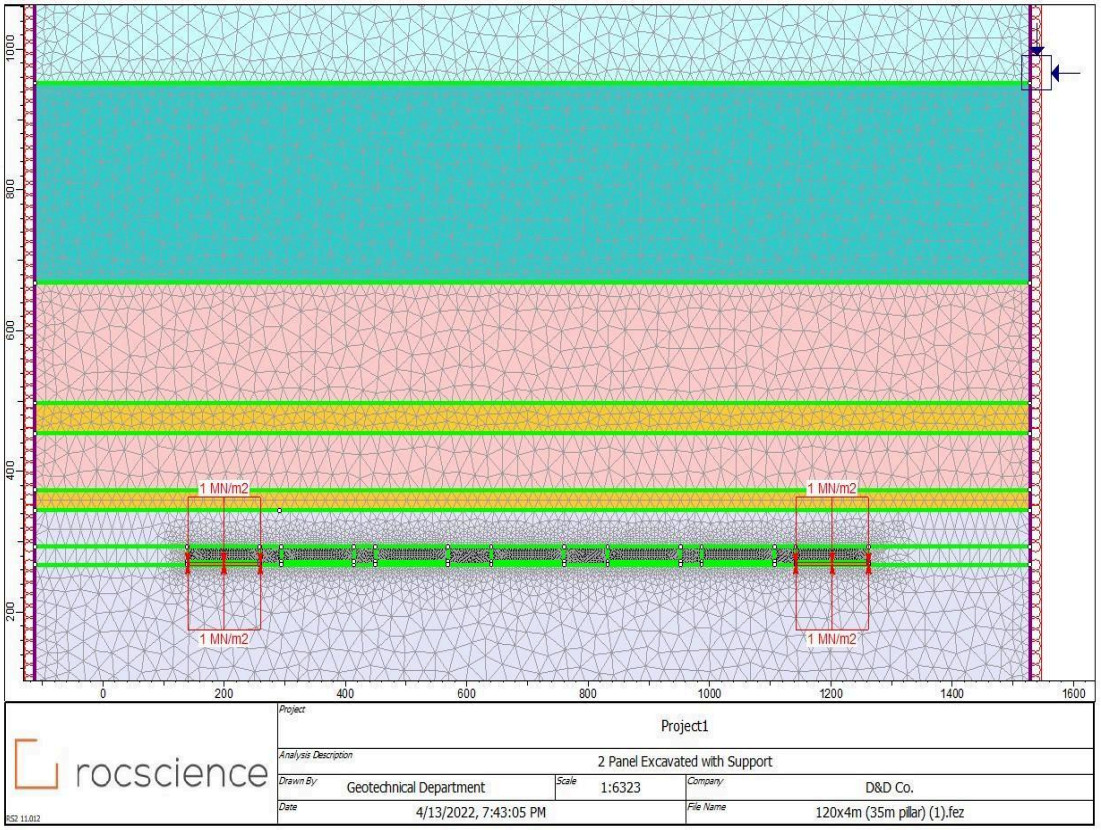


Figure 30.2 Panels are Excavated with Support in Plastic Model

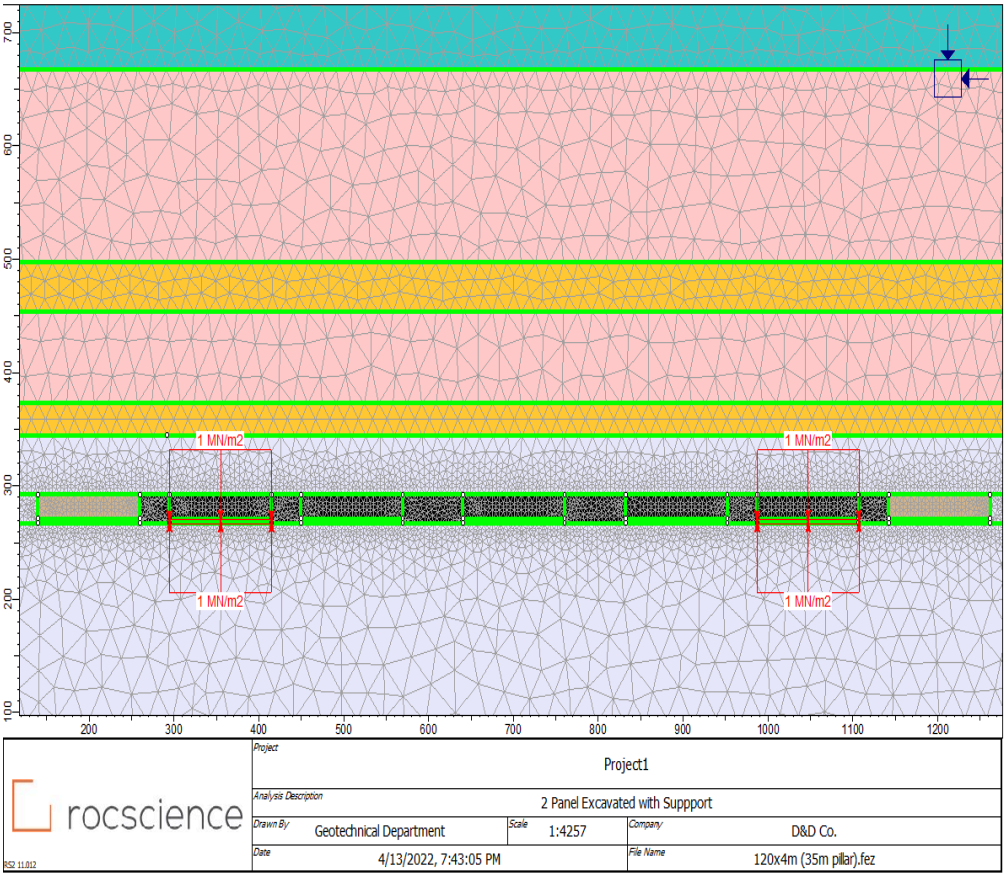


Figure 31.4 Panels are Excavated with Support in Plastic Model

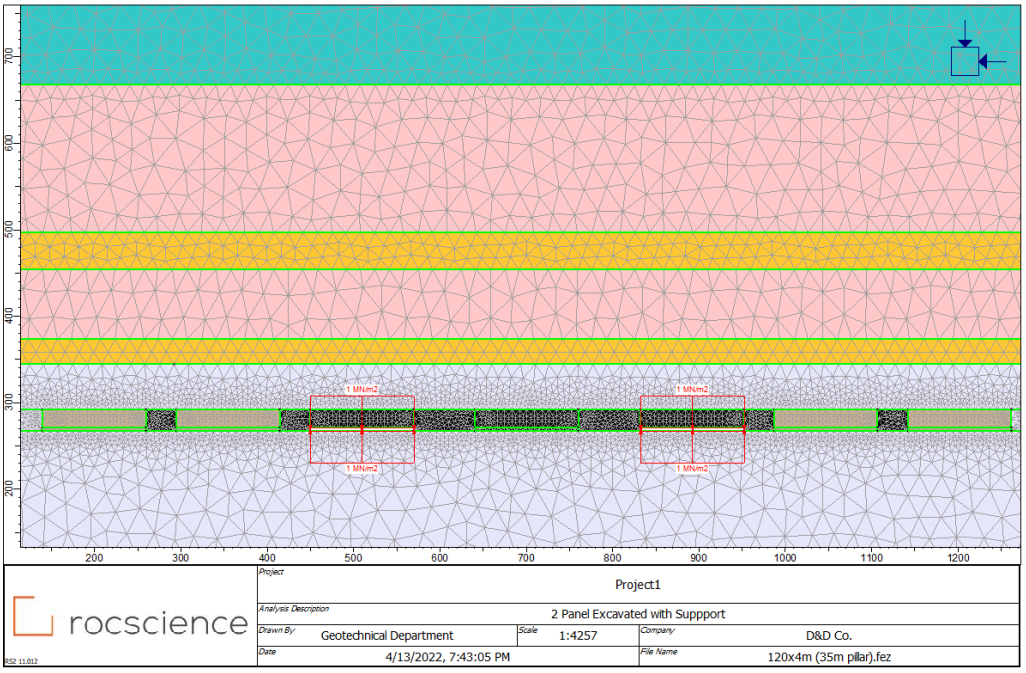


Figure 32.6 Panels are Excavated with Support in Plastic Model

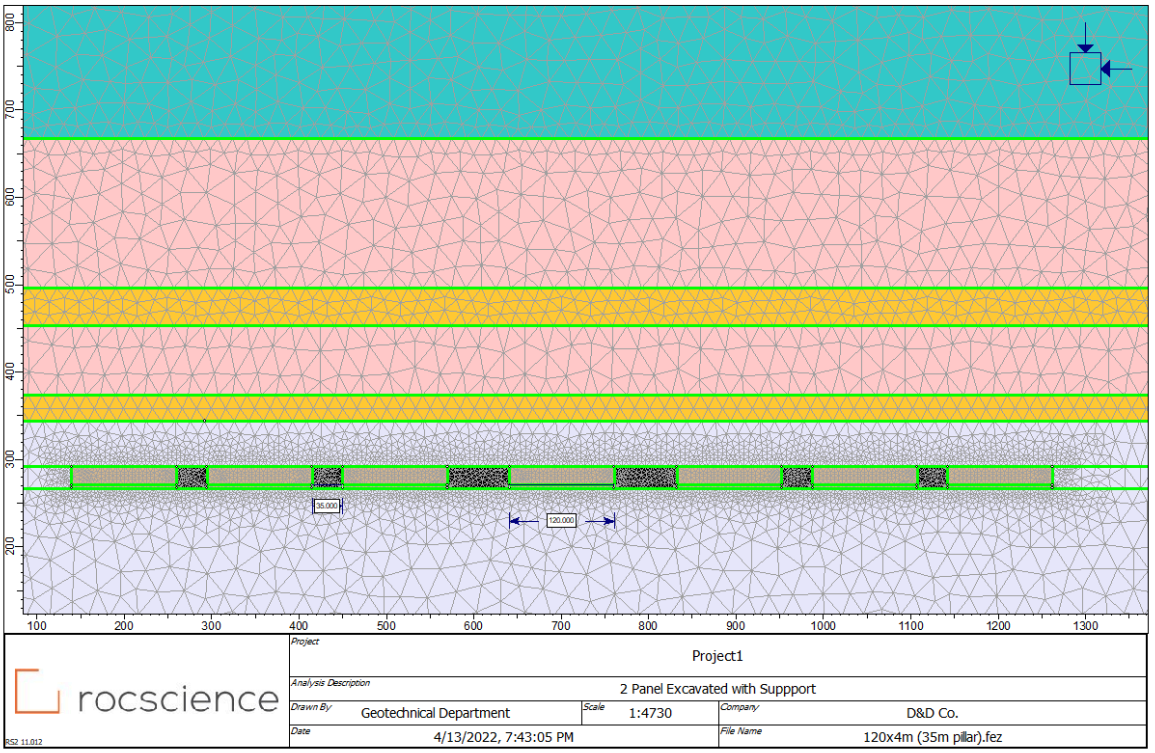


Figure 33.All Panels Are Excavated in Plastic Model

Total displacements and major principal strains are the main factors when deciding the pillar width. Major principal strains of each step can be seen in the Figure 34 Figure 35 Figure 36, and Figure 37. As mentioned before, major principal strain should not exceed 7% and according to Figure 36, in this model, maximum principal strain is inacceptable level on the abutment points.

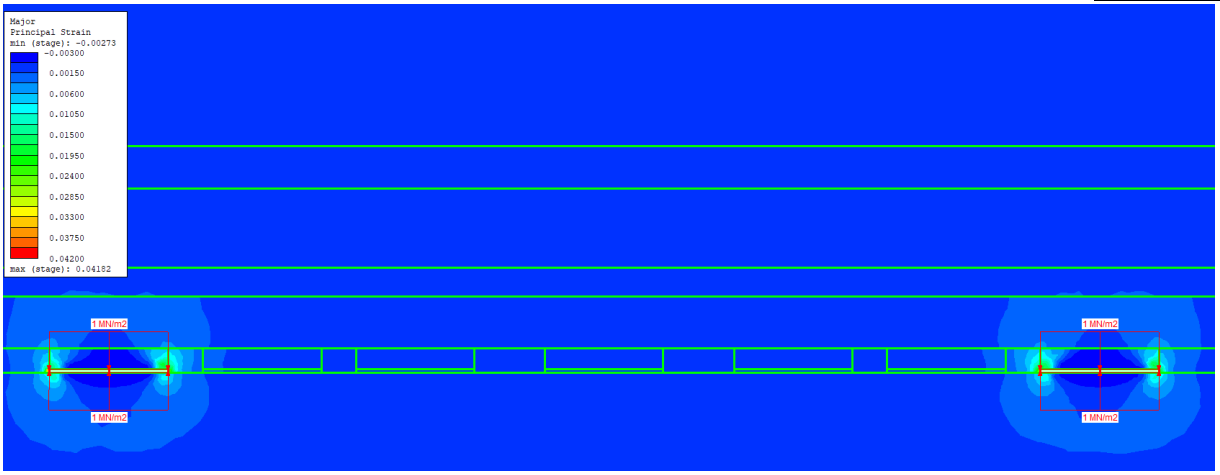


Figure 34.Major Principal Strain of 2 Panels Are Excavated in Plastic Model

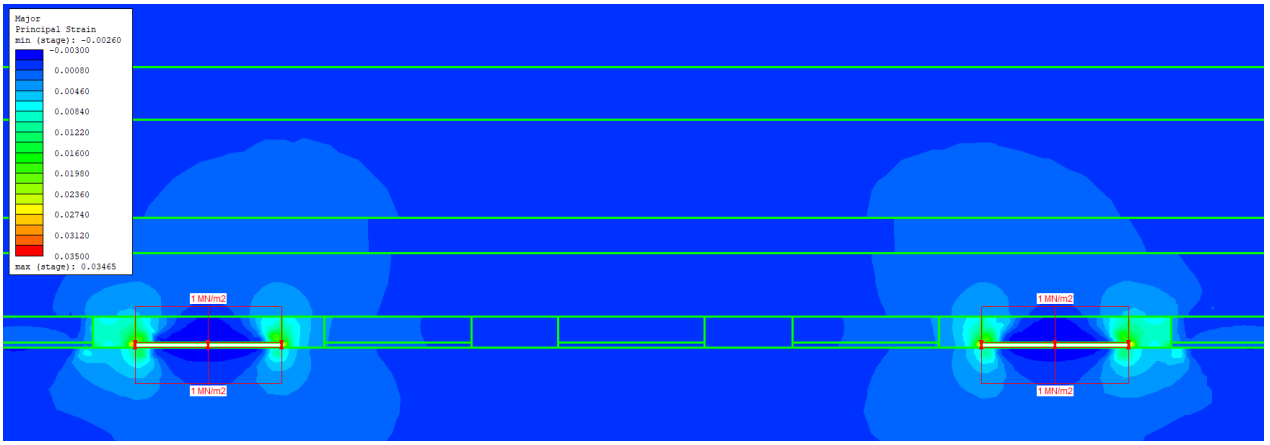


Figure 35.Major Principal Strain of 4 Panels Are Excavated in Plastic Model

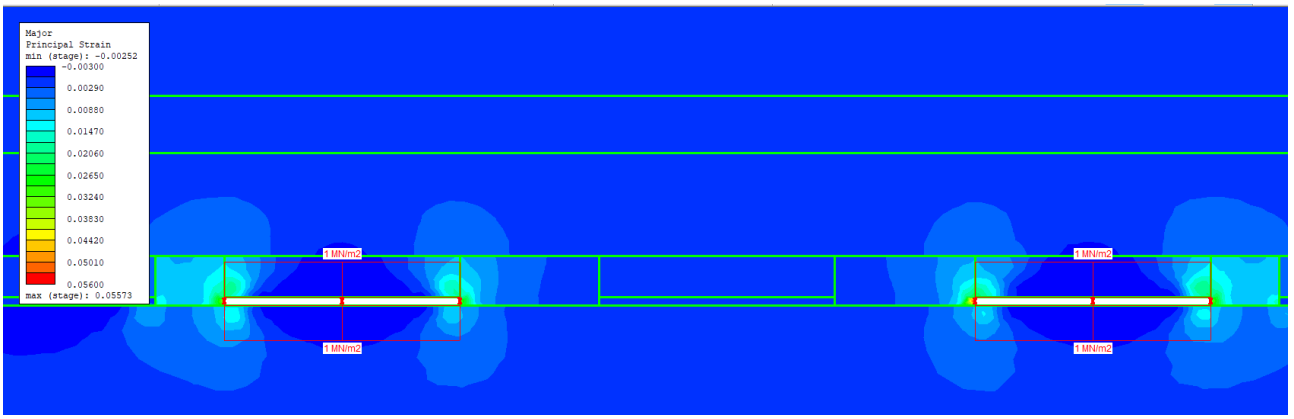


Figure 36.Major Principal Strain of 6 Panels Are Excavated in Plastic Model

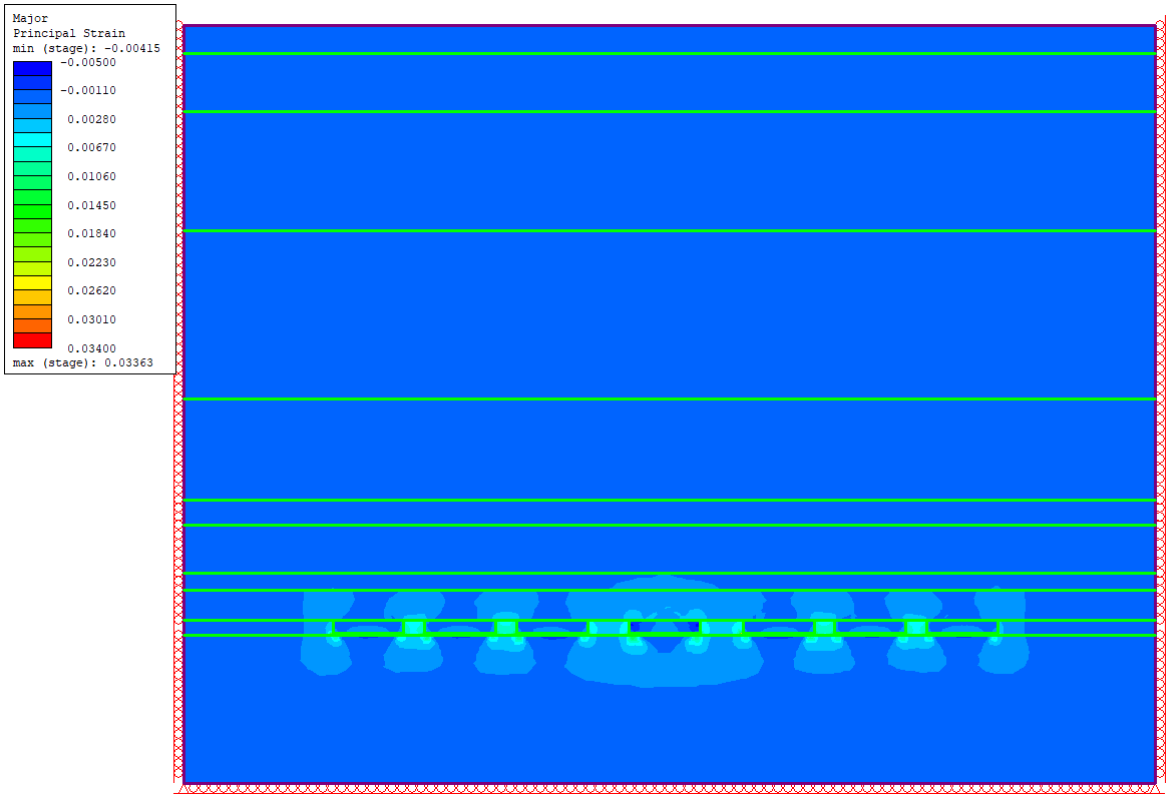


Figure 37.Major Principal Strain After All Panels Are Excavated in Plastic Model

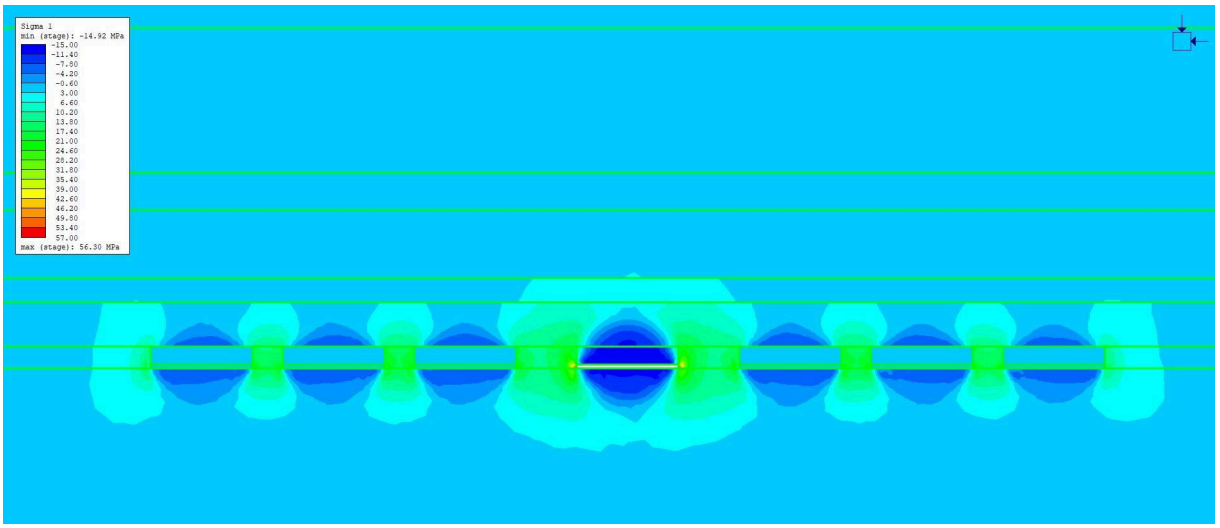


Figure 38.Major Principal Stresses After All Panels Are Excavated in Plastic Model

Figure 37 shows the major principal strain magnitude and distribution after all coal seam excavated and Figure 38 shows the major principal stresses. After excavation, excavated areas filled with caved loose material.

6.2. Surface Subsidence

A subsidence is expected in the underground LTCC mining method. However, it needs to be checked if this subsidence will damage or endanger the project area. When the total displacement after the excavation of the largest coal seam is analysed, it is seen that the maximum displacement on the surface is around 3,5 m. Also, the draw angle is observed to be around 30 degrees.

Furthermore, the development studies are carried out according to this case. Figure 39 shows the draw angle and the total displacement on the surface from cross section.

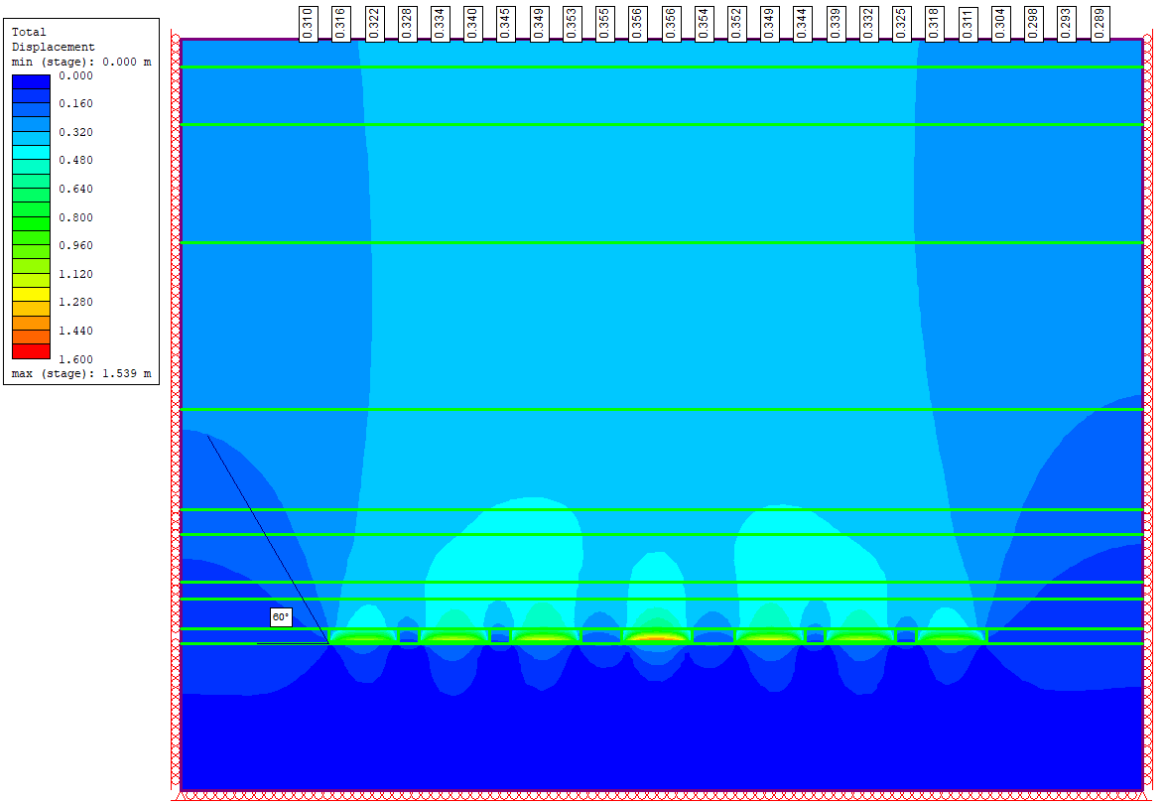


Figure 39.Total Displacement After All Panels Are Excavated in Plastic Model

The amount of the subsidence is calculated in Appendix C.

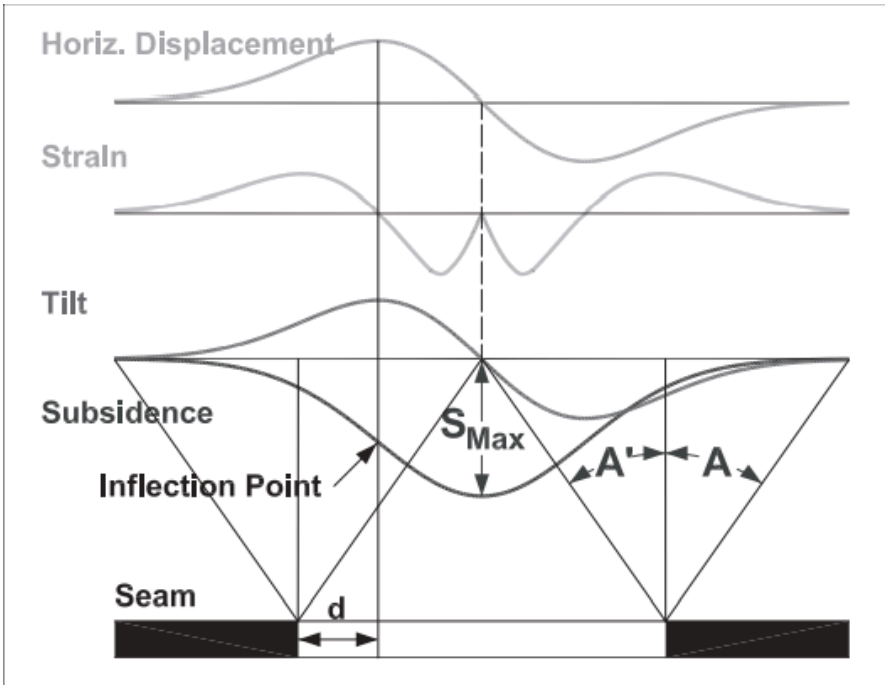


Figure 40. Subsidence for a Critical Panel Width (Bise, 2013)

Since maximum subsidence is not observed when the middle panel is mined, maximum subsidence will be observed when 1189 m of excavation is made from the lower panel.

6.3. Global Modelling

In this coal project, there are 3 different main coal seams. The 2 coal seams are located at approximately the same elevation, and the other one is located about 80 m above the others. When looking at the cross section of the model, it is seen that the upper seam is approximately has 850 m width, and the two lower veins are 1100 m and 1400 m, respectively. The main purpose of this modelling is to determine which seam to start from first is better. When a seam is excavated, it should not affect the others, or its effect should be minimized to obtain maximum stability and recovery. Figure 41 shows the global modelling of the project from side view.

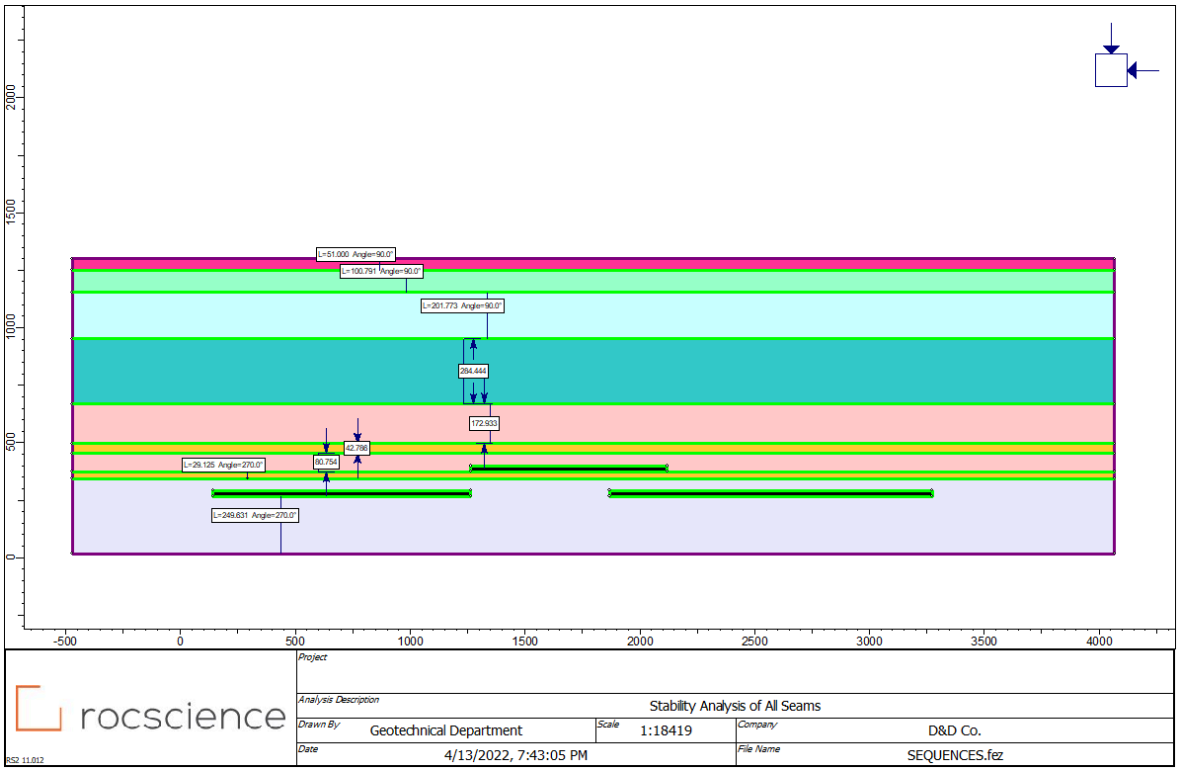


Figure 41.Global Seam Modelling

After the necessary models are made, it is seen that starting with one of the lower seams damaged the structure and the stability of the seam at the top. For this reason, it is seen that it would be better to excavate the upper seam first and then move on to the other two seams. In addition, after the upper seam is excavated, it is seen that approximately 250 m of the 400 m long seam in the lower right is affected by this excavation. Figure 42 shows how the area is affected after the excavation of the middle seam.

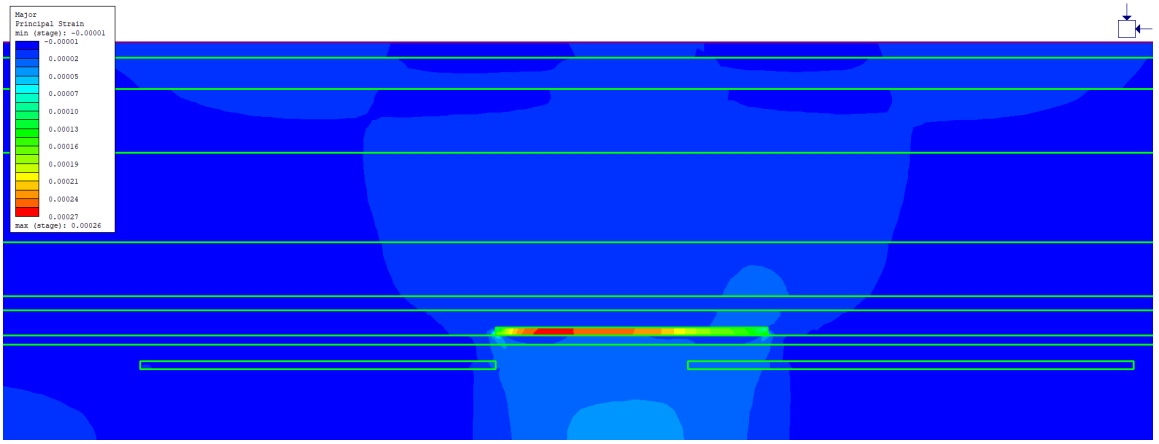


Figure 42.Major Principal Strains After the Middle Seam Is Excavated in Plastic Model

The major principal strain level in the global seam model after excavating the middle seam and then excavating the other seams is shown in Figure 43. According to this figure, the greatest major principal strain level is at 0.053%.

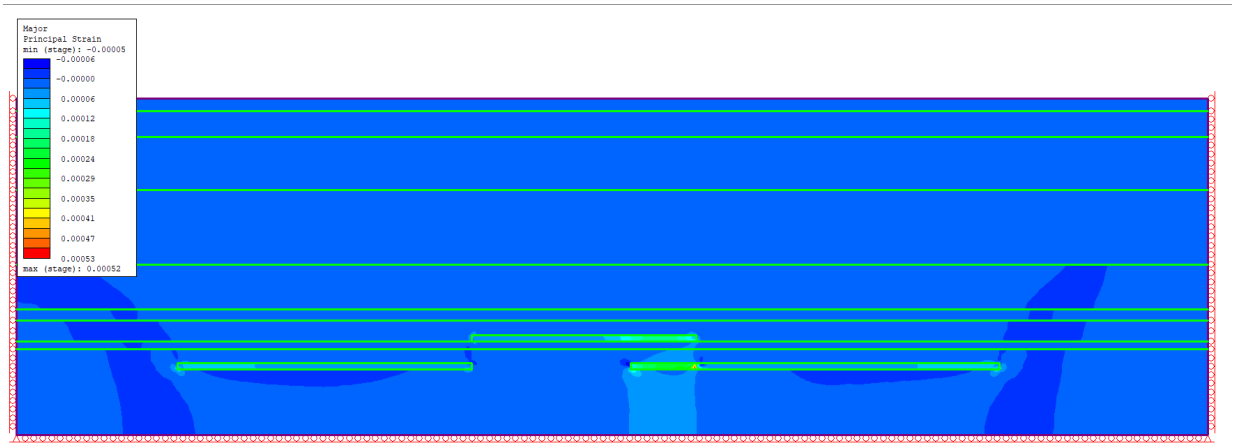


Figure 43.Major Principal Strains After All Seams Are Excavated in Plastic Model

6.4. Ramp Design

Firstly, stability study for the ramp carried out and Figure 44 shows the main model of the ramp with seams.

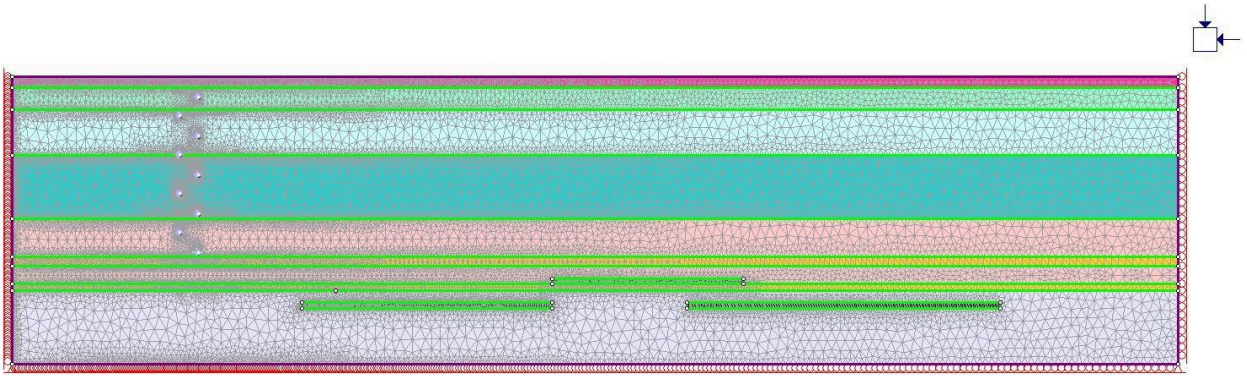


Figure 44 Main plastic model of the ramp with seams.

The most challenging part of the ramp in terms of rock stability comes out as the ramp get closer to the seam. Without any support it is not safe to drive a ramp.

Figure 45 Total displacement of the ramp without any support in plastic modelFigure 45 shows the total displacement on the ramp without any support in the part closest to the seam. Also, Figure 46 shows the major principal strain.

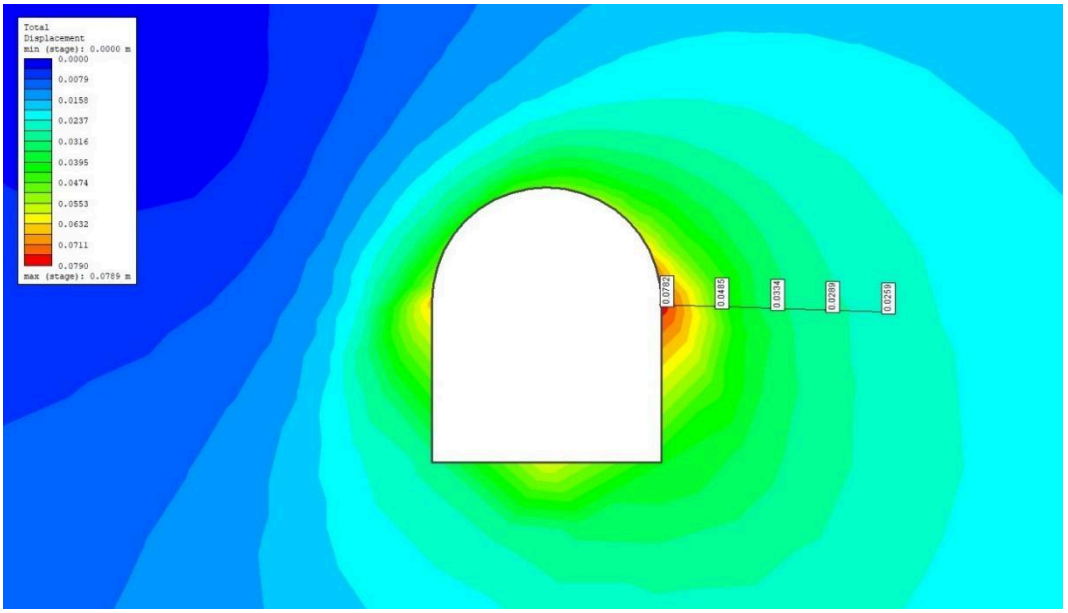


Figure 45 Total displacement of the ramp without any support in plastic model

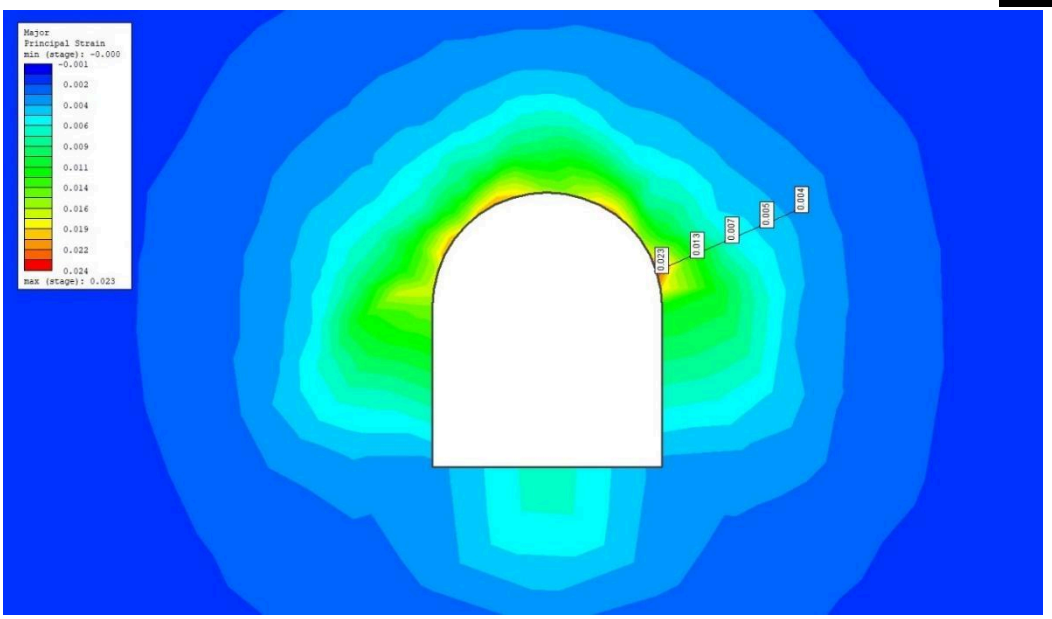


Figure 46. Major principal strain of the ramp without any support in plastic model

In order to satisfy the safe conditions in the ramp, supports are needed. For this reason, necessary bolts and liners are added into ramp. Figure 47 Total displacement of the ramp with support in plastic modelshows the total displacement on the ramp with support in the part closest to the seam. Also, Figure 48 Major principal strain of the ramp with support in plastic modelshows the major principal strain.

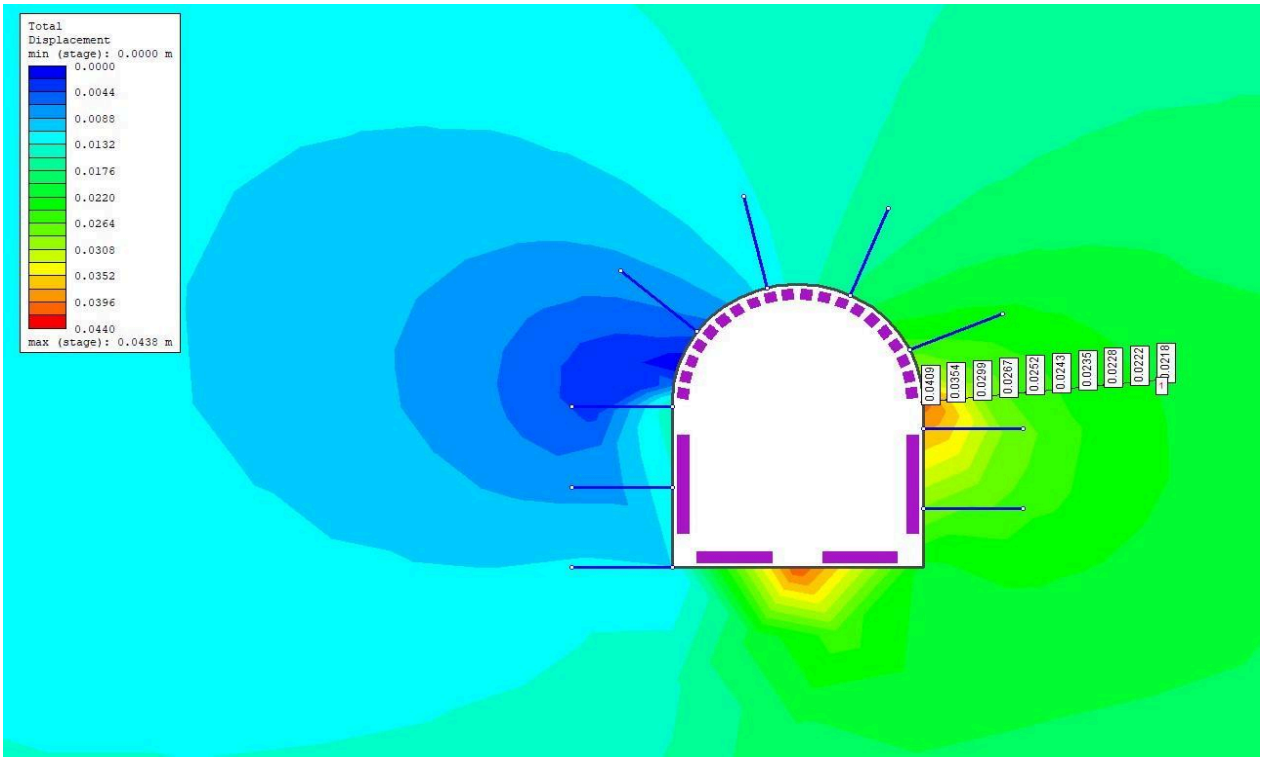


Figure 47 Total displacement of the ramp with support in plastic model

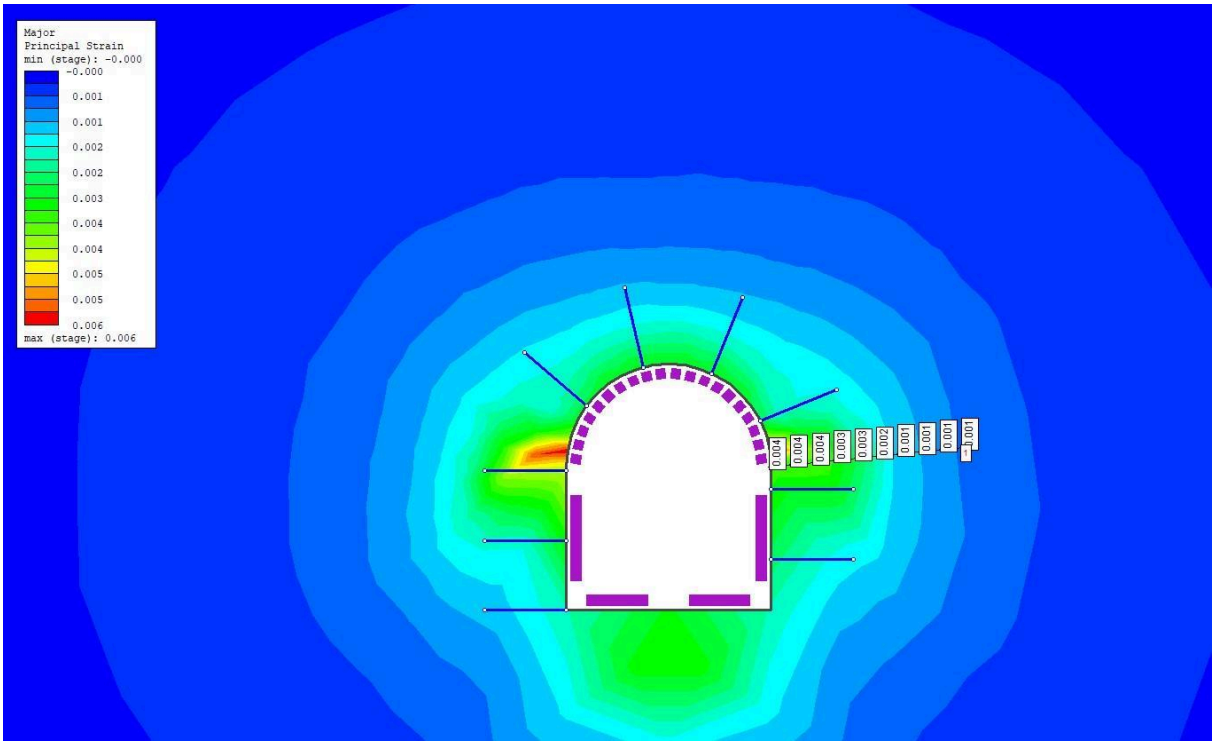


Figure 48 Major principal strain of the ramp with support in plastic model

6.5. Shaft Design

After the ramp design, shaft design studies are carried out for different rock types Figure 49 shows the major principal stresses on the shaft and Figure 50 shows the total displacement on the shaft. According to these studies, shaft drilling is safe in terms of rock stability.

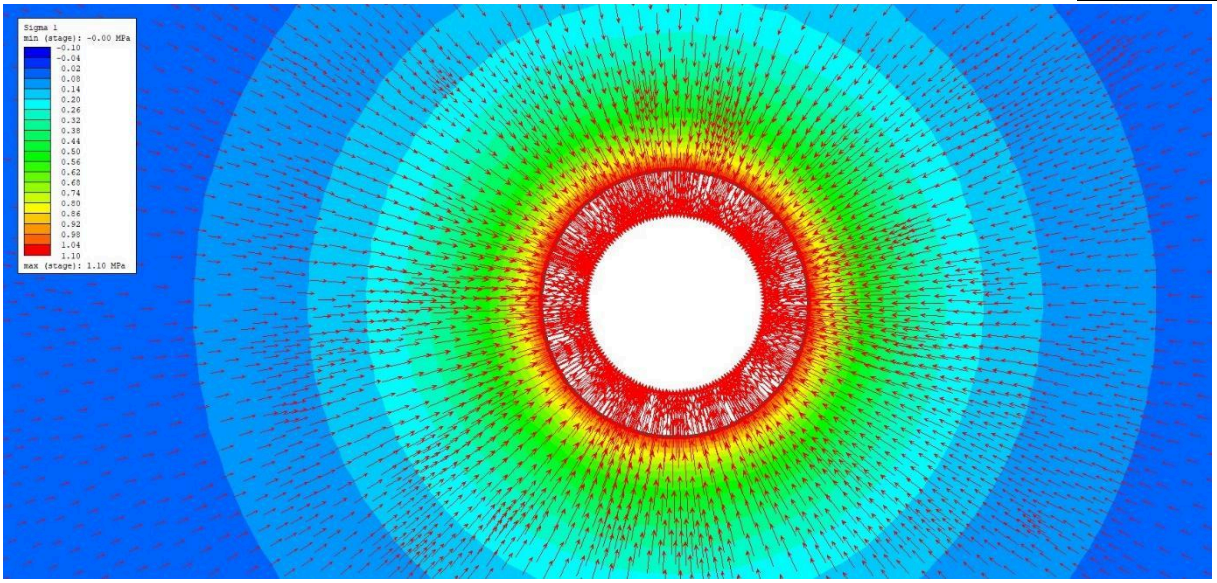


Figure 49 major principal stresses on the shaft in plastic model

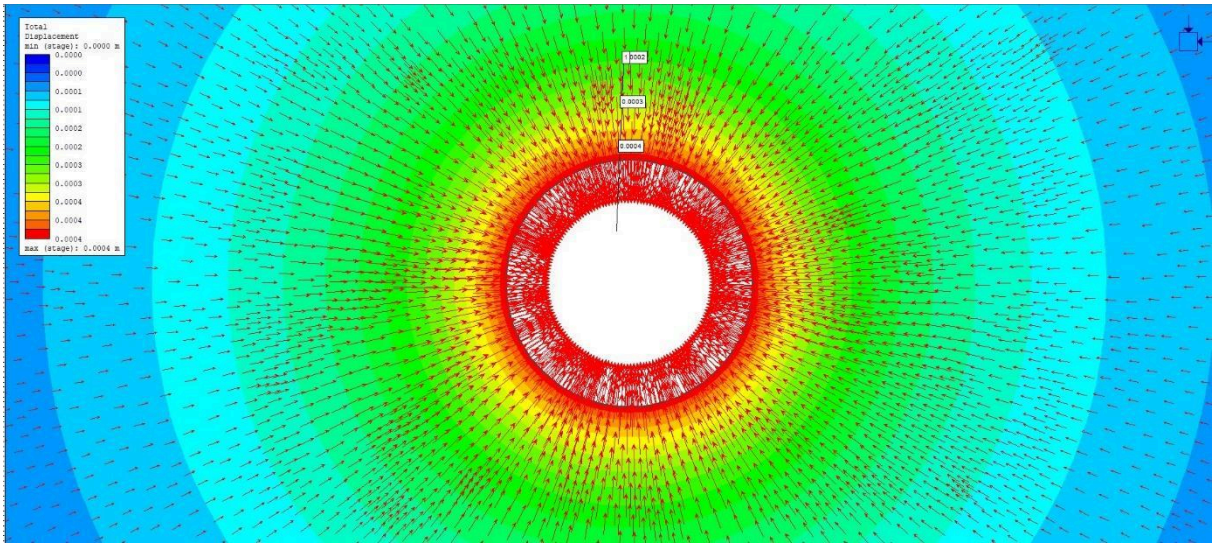


Figure 50 Total displacement on the shaft in plastic model

6.6. Waste Dump Design

According to this project, waste material will come out during the development work such as ramp, shaft and panel opening. Therefore, the waste dump design with the required capacity is made by using RS2 and Slide 2 software. Firstly, by using RS2, stability of the waste dump examined. Figure 51 shows the main plastic model of the waste dump.

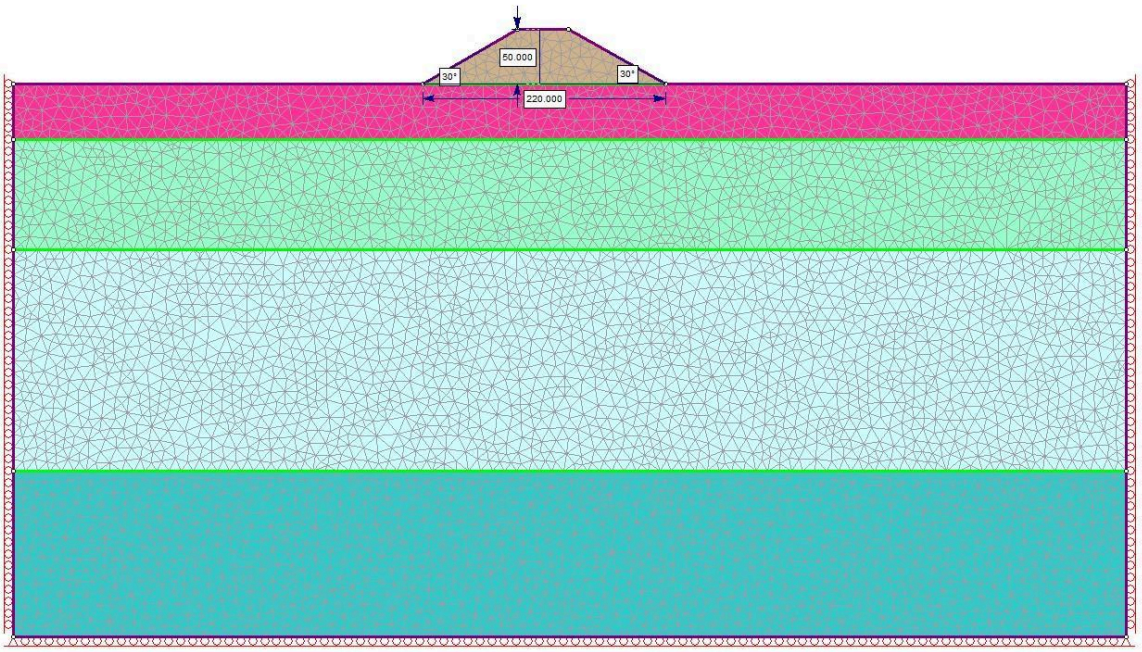


Figure 51 Main plastic model of the waste dump

After necessary observations is done on RS2, plastic model transferred to Slide 2 software and it is seen that waste dump design is safe in terms of sliding conditions. Figure 52 shows the main plastic model of the waste dump and the safety factor on the Slide 2 software. Also, water table line is added into model.

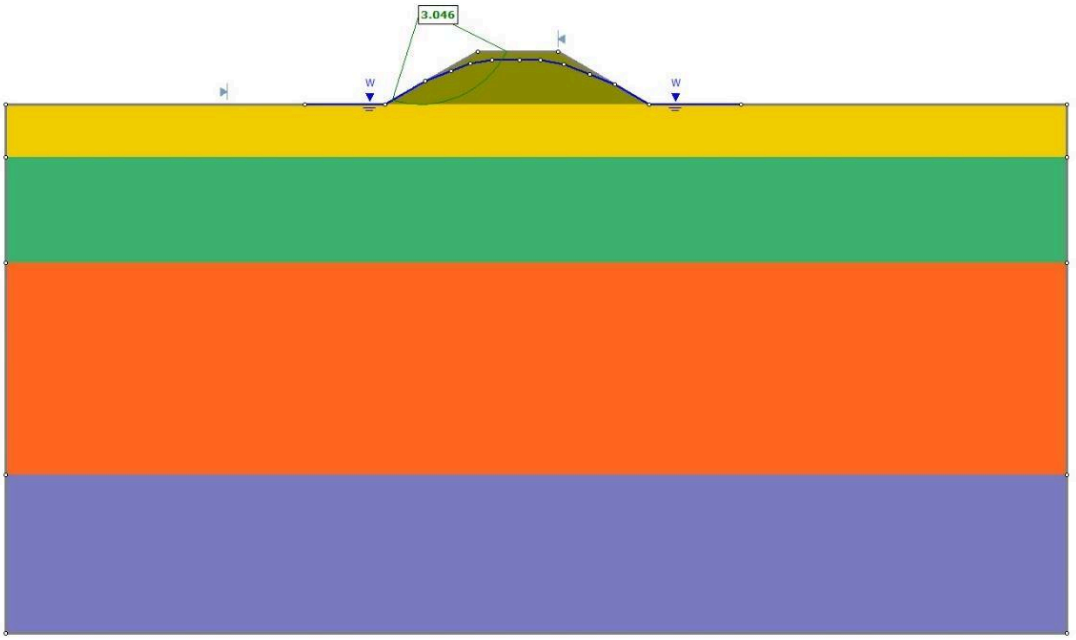


Figure 52 Main model and safety factor in plastic model

6.7. Bearing Capacity

To support the panels, the hydraulic supports are used. The load of the hydraulic supports is 1 MPa. Since hydraulic supports create load on the floor of the panel, the bearing capacity must be examined to prevent failure.

When calculating the bearing capacity of a footing under load, the limit equilibrium method must take into account the variety of failure modes that can occur, such as cracking, crushing, wedging, punching and shearing. The formula of bearing capacity in terms of Mohr-Coulomb failure criteria is,

$$q_f = 2 * C_r * ta\left(45 + \frac{\varphi_r}{2}\right) + 2 * C_p * ta\left(45 + \frac{\varphi_p}{2}\right) * \left(45 + \frac{\varphi_p}{2}\right)$$

Where, q_f= bearing capacity, MPa; C_r =residual cohesion, MPa; C_p =peak cohesion, MPa; φ_r = residual internal friction angle, degree; φ_p= peak internal friction angle. The panels are opened into Limestone and Coal rock layers. Therefore, their bearing capacity must be calculated to analyze and prevent failure.

Since the bearing capacities of Limestone and Coal are greater than the hydraulic supports' load, there will not be a failure because of bearing capacity.

The calculations are shown in the Appendix D.

7. DEVELOPMENT DESIGN

Under the topography, there are 3 coal seams separated by reverse fault. They are named Middle Seam, North Seam, and South Seam. The middle seam is located 930 m under the topography whereas the north and south seams are located 1030 m under the topography. The side view of the seams are shown in Figure 53.

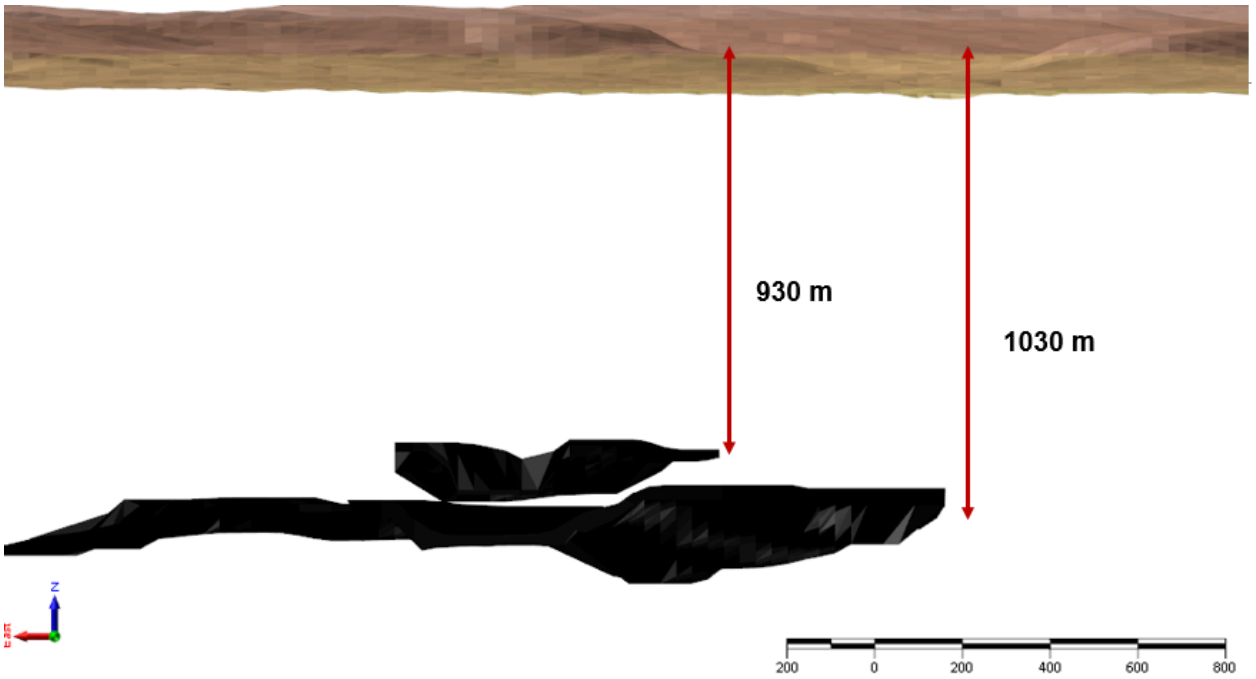


Figure 53. Side view of Seams

The top view of the seams is shown in Figure 54. The amount of reserve is 14.9 million tonnes. To extract the coal reserve, the development stage must be completed first. The development stage consists of the main ramp, secondary ramp, conveyor belt decline, ventilation shaft, levels, head gate, and tailgates. The 3D map of the development is shown in Figure 55. While opening the main access roads, the waste is extracted. The density of waste material is taken as 1.7 tonne/m^3.

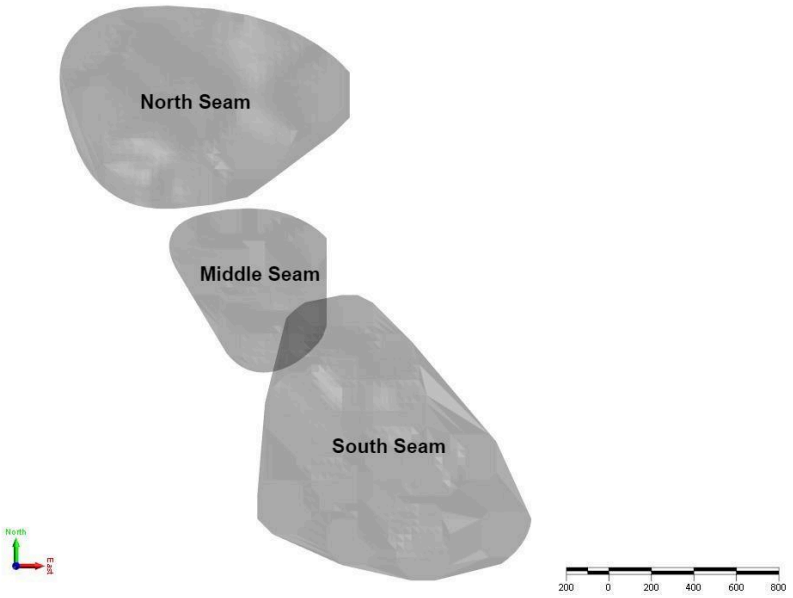


Figure 54. Top View of Coal Seams

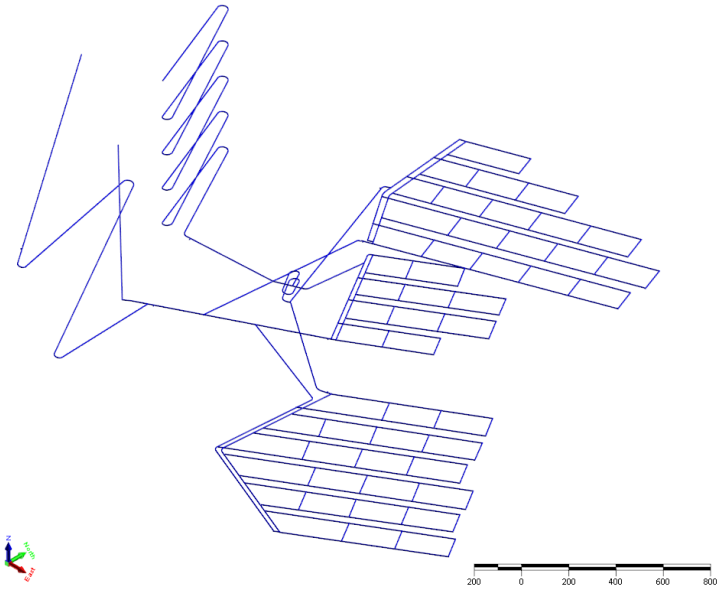


Figure 55. 3D map of Development

7.1 Ventilation Shaft Design

To provide ventilation to the mine site, a circular ventilation shaft is driven from the topography with a length of 817 m. Diameter of the circular ventilation shaft is 5 m. The amount of waste is listed in Table 14.

Table 14. The amount of Excavated Waste Material for Shaft

	Volume (m^3)	Tonnage (tonne)
Ventilation Shaft	15,987.819	27,257.1625

7.2 Main Ramp and Secondary Ramp Design

To transmit the equipment, hydraulic supports, shearers, and man to the working area, the main ramp is driven. The length of the main ramp is 7500 m with 12% which is equal to 6.8° to the downward.

After reaching the middle seam, the secondary ramp is driven to access the north and south seams. The length of the secondary ramp is 655 m. The gradient of secondary ramps is the same as the main ramp.

The cross-section of the main ramps and the secondary ramp is 5*5 in circular arch shape. The radius of turning road is 20 m for both ramps.

While driving the ramps, waste material is extracted. The amount of waste material is listed in Table 15.

Table 15 Amount of Excavated Waste Material for Main and Secondary Ramp

	Volume (m^3)	Tonnage (tonne)
Main Ramp	246,039	418.266
Secondary Ramp	18.012	30.621

7.3 Conveyor Belt Ramp

After mining, the material is needed to be transported to the topography. The haulage is provided by conveyor belts. Therefore, conveyor belt ramps are driven from the topography. The length of the conveyor belt ramp is 4170 m. The gradient of the conveyor belt ramp is %20 which is equal to 11.3° downward. The cross-section of the conveyor belt ramp is 5*5 m in circular arch shape. The amount of waste material is listed in Table 16 while driving the conveyor belt ramp.

Table 16 Amount of Excavated Waste Material for Conveyor Belt Ramp

	Volume (m^3)	Tonnage (tonne)
Conveyor Belt Ramp	134,921	229,365

7.4 Drifts

There are 6 drifts to reach the seams from the main ramp, ventilation shaft, and conveyor belt ramps. They are name D1, D2, D3, D4, D5, and D6. The length of drifts and extracted waste material is listed below in Table 17.

Table 17 Length and Amount of Excavated Waste Material for Drifts

Drifts	Length (m)	Volume (m^3)	Tonnage (tonne)
D1	912	20,390	34,663
D2	927	20,760	35,292
D3	1090	24,438	41,544
D4	1035	23,242	39,820
D5	539	14,060	23,902
D6	407	10,219	17,372

7.5 Roadways

To set up production panels, roadways are driven into seams. Man, equipment and material transportation is provided by roadways. The total length of roadways is 30,723 m. Extracted material from the roadways is considered ore because the gates are driven into coal seams. The total tonnage of extracted material from the roadways is 853,597 tonne.

In Figure 56, the development and panels are shown.

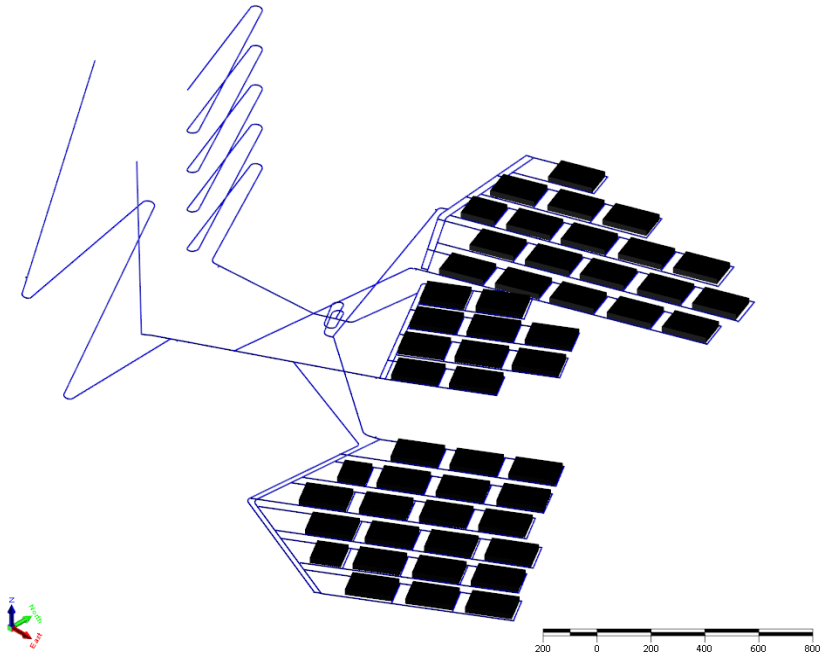


Figure 56.The Panels and Development Design

8. BLAST DESIGN

The blasting design was prepared to open the ramp, shaft and decline. The opening widths for the shaft and ramp are calculated as 5x5 m. For the drilling machine, Sandvik DD422iE Jumbo is chosen. Blasting materials are purchased from the Orica-Nitro company. The parallel hole cut technique is utilized for high automation in drilling. Moreover, it supplies better advance (95 % of drilled depth), longer advance in each round, good fragmentation, and consistent drilling pattern.

8.1 Development Blasting

2 types of materials are purchased. One of them is Powergel Magnum 365 with the size of 38x400 mm and 545 g for development work. Other type is Trimex with a size of

19x735mm and 220 g weight. Trimex will be used in order to make a smooth surface on the roof and wall. Orica LP non-electrical detonator is chosen for safety blasting.

The parameters, tables and figures of blasting are explained in the Appendix E.

8.2 The Development Blasting Near Coal

Blasting will also be made in coal to bring production machines to the panel in order to start production. Since the ore is coal, the explosive will be changed and permissible Super Ajax-Grizu, safe emulsion explosive will be used instead of Orica Magnum 365 as it progresses through the ore. The shape of the openings will be the same and it can be seen from the Figure 57.

The parameters, calculations and tables can be shown in the Appendix E.

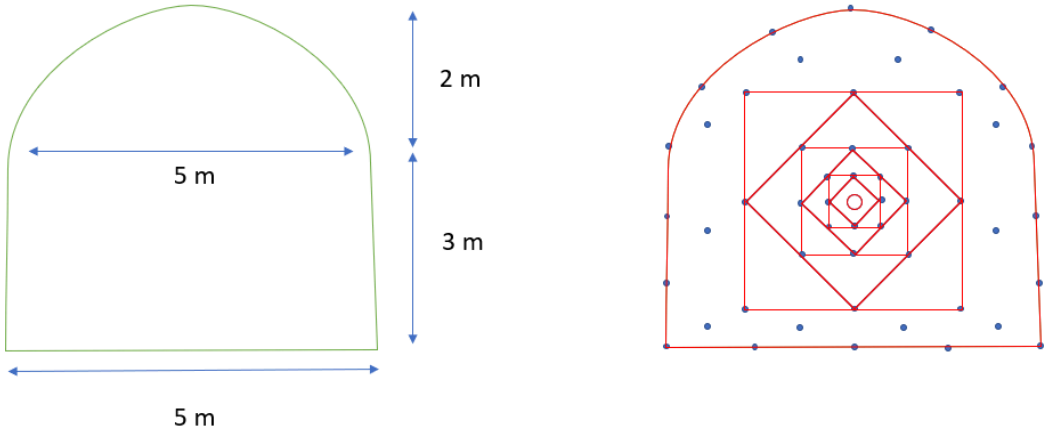


Figure 57. Blasting pattern of development and decline

3

8.3 Shaft Blasting

While the progress on the ramp continues, it is aimed to reach the upper ore body with the shaft and to start production of ore in earlier years. The chosen method is ring drilling for shaft blasting.

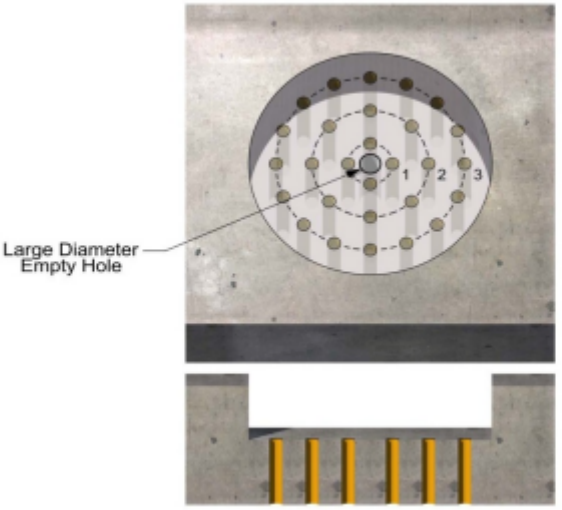


Figure 58. Ring Drilling with Burn Cut Center (Engineers, 2018)

The calculations and designs can be shown in the Appendix E.

9. DESTRESS BLASTING AND COAL BURST

Coal burst is perhaps the most dangerous issue experienced in underground coal mines, which happens in various areas in an assortment of mining frameworks and tasks. This event is a dynamic and devastating energy release. The significant energy factors that ought to be considered to assess the chance of the stone/coal burst events are strain energy, outside work done, active energy, and interior energy or put away energy, which is separated by the strain. energy. When destress blasting is applied in a proper time and way, it is the only way for protecting the results of rock burst occurrence. The level of success of the method varies depending on the method applied. There are different methods of dealing with underground stress. (Zhang Chengguo, 2016)

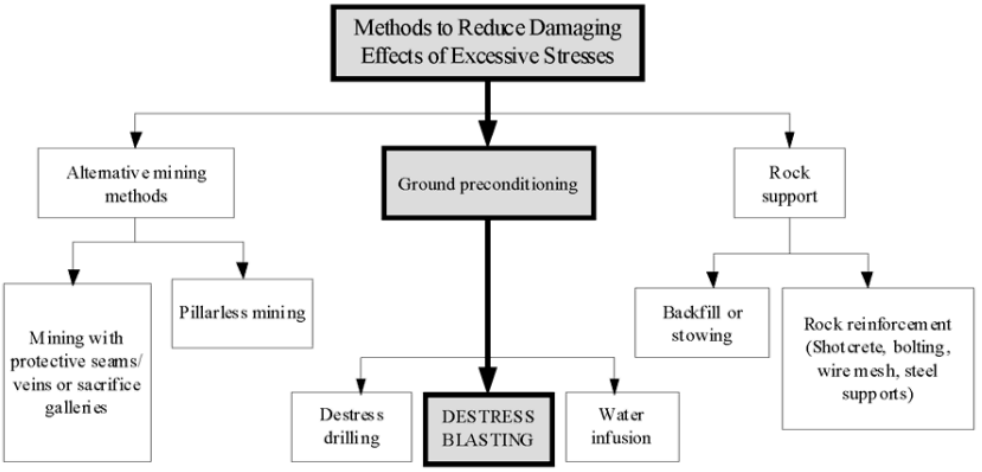


Figure 59.Methods to contain damaging effects of strain bursting (Mitri, 2000)

As the name suggests, this method includes stress and blasting. When an underground cavity is opened, it is applied to disperse the excessive stress accumulated there and to destress the rock. First of all, the geological features of the mine are examined and reported. Identification of all fractures under stress and how the fractures progress and grow are examined. Current procedures have the ability to increase efficiency by extending the duration of fractures through improved exploitation of explosive energy. (Zhang Chengguo, 2016)

9.1. Application of Destress Blasting

Utilization of destress blasting can be characterized into two classifications, to be specific - proactive destress blasting and preconditioning destress blasting. While performing normal blasting, proactive destress blasting is also applied. In this way, the stress can be transferred further into the rock. It aims to transmit the stress in the direction corresponding to the excavation progress. On the other hand, the preconditioning destress blasting is intended to prevent the production of critical stress levels from scheduled mining activities, and the advantages are expected to endure until the mining activities are completed in the preconditioned region. The proposed mining activities near/beneath / in-between previously partly mined / mined regions are the application area of precondition destress blasting. (Mitri, 2000) Also, Figure 60 shows how it can be applied in coal mines.

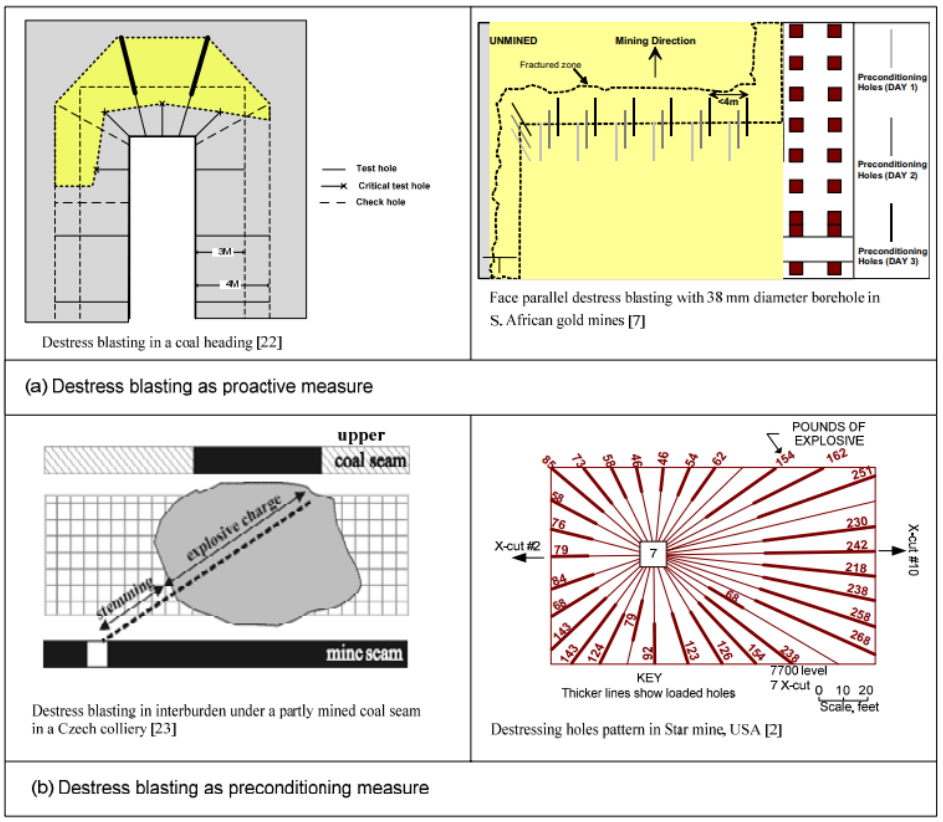


Figure 60.Example applications of destress blasting in coal and non-coal mines (Zhang Chengguo, 2016)

9.2 DESTRESS DRILLING

As the near-surface mines were depleted, the miners began to go deeper into the ground. One of the methods of coping with increasing stress as you go deeper is destress drilling. In terms of experimental and computational study, Liu et al. investigated the destress drilling relief mechanism for rock burst prevention. (J. H. Liu, 2014)

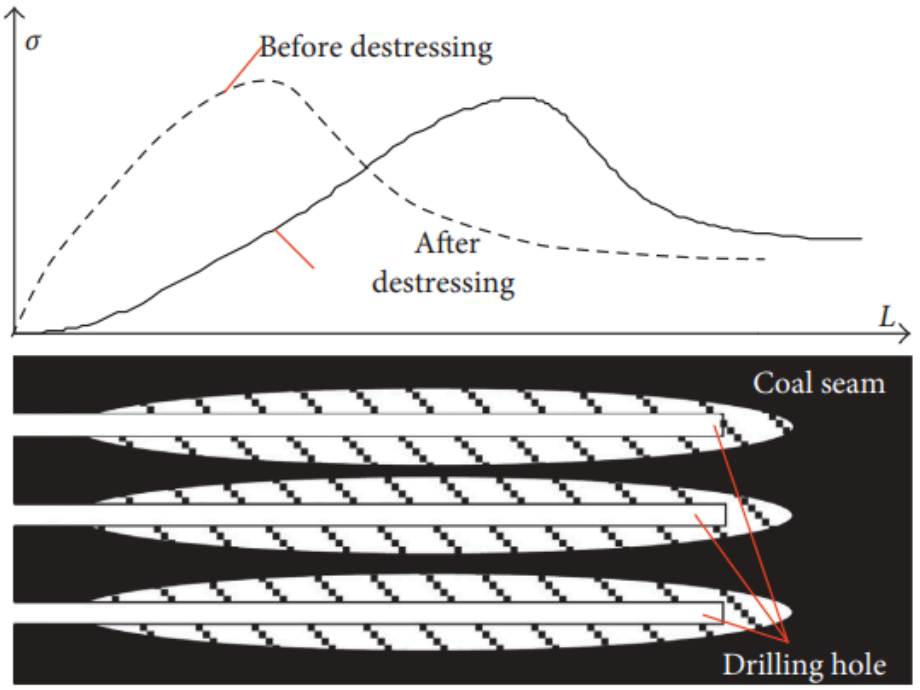


Figure 61.Sketch of destress drilling in coal seam

The mechanism of destress blasting has some important parameters such as; the diameter, space and length of the drill hole. (Tong-bin Zhao, 2018) The destress blasting method, which is widely used in China, is made by drilling large diameter holes. When drilling holes are opened, plastic deformation areas will occur in X-Shape style. When drilling holes are opened, plastic deformation areas in the form of X-Shape will occur. If the density of the drilling wells is high enough, these plastic areas will be connected to each other and thus destressing will be occurred, as shown in Figure 61. This will result in the transfer of the high stress concentration area to the deep coal wall. The stress drilling method works primarily by preventing the creation of high-stress dense zones or by lowering the stress concentration. (J. H. Liu, 2014) Destress drilling method will be applied in this project, as it safely reduces stress and provides a healthier working environment.

10.MATERIALS HANDLING AND EQUIPMENT SELECTION

This part of the report will cover the selection of equipments to be used in development, materials handling, man transportation, rock support and production part of the coal.

10.1 Development Equipments

In the development part of the project, there are shaft, main ramp, conveyor belt ramp, drifts and roads that required for production. This part will cover the equipments required for development.

10.1.1 Load-Haul-Dump (LHD)

LHD units are required during the development part, during the main ramp opening. While selecting the required LHD unit, parameters such as the amount of waste material, the capacity, and dimensions of the LHD are taken into consideration. According to these parameters, two Epiroc Scooptram ST2D LHD units are chosen. One for loading and other for spare unit. Hydraulic cycle time of the LHD is given below. Also, grade performance of the LHD unit is given in Figure 62.

Table 18 Hydraulic Cycle Time of Epiroc Scooptram ST2D (Epiroc, 2022)

Motion Times	
Boom Raising	3.7 sec
Boom Lowering	3.7 sec
Dumping	4.0 sec
Total	11.4 sec

Grade performance

Standard configuration, empty bucket

%	Grade	0.0	2.0	4.0	6.0	8.3	10.0	12.5	14.3	16.0	18.0	20.0	25.0
Ratio	Grade					1:12	1:10	1:8	1:7			1:5	1:4
km/h	1st gear	3.4	3.3	3.3	3.2	3.2	3.2	3.1	3.1	3.0	3.0	2.9	2.8
	2nd gear	6.8	6.6	6.4	6.2	6.0	5.8	5.5	5.3	5.1	5.0	4.6	3.6
	3rd gear	11.3	10.6	10.3	9.8	9.2	8.2	6.9	5.7	5.3			
	4th gear	16.2	15.1	13.1	11.3								

3% rolling resistance assumed. Actual performance may vary depending on the application.

Grade performance

Standard configuration, loaded bucket

%	Grade	0.0	2.0	4.0	6.0	8.3	10.0	12.5	14.3	16.0	18.0	20.0	25.0
Ratio	Grade					1:12	1:10	1:8	1:7			1:5	1:4
km/h	1st gear	3.3	3.2	3.1	3.0	2.9	2.9	2.8	2.8	2.7	2.6	2.5	2.5
	2nd gear	6.7	6.4	6.2	5.9	5.6	5.4	4.9	4.5	4.2	3.8	3.4	
	3rd gear	11.0	10.2	9.7	8.6	7.0	6.0						
	4th gear	15.0	13.3	10.1									

3% rolling resistance assumed. Actual performance may vary depending on the application.

Figure 62. Grade performance of Epiroc Scooptram ST2D (Epiroc, 2022)



Figure 63. Epiroc Scooptram ST2D (Epiroc, 2022)

10.1.2 Truck

During the development works, a truck is also required next to the LHD to haul the waste materials that comes from the development blasting at the main ramp opening. Considering the necessary parameters, CAT AD63 mining truck can be chosen. For the main road development 4 trucks are enough to doing the job and additional 2 more trucks required as a spare unit.



Figure 64. CAT AD63 (CAT, 2022)

10.1.3 Face Driller

Face driller is required when development blasting. In this project, 3 m of progress is expected for each blast during ramp development. Therefore, Sandvik DD422iE can be chosen for Jumbo driller. Sandvik DD422iE has 2 booms, and it can reach over 3 m hole depth. 2 Jumbo driller are required for continuous development. One for drilling and other for spare unit.



Figure 65. Sandvik DD422iE (SANDVIK, 2022)

10.1.4 Road Header

Coal is an ore that can easily burn and explode. Therefore, it is not safe to blast the parts near or inside the coal and for roadways near the seams, road header can be used. While choosing the road header, the characteristics of the rocks and the required excavation speed are taken into consideration. According to these, CTR300A from CREG can be chosen. Header has 3.6m with and 3.74m height. Cutting ranges are 4.2m-6.5m for height and 4.2m-7.0m for width. İts available for opening both roadway development and creating entrances for coal seam. At least 3 units are required for continuous development and 1 for spare unit.



Figure 66. CREG CTR300A (CREG-Germany, 2022)

10.1.5 Pneumatic Manual Rock Drill

27 kg Toyo TY24C pneumatic manual rock drill can be used for ventilation shaft sinking operation. This equipment can drill up to 5 m long and it has 32-46 mm diameter which can be used as blasthole as well. In shaft blasting design, there are 20 blasthole, so 3 of those drills can be enough to do the job in a day.

Details	Unit	Arguments
Weight	kg	27
Length	mm	610
Cylinder diameter	mm	66.7
Piston diameter	mm	68
Air consumption	M3/min	2.7
Shank size	mm	H22*108卤1
Working pressure	Mpa	0.5-0.63
Drilling diameter	mm	32-46
Drilling depth	m	5
Air hose inner diameter	mm	19

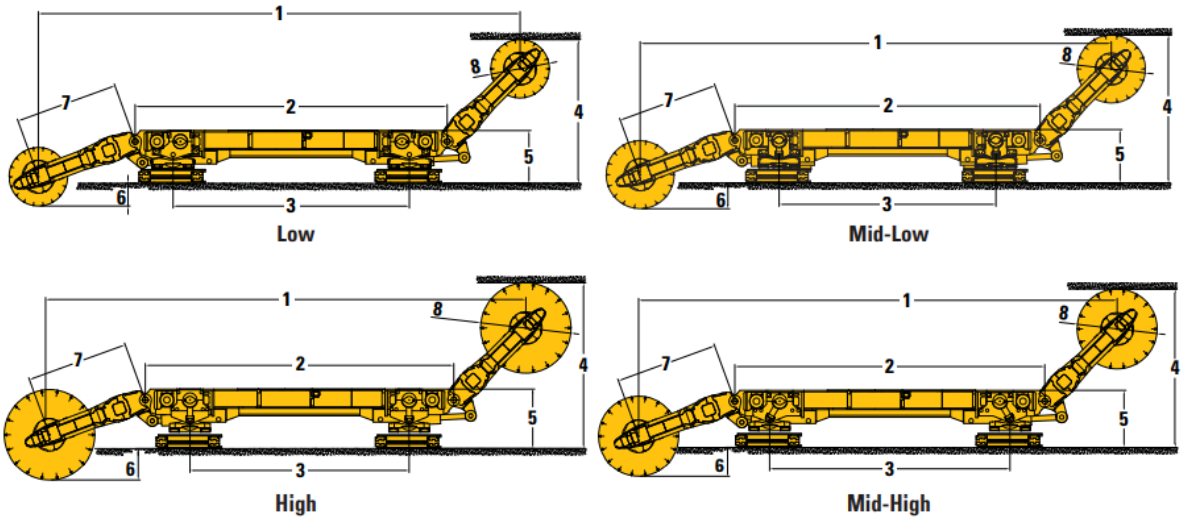
Figure 67. 27kg Toyo TY24C (Chinacoalintl, 2022)

10.2 Production Equipment

After the development work, the coal seam is reached, and production can start. Required equipment for coal production are shearer, hydraulic support, and armored face conveyor (AFC).

10.2.1 Shearer

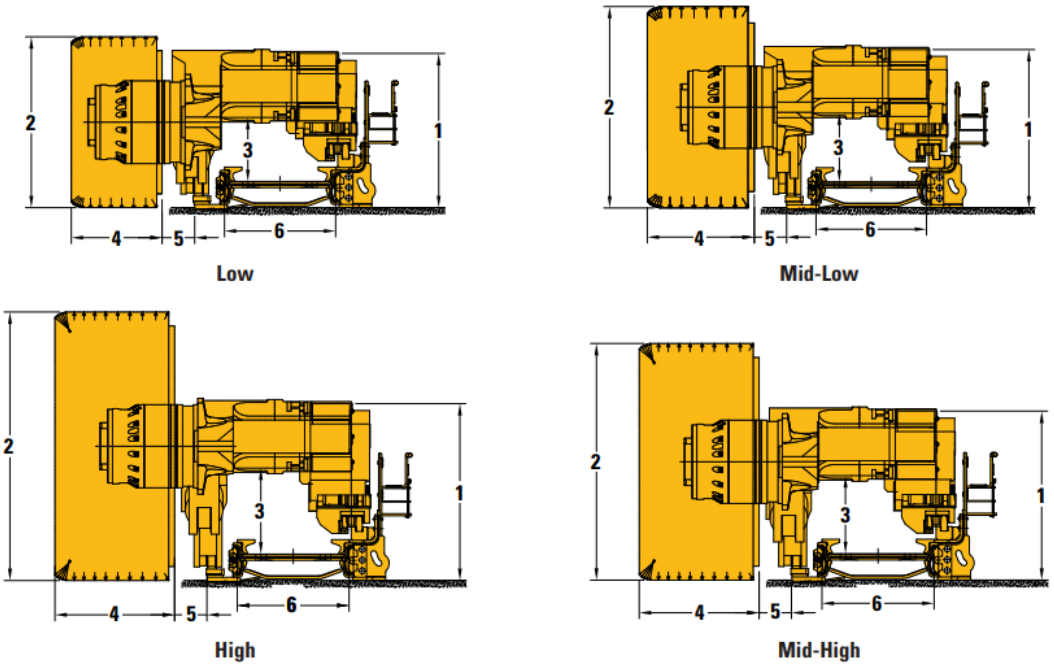
The 2 CAT EL2000 is chosen for shearer with 1.8-4.5m seam range. Determined panel height is 4m for this mine and these range is enough to work with it. In addition to these, maximum drum diameter is 2.5 m which allows faster production. This shearer is working with AC power. According to the stability studies, shearers can work on two different panels at the same time if the production rate is to be increased. According to the cutting speed and drum diameter, the production of shearer is enough to provide the required production.



Dimensions (All dimensions are approximate.)

			Low		Mid-Low		Mid-High		High	
1	Distance between Drums with Arms Horizontal		14 155 mm	537 in	14 155 mm	537 in	14 155 mm	537 in	14 155 mm	537 in
2	Distance between Ranging Arm Hinge Points		8525 mm	336 in	8525 mm	336 in	8525 mm	336 in	8525 mm	336 in
3	Distance between Trapping Shoe Centers		6453 mm	254 in	6056 mm	238 in	6612 mm	260 in	6056 mm	238 in
4	Cutting Heights	Maximum	3930 mm	155 in	4135 mm	163 in	4360 mm	172 in	4570 mm	180 in
		Minimum	1800 mm	71 in	1900 mm	75 in	2200 mm	87 in	2500 mm	98 in
5	Height to Top of Machine Main Body		1444 mm	56.8 in	1495 mm	58.8 in	1571 mm	61.8 in	1647 mm	64.8 in
6	Shearer Drum Undercut of Floor		619 mm	24 in	718 mm	28.3 in	793 mm	31.2 in	866 mm	34 in
7	Ranging Arm Length (Hinge to Drum)		2815 mm	111 in	2815 mm	111 in	2815 mm	111 in	2815 mm	111 in
8	Diameter of Shearer Cutting Drum		1600 mm	63 in	1900 mm	75 in	2200 mm	87 in	2500 mm	98 in

Figure 68. Dimensions of CAT EL2000 (CAT, 2022)



Dimensions (All dimensions are approximate.)

		Low		Mid-Low		Mid-High		High	
1 Machine Height over Main Body		1444 mm	56.8 in	1495 mm	58.8 in	1571 mm	61.8 in	1647 mm	64.8 in
2 Ranging Arm Cutting Drum Diameter		1600 mm	63 in	1900 mm	75 in	2200 mm	87 in	2500 mm	98 in
3 Vertical Tunnel Clearance		598 mm	23.5 in	649 mm	25.5 in	725 mm	28.5 in	801 mm	31.5 in
4 Maximum Cutting Drum Overall Width		1150 mm	45.3 in	1150 mm	45.3 in	1150 mm	45.3 in	1150 mm	45.3 in
5 Clearance from Drum to AFC Toeplate		300 mm	12 in	300 mm	12 in	300 mm	12 in	300 mm	12 in
6 AFC Pan Width	Maximum	1342 mm	53 in	1342 mm	53 in	1342 mm	53 in	1342 mm	53 in
	Minimum	1032 mm	41 in	1032 mm	41 in	1032 mm	41 in	1032 mm	41 in

Figure 69. Dimensions of CAT EL2000 (CAT, 2022)

10.2.2 Hydraulic Roof Support

Hydraulic roof supports are a support equipment used in the production phase in the LTCC method. When determining the dimensions of the coal panels, 1 MPa support is applied to plastic models. Therefore at least 1 MPa support is needed for the project. According to this information CAT AEHQ6801 is chosen. It supports up to 1.1 MPa. Addition to these, it also has rear armored conveyor to letting longwall top coal caving. That means, this equipment increases overall production capacity.



Figure 70. CAT AEHQ6801 Roof Support System (CAT, 2022)

Cat Roof Supports		
Height Closed	500-2900 mm	19.7-114.2 in
Height Extended	1100-6100 mm	43.3-240.2 in
Shield Centers	1500-2050 mm	59-80.7 in
Nominal Leg Forces	2 × 800-2 × 7156 kN	2 × 90-2 × 805 ton
Shield Density	up to 111 tonne/m²	up to 12.5 ton/ft²
Hydraulic Leg	up to 500 mm	up to 19.7 in
Canopy	Rigid design (with telescopic flipper)	
Canopy Length	2997-8000 mm	118-315 in
Forepole Length	1980-4520 mm	78-178 in
Rear Projection	850-1448 mm	33.5-57 in
Base	Split design OR rigid design with telescopic relay bar/pusher cylinder	
Relay Bar		
Direct Mounted OR Reverse Mounted	650-1067 mm effective web	25.6-39.4 in effective web
Shifting Cylinder (with Reedrod)	700-1080 mm	27.6-42.5 in
Shield Pull Force	185-565 kN/ 315/320/350 bar	20.7-63.3 ton/ 4568/4640/5076 psi
Conveyor Push Force	89-815 kN/ 315/320/350 bar	10.0-91.5 ton/ 4568/4640/5076 psi
Side Seals – on Canopy, Caving Shield (and rear lemniscates)		
Both Sides Fixed/One Side Fixed, One Side Movable, Rehandable/Not Rehandable	up to 300 mm stroke	up to 11.8 in stroke
Materials	High strength steel	
Test	EN 1804, different test program with up to 90,000 cycles	
Weight	6.5-63.5 tonne	7.15-70 ton

Figure 71. CAT AEHQ6801 Specifications of Roof Support System (CAT, 2022)

10.2.3 Armored Face Conveyor (AFC)

AFC transfers the coals during production from face to the panel entrance. These AFCs are located under the shearer and coals are directly fall into AFC. As an AFC, PF3 from Cat is chosen for armored face conveyors with up to 1300 tph capacities. Typical technical data is given Table 19. Also, speed vs. capacity graph of the AFC is given in Figure 72.

Table 19.Typical technical data of CAT PF3

Type	Width Outside	Top Plate Thickness	Bottom Plate Thickness	Dogbone Breaking Force	Production Average per Hour
CAT PF3	822 mm	30 mm	20 mm	2000 kN	1300 tonnes

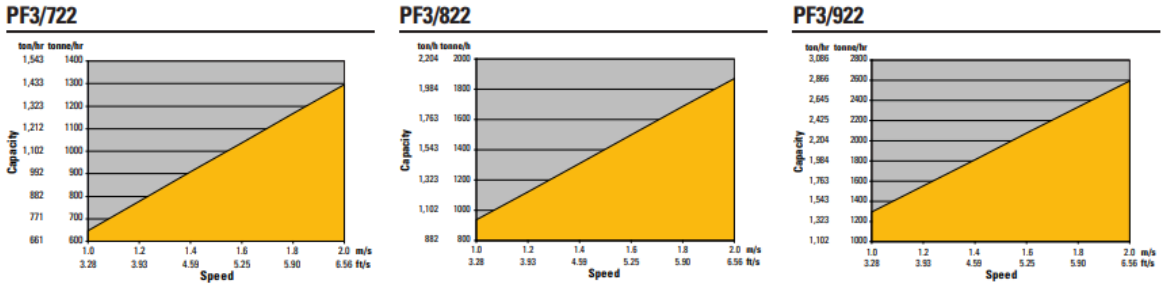


Figure 72.Speed vs. Capacity Graph of the CAT PF3

10.3 Material Transportation

The coals obtained from the production will be transferred to the surface by conveyor belt by using the conveyor belt ramp. Since the conveyor belt ramp has 20% grade which is 11.31°, selected conveyor belt should work effectively in this condition. Pylon Plus 1000/5 from Continental is selected because of its specifics. It is suitable for underground coal haulage and it has availability for required slope.



Figure 73.Continental Pylon Plus (Continental Industry, 2022)

10.4 Development Supporting

According to the development and rock stability studies, bolts and concrete should be applied into the ramp road. Considering this situation, supporting equipment are selected.

10.4.1 Roof Scaling Machine

Roof scaling machine is used for to increase the safety before the application of concrete. For this reason, roof scaling machine must be used for development supporting. SCALER S320 from GETMAN is selected for this job. It has availability of 5.5 m wide and up to 9 m height for galleries. (GETMAN, 2022) For our design, we need at least 2 of that equipment.

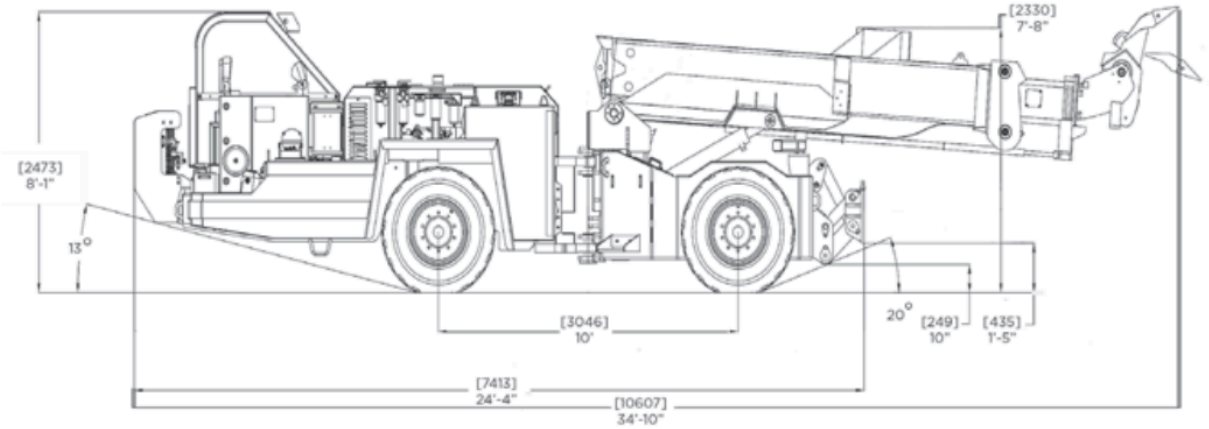


Figure 74. Dimensions of equipment in millimeters (Getman, 2022)

10.4.2 Mixer

COGUARO 4 from CITA is selected as concrete mixer truck. According to truck specification, the size of it fits for access roads of coal seam. For our design, at least 4 of them are required to supply the shotcrete operation.

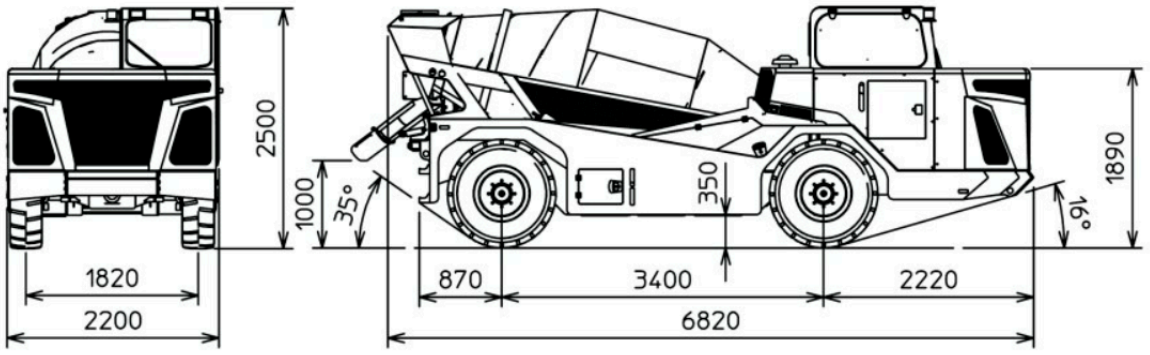


Figure 75. Dimensions of equipment in millimeters (CITA, nd.)

10.4.3 Shotcrete

Dingo from CITA is selected for shotcrete operation. Specification of it is enough for our design. Additionally, for our design we need 2 of them and 1 for spare unit.

10.4.4 Concrete Station

Mobil Mix 30 from Ibar Machine, selected as a mobile concrete batching plant. It has 30 m^3/hr capacity which is enough for our development stage.

10.4.5 Bolting

Sandvik DD422iE is selected for bolting operation. The equipment is optimal for our access roads. Additionally, for our design 2 of them required.

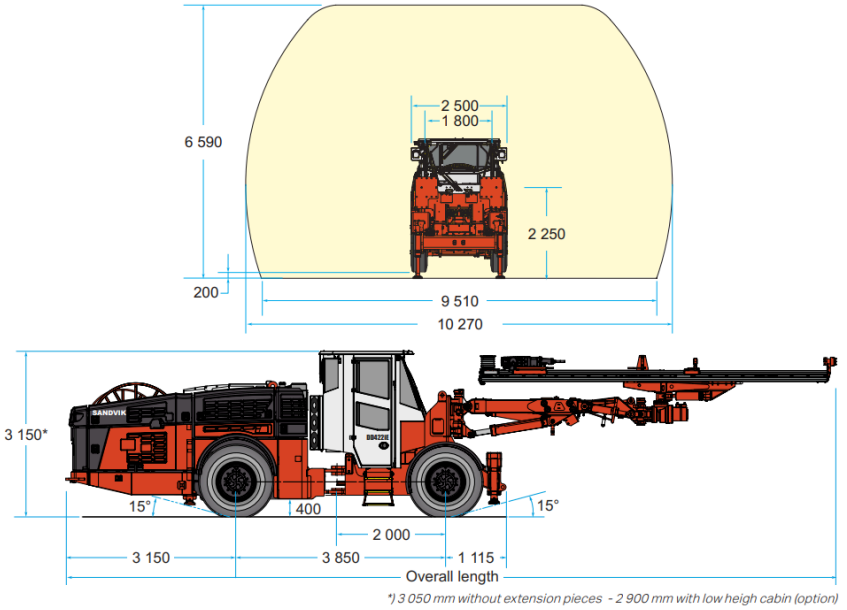


Figure 76. Equipment Dimensions Millimeters (Sandvik, nd.)

11. PRODUCTION SCHEDULING

Under the production scheduling title, panel plans, production program and sales are covered and studied.

11.1 Panel Plans

The naming of the panels is shown in Figure 77 on the resource modelling made for the mining project.

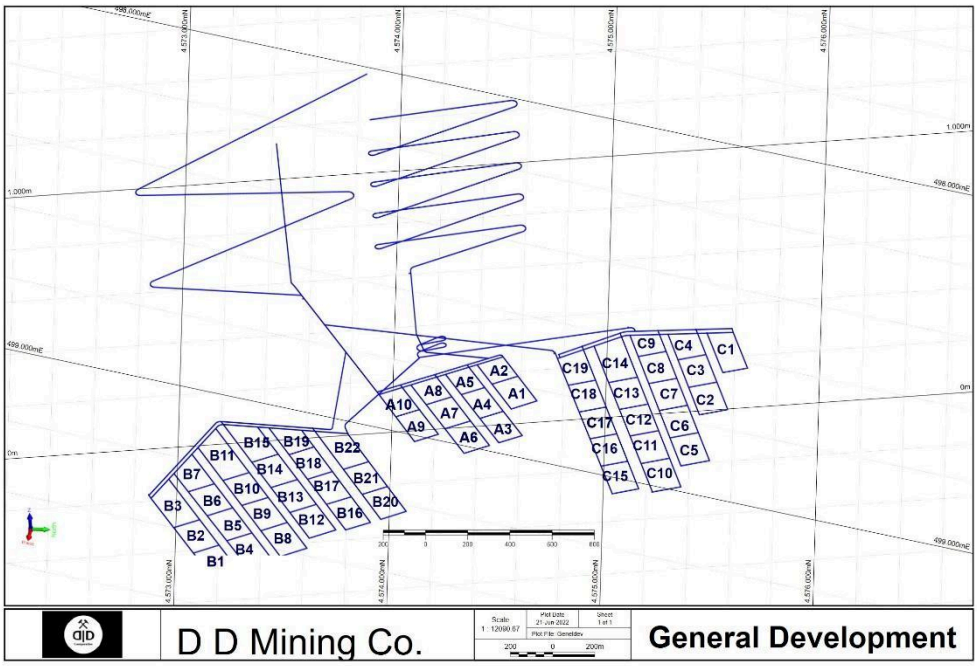


Figure 77.Coal Panels Names

Panel size is 120*180 m^2 and pillars are 35 m. Because the coal mine is located in a faulted location, stability studies have been carried out to prevent any collapse and to extract as much coal as feasible. Due to the fault, approximately 250 m of the northern part of the south seam will not be taken after the middle seam is removed. Therefore, Thus, possible collapse and accidents were avoided. After the stability study, the panels were drawn on Micromine with the dimensions decided and a total of 46 panels were planned.

11.2 Production Program

After the development works, conveyor belt roads, man access roads, head gates, and tail gates will be opened by using road header to get the seams. Since it would be dangerous to blast in and near coal, these roads were opened by using road header. Coal

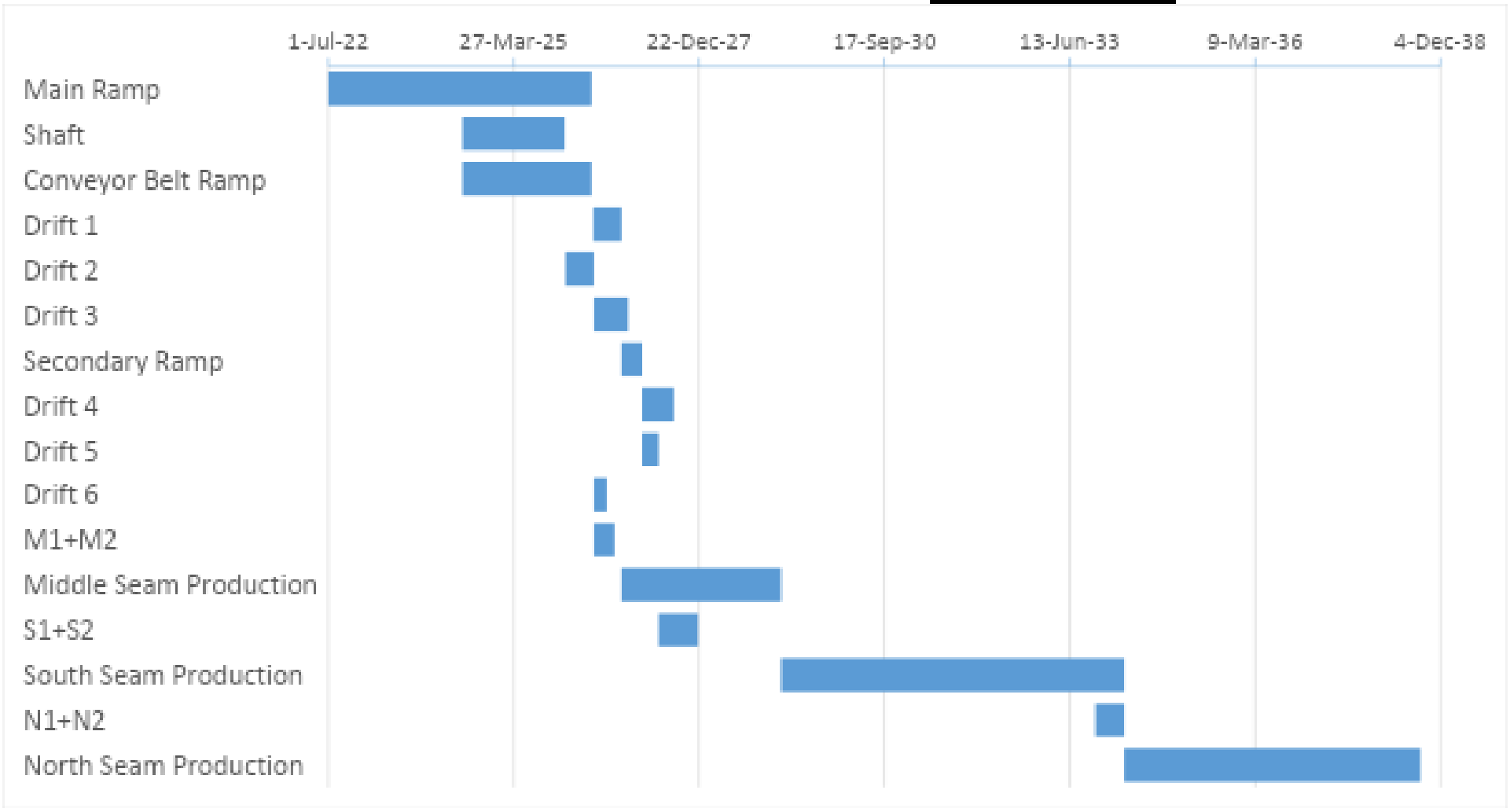
production will start after the development work is completed. First, production will be made from the middle seam, then the south seam and lastly the north seam will be mined, and the production will be completed. Production sequence is given in figure 29. While making production calculations, it has been taken into consideration that the panel dimensions are 120*180*4 m where the shape is possible and there will be an average of 6 m of caved coal. Also, face cutting speed is taken as 480 minutes/meter which is equal to 8 hours/meter. According to this, daily progress is founded as 2.5 m. It means, total panel cutting day is found as 72 days. Preparation stage such as installation of hydraulic supports for the panel opening is taken as 12 days. According to this, production schedule is created.

Table 20 Production Schedule of Panels

Panel Numbers	Panel Tonnage (tonne)	Total Extracted Tonnage (tonne)	Face Cutting Speed (hr/1 m advance)	Total Time for Full Panel Excavation (day)
Middle Seam				
A 1	112,320	280,800	8	84
A 2	112,320	280,800	8	84
A 3	112,320	280,800	8	84
A 4	112,320	280,800	8	84
A 5	112,320	280,800	8	84
A 6	112,320	280,800	8	84
A 7	112,320	280,800	8	84
A 8	112,320	280,800	8	84
A 9	112,320	280,800	8	84
A 10	112,320	280,800	8	84
South Seam				
B 1	112,320	280,800	8	84
B 2	112,320	280,800	8	84

B 3	112,320	280,800	8	84
B 4	112,320	280,800	8	84
B 5	112,320	280,800	8	84
B 6	112,320	280,800	8	84
B 7	74,880	187,200	8	60
B 8	112,320	280,800	8	84
B 9	112,320	280,800	8	84
B 10	112,320	280,800	8	84
B 11	112,320	280,800	8	84
B 12	112,320	280,800	8	84
B 13	112,320	280,800	8	84
B 14	112,320	280,800	8	84
B 15	112,320	280,800	8	84
B 16	112,320	280,800	8	84
B 17	112,320	280,800	8	84
B 18	112,320	280,800	8	84
B 19	66,144	165,360	8	54.4
B 20	112,320	280,800	8	84
B 21	112,320	280,800	8	84
B 22	112,320	280,800	8	84
North Seam				
C 1	112,320	280,800	8	84
C 2	112,320	280,800	8	84
C 3	112,320	280,800	8	84
C 4	112,320	280,800	8	84

C 5	112,320	280,800	8	84
C 6	112,320	280,800	8	84
C 7	112,320	280,800	8	84
C 8	112,320	280,800	8	84
C 9	112,320	280,800	8	84
C 10	112,320	280,800	8	84
C 11	112,320	280,800	8	84
C 12	112,320	280,800	8	84
C 13	112,320	280,800	8	84
C 14	112,320	280,800	8	84
C 15	112,320	280,800	8	84
C 16	112,320	280,800	8	84
C 17	112,320	280,800	8	84
C 18	112,320	280,800	8	84
C 19	112,320	280,800	8	84



In Figure 28, production and development planning is given as a gantt chart.

11.3 Sales

The coal reserve has high calorific value which is between 5000-7500 kcal. Since the calorific value is high, the mine coal is planned to sell iron and steel factory. The nearest iron and steel factory is KARDEMİR, where is located 35 km. The price is 155$ (TTK,2022)

12.VENTILATION

Ventilation is the air flow design provided inside the mine to control the amount of air, air velocity and harmful gases in the mine. Its main purposes are to create a healthy environment for the workers by providing sufficient and correct air inside, to provide sufficient air quantity for the machines and to bring the gases released during mining to values that will not affect human health.

Required air is set regarding the number of working people, amount of methane and dust, IC engines used, and blasting done in the face. The required air amount is determined at 12,500 m^3 / min.

In underground mining, there are different velocities limits according to the locations. The limits can be seen in the Table 21 below (Demirel, 2021). The air velocity is arranged as 8 m/sec limit value.

Table 21 Regulation limits (Demirel,2021)

Location	**Air Velocity(m/sec)**
Ventilation Shaft	20
Hoisting Shaft	10
Main Ventilation Tunnels	8
Main Access Tunnels	6
Conveyor Galleries	5
Face	4

The development was taken as an output from Micromine software, and this output was used for network design in the VentSim software. The Figure 79 shows the network design. Friction factor values are taken as in the Table 22 below (Demirel,2021). Air resistance is calculated by the VentSim.

Table 22. Friction Factor for the Locations (Demirel,2021)

Location	**Friction Factor (f)(kg/m^3)**
Arched Levels (5x5)	0.011

Shafts	0.003
Production Drifts	0.016
Ramps	0.013

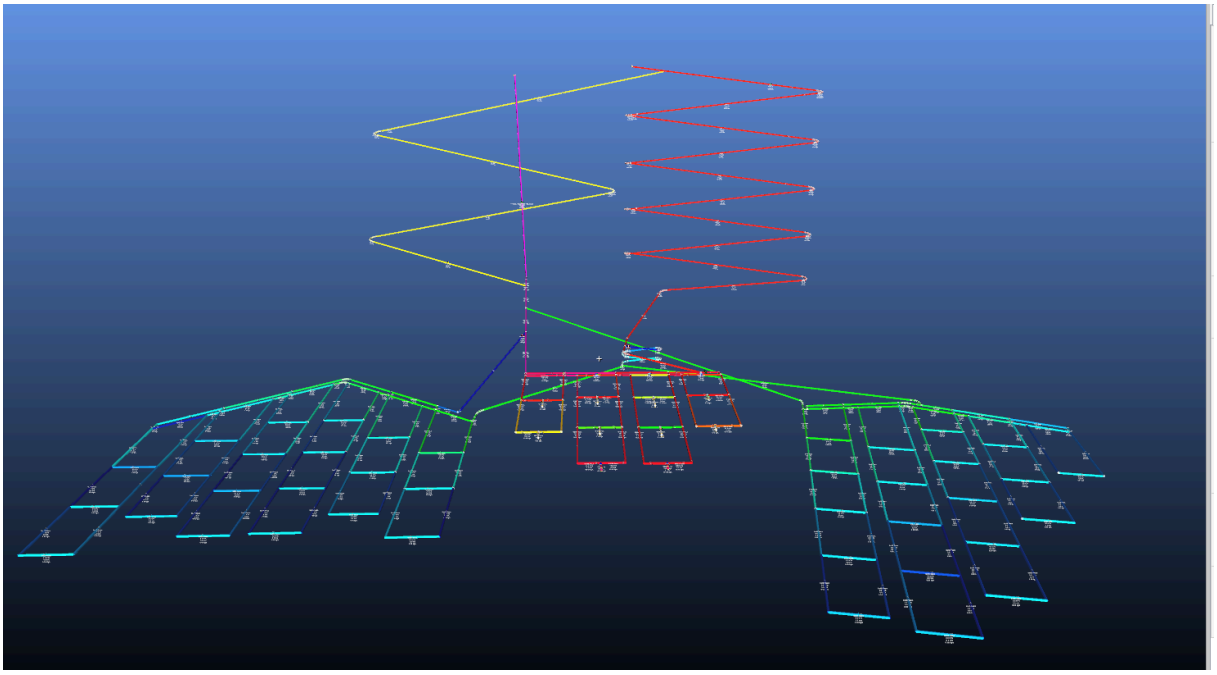


Figure 79. The Ventilation Network Design

There are 1 ventilation shaft and 2 ramps in the main design. Since it is a coal mine, an exhaust type fan was used considering the risks. This fan is placed on the ventilation shaft. 300m^3/sec air flow is provided. This value has been calculated by considering the amount of air required by the equipment to work in the mine due to the engine power, the number of people to work, methane emissions and other factors and is given.

Table 23 The gas emissions

The Gases	
Methane (CH4)	0.3 m^3/sec
Dust	0.1 m^3/sec

The air flow conditions when the back of the middle panels is operating are as shown in Figure 80. Air Flow Conditions (Middle Seam). After the coal work on the panels is finished, the panels will be closed with doors and their connection with other work areas will be closed for safety and precaution.

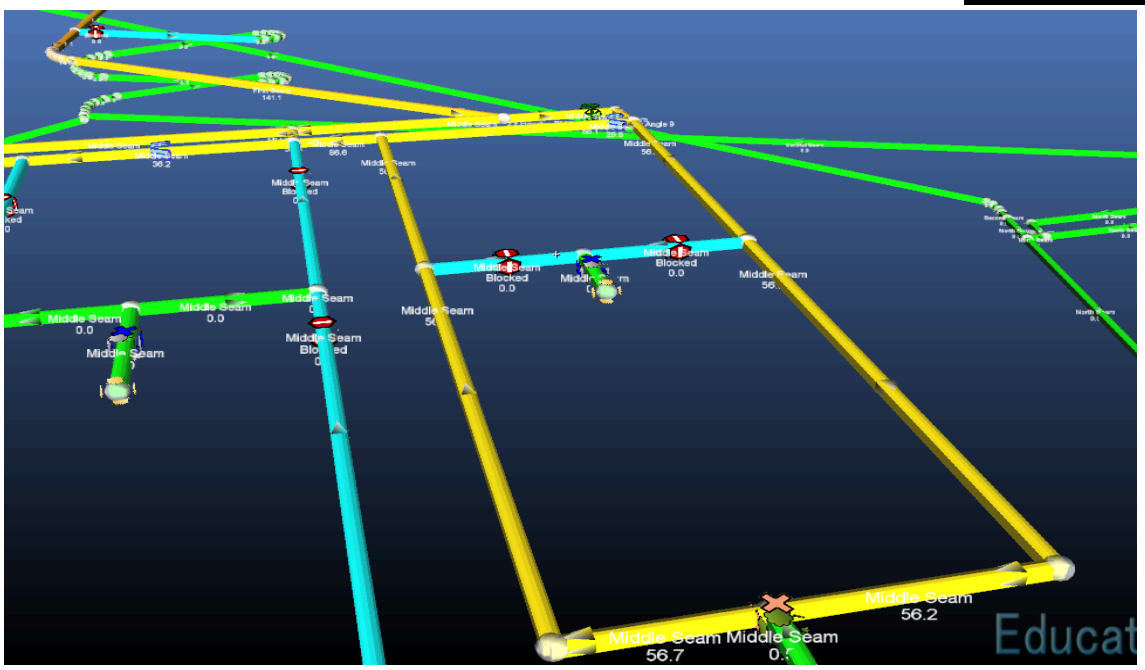


Figure 80. Air Flow Conditions (Middle Seam)

Flow rates have been adjusted in accordance with the regulations, with reference to the values. Thanks to the regulators placed in the drift, the air flow rates provided to the inside of the panel have been reached to the ideal and required values and kept the methane emission at minimum levels.

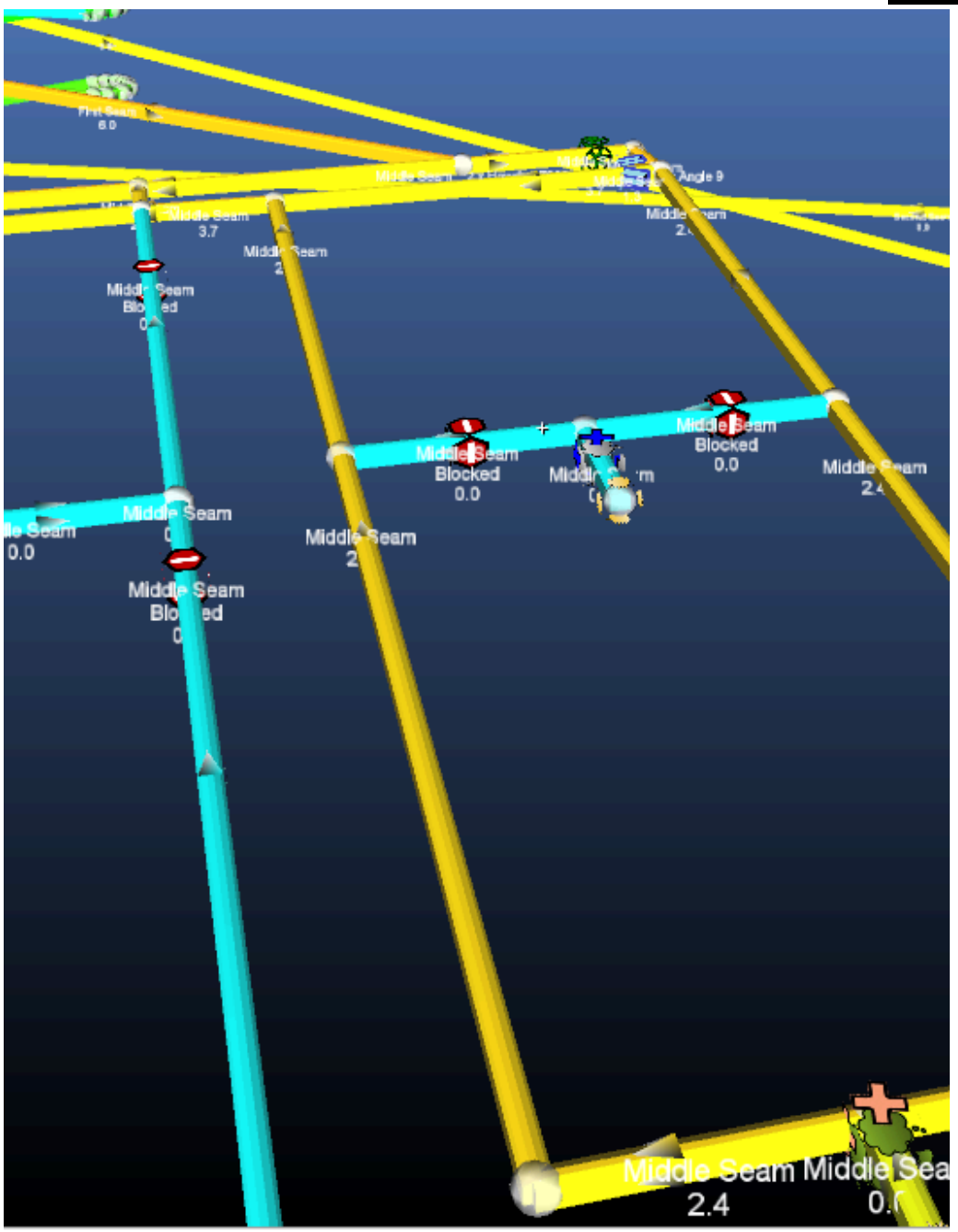


Figure 81. The Air Velocity Values of Middle Panel Drifts

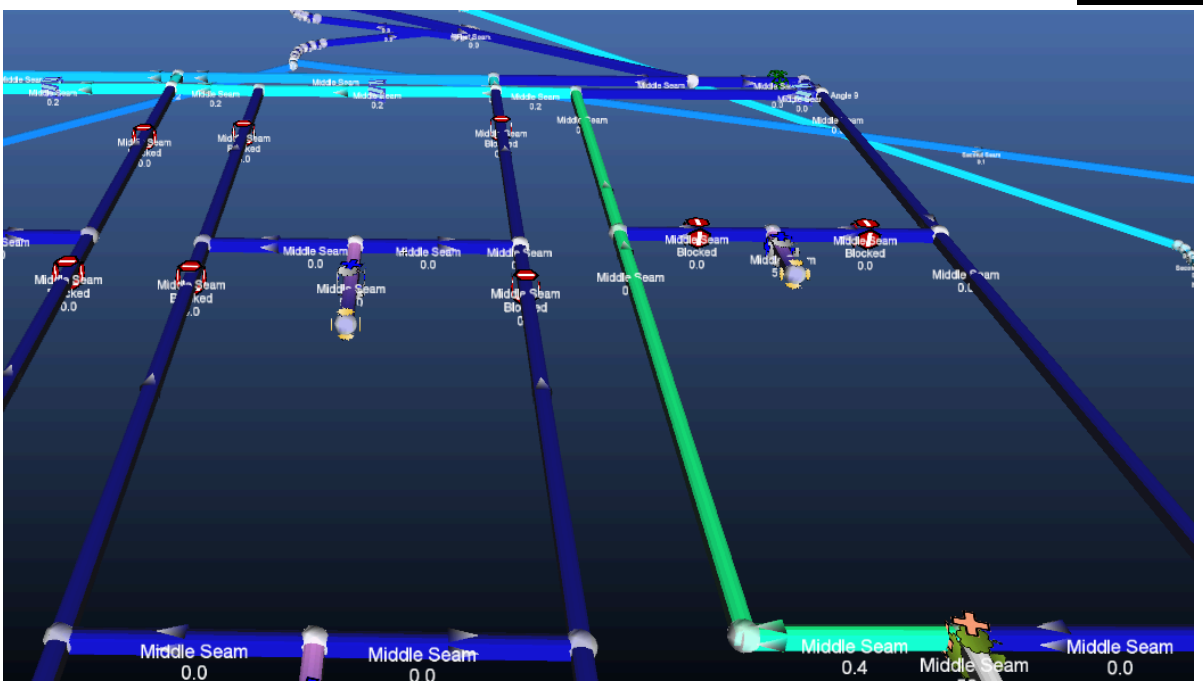


Figure 82. The methane emission of Middle Seam

Ventilation conditions were checked for methane, which is one of the main problems in coal mining. The limit value for methane should be below 1% and the methane status appears to be 0.4% as seen in Figure 82. Control was provided for methane values in each different panel and the suitability of methane values was ensured.

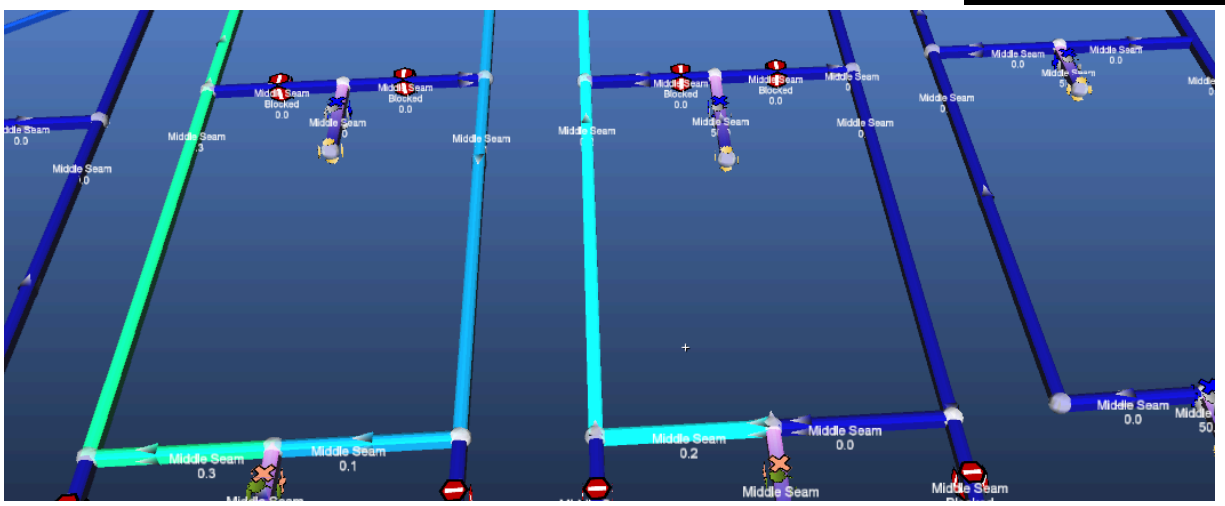


Figure 83. The methane emission of Middle Seam Both Panel

For dust, which is one of the other troublesome situations in coal mining and carries the risk of firing, different samples were made in each panel and extra precautions were taken as shown in Figure 83. Due to the air flow and flow rates provided, limit values are provided for dust.

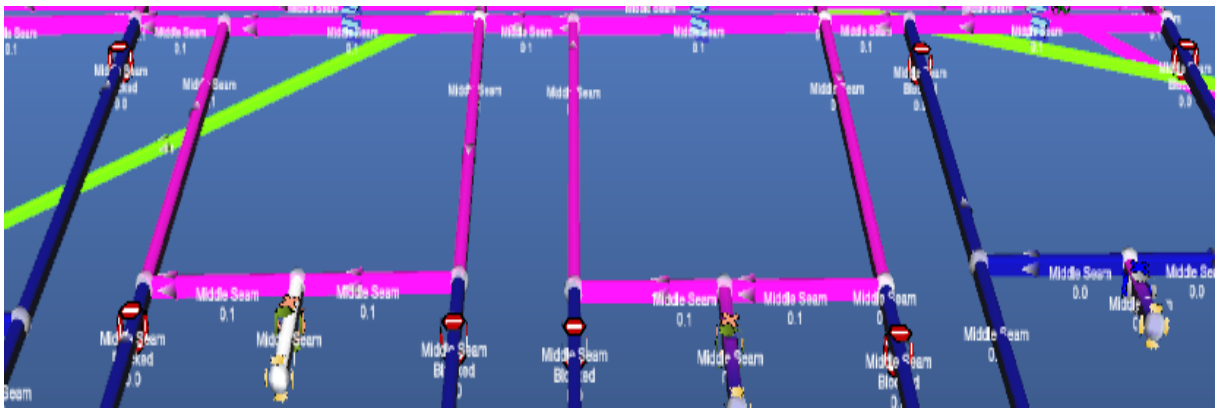


Figure 84. The Dust Emission of Middle Seam

12.1 Fan Selection

For production process, Howden 11200-AMF 6600 is chosen. The flow rate is about 172m^3/sec. Fan efficiency is approximately %75.

	m³/s	%
	154.7	73.3
	167.3	74.9
	179.9	75.2
	192.5	73.8
	205.1	70.1
	217.7	62.9
	230.3	50.3
	242.9	28.5
	250.5	8.2

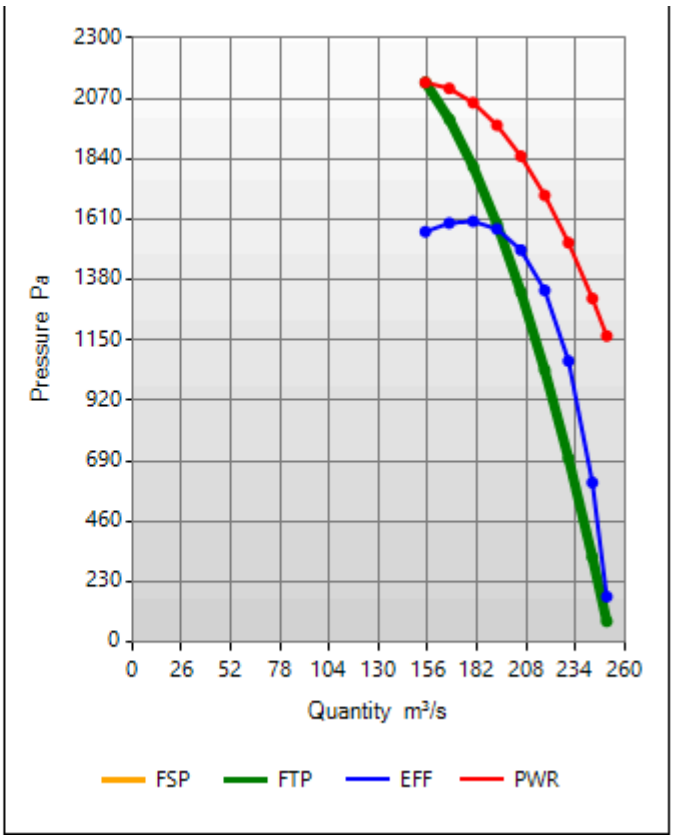


Figure 85. The Main Exhaust Fan Curve

For development process, Howden 8400-VAX 3150 is chosen. The flow rate is about 100 m^3/sec. Fan efficiency is approximately %65.

	m³/s	%
	68.6	65.4
	75.0	68.8
	81.4	70.9
	87.8	71.5
	94.1	70.0
	100.5	65.4
	106.9	56.4
	113.3	40.6
	119.7	13.2
	120.8	6.9

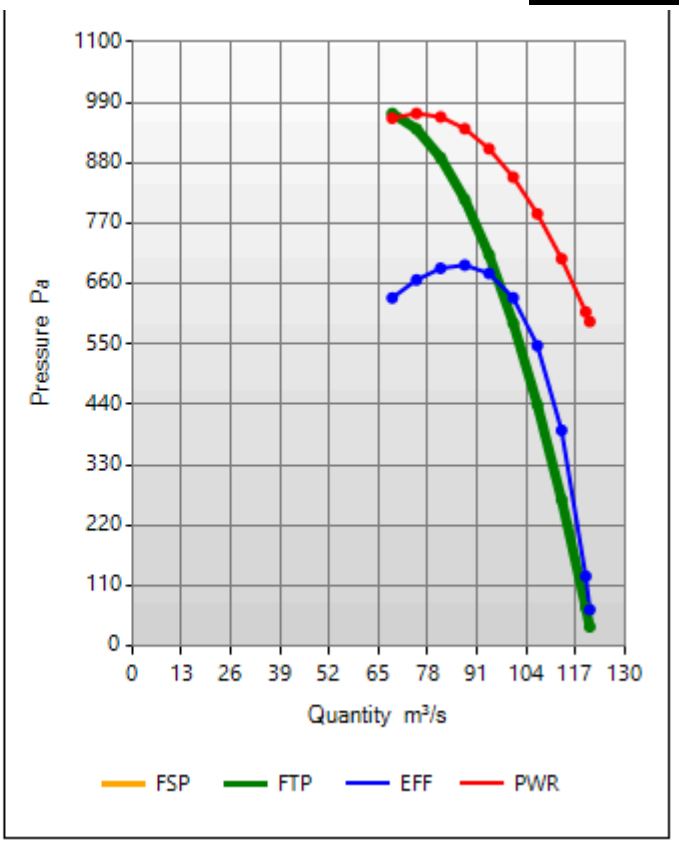


Figure 86. The Development Process Fan Curve

12.2 Methane Control

Coal-derived methane is methane gas formed and stored in coal seams. The carbonization process of plant-based organic materials is usually at 50 degrees, and methane occurs during the formation process. With increasing carbonization rate, the amount of methane gas also increases.

This methane, which is formed in coal seams, is found in cracks, fractures, pores, attached to the coal surface or dissolved in water (Gürdal ve Yalçın, 1992).

In Longwall mining, it is aimed to prevent fire pit explosions and absorb gas through boreholes of methane in the panels or on the side of the gob.

12.2.1 Drainage with Surface Drilled Holes

Methane mining is done through wells drilled by accessing coal seams from the surface. These holes, which are thrown into the coal seams, reduce the amount of methane by pulling the methane gas with the help of pump. Generally, 50-90% of the gas in the seam can be removed by the method (Hartman et al., 1997).

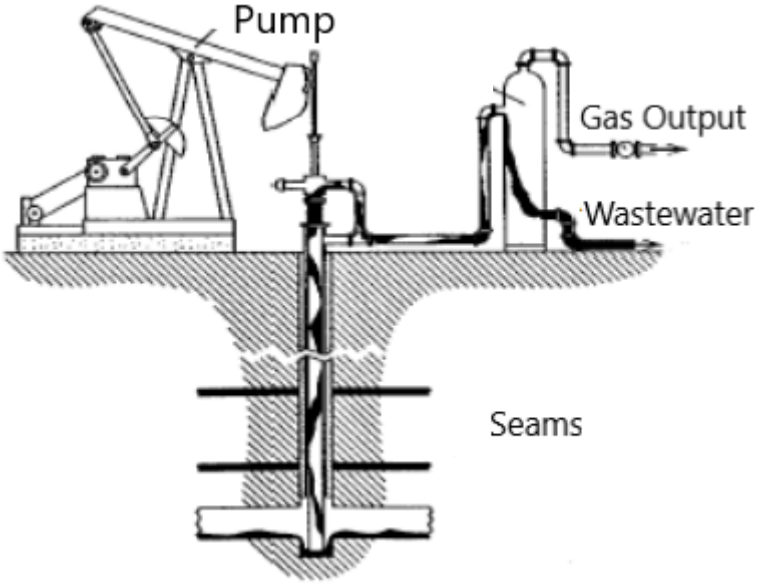


Figure 87. Vertical Drainage Drill (Hartman et al., 1997)

The total amount of methane produced varies depending on factors such as coal seam, gas content of the layer, permeability, drainage time, pump, and production system (Xue and Guo, 2003). With this method, the gas emission in the mine can be reduced and the extracted methane can be used commercially. The extracted methane has a high calorific value.

12.2.2 Drainage with Wells Drilled into the Gob

After the coal is cut by the shearer, methane is released in the gobs behind the hydraulic supports. The drills to be made are drilled behind the panels to be worked and up to the part that is expected to be collapsed. Methane is transported to the surface by pumps at the surface, and the methane obtained by this method can generally be used in coal cleaning,

drying coal, heating buildings, electricity production and internal combustion engines (Gatnar and Tor, 2003).

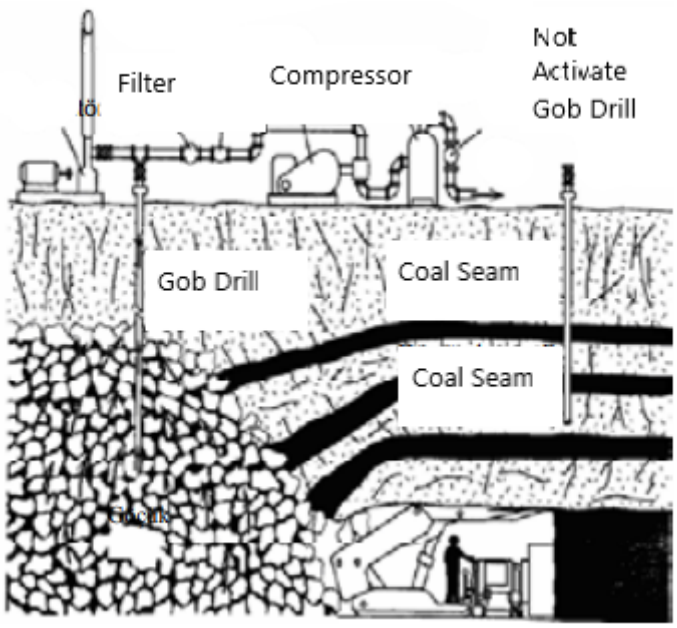


Figure 88. Gob Drill Design (Hartman et al., 1999)

Although it varies depending on geological conditions, a value between 30-70% of methane emission can be gained in general.

12.2.3 Drainage with Drills Drilled into Panels

While the panels are cut by the shearer in coal production, the methane emission can be reduced by drills that are drilled into the not yet produced areas at the back, a little shorter than the face dimensions. In this way, the potential of methane gas, which is expected to leak into the area when cut, is reduced.

Those smaller than 300 m work with 20%, and those larger than 300 m work with 20-40% efficiency. It has low efficiency due to its short drainage time, and although the amount of gas is low but has good quality.

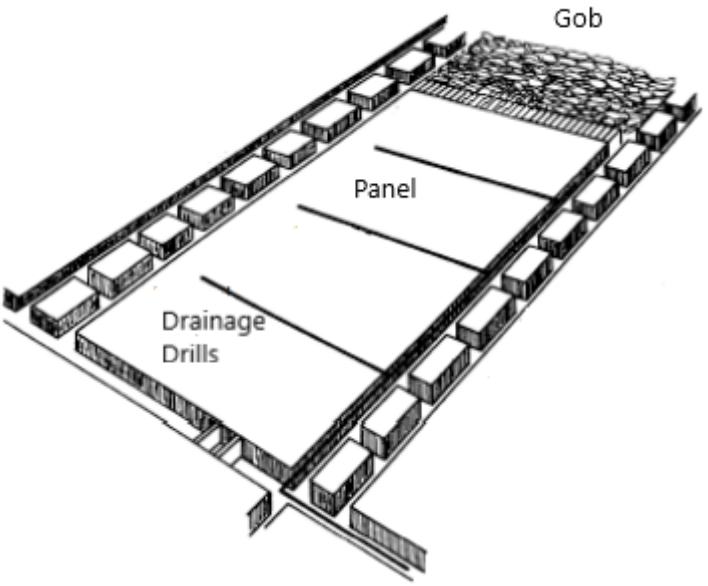


Figure 89. Drainage Drills inside the Panel (McPherson, 2004)

12.2.4 Methane Drainage with Cross Drills

It is a method of drainage of methane through the cross drills thrown into the collapsed areas through the openings in the mine. Horizontal holes are connected by a line that transfers the methane to the surface. Drainage efficiency is generally low (below 20%) and methane is impure (Schwoebel, 2001).

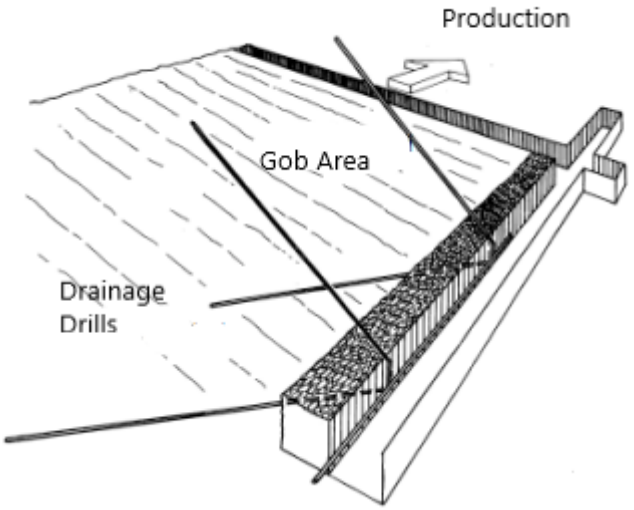


Figure 90. The Methane Drainage Method with Cross Drills at Gob (McPherson, 2004)

12.3 Dust Control

Dust protection is especially important for coal mines. Dust in small particles has the risk of colliding, sparking, and burning. This carries a high degree of risk. Different methods can be used to control these dust particles in coal mines. Water is the main influence in these methods.

As the first method, the most widely applied is water spray. Thanks to the water applied on the dust particles flying in the air, the dust particles precipitate, and their risk of colliding decreases.

The second method is coal seam water injection. The main source of dust in coal mines is the production part. Thanks to the water injected into the coal during the cutting phase of the coal, the proportion of dust particles decreases, and less dust is emitted to the area.

The third method is to use wet dust scrubber. Thanks to this equipment, in cases where the air equipment creates air flow in the environment, it draws the air in the environment, attracts the dust with its filters and wets it with the sprays inside. In this way, it reduces the dust concentration in the environment. (Zhou & Qin, 2021)

The technique of reducing the concentration by wetting the dust in the environment with water sprays will be applied. For this, water installations will be made to certain places in the region and water spray equipment will be used.



Figure 91. The water sprays applied in coal mine (Zhou & Qin, 2021)

The calculations and tables can be shown in the Appendix F.

13.MINE WATER DRAINAGE MINE-WIDE

In underground mines, the water can cause serious problems in operating. To operate the mine properly, the mine site's water drainage has great significance. The amount of the mine site's water is 105,000 m^3 per month.

$$Q = \frac{105{,}000\frac{m^3}{month}}{30\frac{day}{month}} = \frac{3{,}500m^3}{day} = required\ sump\ capacity$$

$$= \frac{3{,}500\frac{m^3}{day}}{24\frac{hr}{day}} = 146\frac{m^3}{hr}$$

$$= \frac{146\frac{m^3}{hr}}{3600\frac{s}{hr}} = 0.04\frac{m^3}{s}$$

Nominal pipe diameter for main pumps assumed as 11 inches = 279.4 mm

11 inches nominal pipe diameter = 11.75 inches actual diameter = 298 mm

Therefore, area (A) is,

$$A = \pi * \left(\frac{D}{2}\right)^2 = \pi * \left(\frac{0.298\,m}{2}\right)^2 = 0.0697m^2$$

$$V = \frac{Q}{A} = \frac{0.04\frac{m^3}{s}}{0.0697m^2} = \frac{0{,}57m}{s}$$

Nominal pipe diameter for submersible pumps assumed as 5 inches = 127 mm

5 inches nominal pipe diameter = 5.65 inches actual diameter = 143.5 mm

$$A = \pi * \left(\frac{D}{2}\right)^2 = \pi * \left(\frac{0.143\,m}{2}\right)^2 = 0.016m^2$$

$$V = \frac{Q}{A} = \frac{0.04\frac{m^3}{s}}{0.016m^2} = 2.5\frac{m}{s}$$

For 1 main Pump

Friction Loss (h_f)

$$\frac{1}{\sqrt{f}} = -\,2\log log\,\frac{\varepsilon}{3.7D}$$

ε is assumed as 0.2mm

$$\frac{\varepsilon}{D} = \frac{0.2}{298} = 6.7\times10^{-4}$$

f=0.13

$$h_f = f * \frac{L}{D} * \frac{V^2}{2g}$$

The Head for the main pump1 is 88 m.

$$h_f = 0.13 * \frac{88}{0.298} * \frac{0.57^2}{2\times9.81} = 0.636\,m$$

Minor loss:

$$h_m = \frac{k_m \times V^2}{2\times g}$$

$$h_m = \frac{1.1\times0.57^2}{2\times9.81} = 0.01821$$

Velocity Head:

$$Hv = \frac{V^2}{2\times 9.81}$$

$$Hv = \frac{0.57^2}{2\times 9.81} = 0.0165$$

Total Head = Velocity Head (Hv) +Friction Loss + Minor Losses + Elevation Head

Total Head = 0.01821+0.0165+0.63+88 = 88.67m

The duty of the submersible pumps is to transfer the 36.5 m^3 water to the sumps. Therefore, their capacity should be at least 36.5 m^3.

To calculate the power of the pump, the pressure must be calculated first. Since the total head of pump 1 is 90.77m, pressure is calculated relative to the total head.

$$Hp = 88.67m\times 1000\times 9.81 = 869,857\ Pa$$

$$Power = \frac{Q\times H_P}{1000\times Efficiency}$$

The efficiency of the pumps is taken as %65.

$$Power = \frac{0.04\frac{m^3}{s}\times 869857}{1000\times 0.65} = 53.5\ Kw$$

According to calculations, the main pumps' total heads and powers are calculated and listed in Table 24.

Table 24. Total Head and Power of Main Pumps

Main Pumps	Total Head (m)	Power of Pumps (kW)
Pump 1	89	54
Pump 2	90	54
Pump 3	242	145
Pump 4	95	58
Pump 5	88	53
Pump 6	377	228
Pump 7	357	216
Pump 8	246	149
Pump 9	221	134
Pump 10	368	223
Pump 11	96	58

There are 4 submersible pumps at the end of the working panels to transfer the water to the sumps.

The total heads and powers of submersible pumps are calculated and listed in Table 25.

Table 25. Total Head and Power of Submersible Pumps

Submersible Pump 1	Total Head (m)	Power (kW)	Flow Rate $\frac{m^3}{hr}$
Submersible Pump 1	14.5	21	73
Submersible Pump 2	14.5	21	73
Submersible Pump 3	14.5	21	73
Submersible Pump 4	14.5	21	73

The mine site has 11 pools that have different capacities. The pools are classified as Pool1, Pool2, etc. The capacities of the Pools are listed in Table 27.



Figure 92. Submersible pump (Dalgıç Pompa, 2022)

Submersible Pumps are worked for 12 hours per day.

13.1 Sump Design

There are 11 sumps to collect the water. The level of the water is controlled by sensors located in sumps. The sensors make pumps work when the water level reaches 146 m^3. it is pumped sump to sump. The water is pumped to the surface by sump 9 and sump 11. There are two types of sumps in terms of their capacity. The dimensions of two types of sumps are listed in Table 26

Table 26 Dimensions of Sumps

Sumps	Height (m)	Length (m)	Width (m)	Capacity (m^3)
Sump 1	7	10	5	350
Sump 2	5	8	5	200

In Table 27, the capacities of all sumps are listed.

Table 27. Capacities of the Sumps

Sump Names	Sump Capacities (m^3)
Sump 1	$200m^3$
Sump 2	$200m^3$
Sump 3	$200m^3$
Sump 4	$200m^3$
Sump 5	$350m^3$
Sump 6	$350m^3$
Sump 7	$350m^3$
Sump 8	$350m^3$
Sump 9	$350m^3$
Sump 10	$350m^3$
Sump 11	$350m^3$

According to the total head and flow rate, the pumps are selected. The selection of the pump is shown in Figure 93.

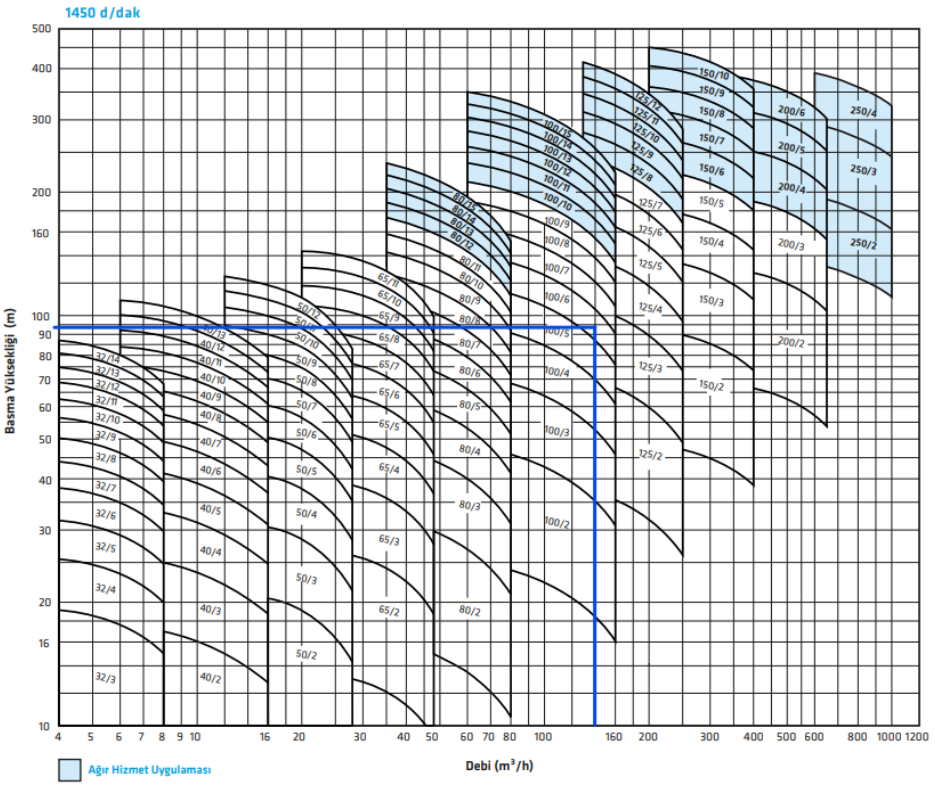


Figure 93. Pump 1 Selection

73 m^3 water is pumped to the Pump1, Pump2, Pump3, and Pump4 by submersible pumps per 2 hours. The water is collected in sumps. After the water level is reaching 146 m^3. The water is pumped to the higher sumps. The working period, head of pumps and model of pumps are listed in Table 28.

Table 28 Working Period of Pumps

Pumps	Flow Rate m^3/h	Head of Pumps	Model of Pump	Working Period (hr)
Pump 1	146	89	SKM –K 100/6	6
Pump 2	146	90	SKM –K 100/6	6
Pump 3	146	241	SKM –K 125/8	6
Pump 4	146	96	SKM –K 100/6	6
Pump 5	292	88	SKM –K 150/3	6
Pump 6	292	378	SKM –K 150/10	6
Pump 7	292	356	SKM –K 150/9	6
Pump 8	292	247	SKM –K 150/6	6
Pump 9	292	222	SKM –K 150/6	6
Pump 10	292	369	SKM –K 150/10	6
Pump 11	292	96	SKM –K 150/3	6

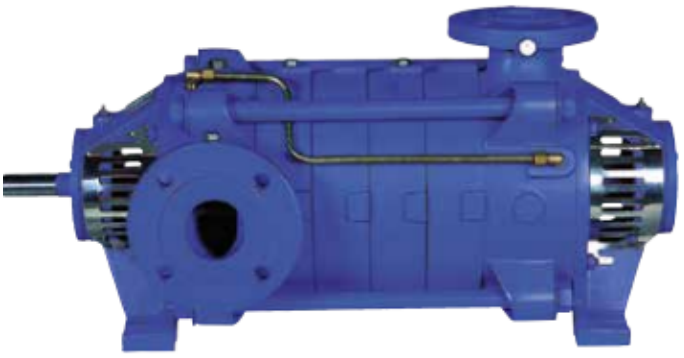


Figure 94. Standart Pump SKM - K 100 /6 (Standart Pompa, 2022)

The location of sump 1 and sump 2 in the development design is shown in Figure 95.

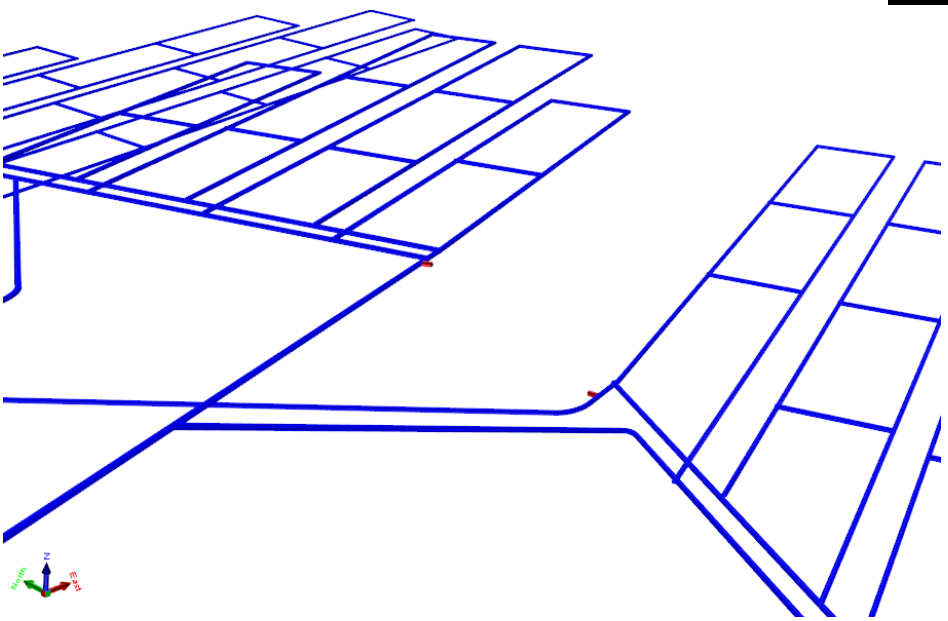


Figure 95. Location of Sump 1 and Sump 2

14.MONITORING AND ENVIRONMENTAL CONSIDERATIONS

The usage of mine monitoring systems is growing day by day because these systems help create a safe environment for the mine site, increase efficiency, and control the operation. Therefore, the mine monitoring system is one of the most significant issues in the mining industry. The data is collected from the operation site, nonworking areas, and ore accesses by mine monitoring systems. These data help to take immediate action for safety if it is necessary before it is too late or to increase efficiency. Therefore, it is critical to have mine monitoring systems on the surface to monitor the mine site.

14.1 Mine Monitoring for Safety

For a safe mining environment, there are lots of parameters to control and monitor. These parameters are listed below:

- Mine Air
- Noise
- Mine Water Monitoring
- Conveyor Belt Monitoring
- Dust
- Temperature and Humidity
- Personal Equipment

14.2 Mine Air Monitoring

According to the underground mine regulations in Turkey, mine climate, fresh air, and gas content of the mine are needed to be regulated. Since the mite site is a living place, the conditions of air can change, therefore the monitoring system should work continuously and must consistently demonstrate the accuracy of the values. The releasing of harmful gases and threshold limit values (MAC) are the monitoring factors for mine air quality.

The limits of MAC values are limited by MSHA (Mines Safety and Health Administration). In Table 29**Error! Reference source not found.**, the minimum acceptable value of the oxygen content, the maximum acceptable value of the carbon dioxide, and the maximum content of the other gases are shown Table 29.

Table 29 Mine Gases and MAC Values (T.C Çalışma ve Soyal Güvenlik Bakanlığı, 2015)

Mine Gases	MAC Values
Oxygen (O_2)	19,50%
Carbon Dioxide (CO_2)	0,50%
Hydrogen (H)	0,8
Carbon Monoxide (CO)	2,50
Hydrogen Sulfide (H_2S)	0,8
Acetylene (C_2H_2)	0,4
Propane (C_3H_8)	0,4
Sulfur Dioxide (SO_2)	5,0 ppm
Methane (CH_4)	1

To monitor the air and gas contents, air monitoring devices and gas ambient air alarm analyzers are used. The monitoring devices include photoionization detectors and flame ionization detectors, single gas, meters, and multiple gas detectors. The gas detectors are shown in Figure 96 Figure 97.



Figure 96.Drager Polytron 7000 (Drager, 2022)



Figure 97.Frager X-am 8000 (Drager, 2022)

While Drager Polytron 7000 is a steady sensor, Frager X-am 8000 is a portable sensor to detect continuously when moving.

When the gas contaminations are at dangerous levels, alarms warn. The level of alarm of sensors is listed in Table 30.

Table 30. Level of Alarm of Sensors

Gas Formula	Gas Name	1. Alarm	2. Alarm
CO	Carbon Monoxide	24 ppm	60 ppm
CH_4	Methane	1 %	1.5 %
O_2	Oxygen	19 %	18%
CO_2	Carbon Dioxide	0.3 %	0.5%
H_2S	Hydrogen Sulfide	10 ppm	20 ppm

If the gas level reaches the alarm level, some actions are needed to be taken. For example, if the content of methane exceeds 1.5 percent, the electricity is shut down.

14.3 Noise Monitoring

The noise level should be controlled in the underground mines for good working conditions. According to the regulations, the available noise level is 85 dB for the workers. The maximum exposure time for workers for 85 dB is 8 hours. Since the noise is created by mine machines, the noise level must be controlled and detected. To detect the noise level in the underground, TES -135A sound level meter is used. The device is demonstrated in Figure 98.



Figure 98. TES - 135A Sound Level Meter (TES Electrical Electronic Corp, 2022)

14.4 Mine Water Monitoring

Mine water monitoring is one of the main progresses in underground mines to continue the operations. Water flow should be organized very well for the ramp, decline, and shaft. In this project, coal seams are highly deep and that is a complex operation for the dewatering process. The efficiency of the equipment can be lower with a high level of water. To prevent this problem, mine water drainage systems, sumps, pumps, and water pools are designed with the most efficient system. There are different water collecting points thanks to the elevation. The underground water is transferred to the surface step by step using that water collecting points with the sumps and pumps system. Also, the flow rate of water is controlled properly, and precautions are taken to prevent highly risky situations. In the sumps, the water is collected. When the water level reaches the desired level, the sensors are warned, and the water is pumped to a higher level.

14.5 Conveyor Belt and Haulage Monitoring

The conveyor belt provides the haulage of the material. The monitoring of the conveyor belt system has great significance for efficient production. Thanks to the monitoring system, production can be controlled instantaneously.

The monitoring of haulage system and conveyor belt system is important to check the production efficiency and prevent dangerous situations. The production can be followed from haulage daily by using monitoring techniques. The materials falling from the conveyor should always be under control. Monitoring of this equipment can avoid any injury or fatality during the operations.

14.6 Dust Monitoring

Dust is one of the biggest problems in underground coal mining operations. Therefore, it has to be controlled and eliminated. Before eliminating the dust, it should be measured. To measure the dust content, a coal dust explosibility meter can be used. The device is shown in Figure 99. According to the regulations, the maximum allowable content (MAC) of dust is 2mg/m^3 and %5. There are methods to eliminate the dust from the working area. They are usage of water sprays and providing sufficient ventilation to the working area.



Figure 99. Coal Dust Explosibility Meter (Technology News, 1997)

14.7 Temperature and Humidity Monitoring

In underground coal mines, temperature is one of the most significant parameters to monitor. Because coal is a flammable commodity by time. Therefore, it has to be controlled, monitored, and eliminated. Moreover, due to the equipment, underground rock, and metabolism of workers, the temperature of the working area is increased. The optimum temperature is 20 °C to create a suitable working condition for workers. To provide 20 °C for the working area, ventilation must be done concerning that.

Before providing optimum temperature and humidity, it must be monitored, and then action is taken. To monitor temperature and humidity, sensors are used.

14.8 Personal Safety Equipment Monitoring

Mining is one of the most dangerous jobs in the world. Therefore, the safety of the workers must be provided not to lose a life and cause any injury in the working area. To provide and follow safety for workers in the underground, the workers must be tracked. In Figure 100, the worker tracking system is shown. For the tracking system, the fibre line is connected through the mine site.

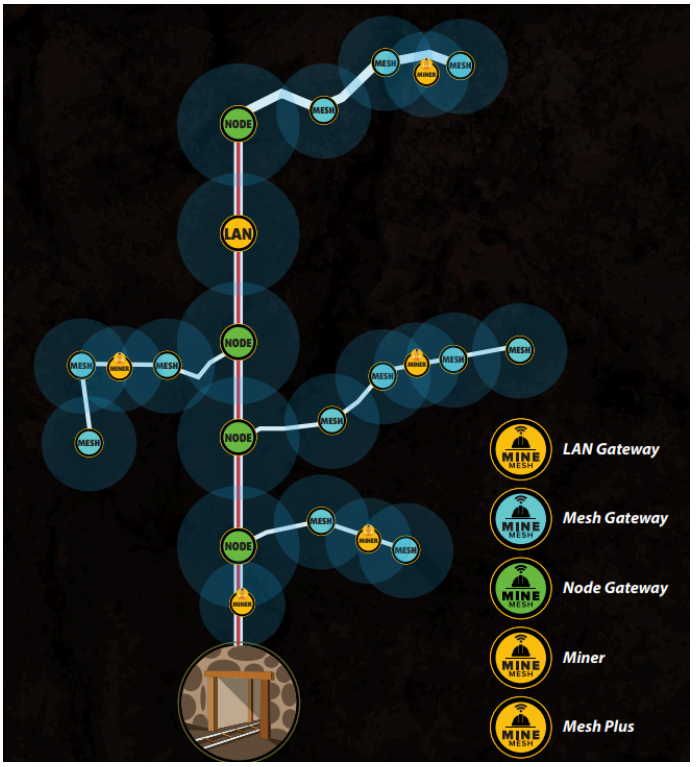


Figure 100 Underground Mining Tracking System

According to the regulations (T.C Çalışma ve Soyal Güvenlik Bakanlığı, 2015), underground workers must have their oxygen masks in case of dangerous situations. In Figure 101, the mask is shown.



Figure 101 Drager Oxy K 30 H (Drager, 2022)

To prevent dust exposure of workers, the protective masks are used. In Figure 102, the protective mask is shown.



Figure 102. Dräger X-plore® 1900 (Drager, 2022)

14.9 The Monitoring of Mining Activities

Monitoring is important for any type of job and should be checked for the effectiveness of the overall workflow. Both the transportation system, the production part, the human tracking and the equipment tracking should be continuously controlled via online data. In this way, any risky situation can be avoided. All of these incoming data can be collected at one point and the workflow in the mine can be regulated. Data can be kept in the database program, which is under the supervision of authorized people, and historical records can be made, and analyzes can be made by accessing the desired documents, and it is of great importance in terms of sustainability. Mining activities are controlled by Mine Mesh system as shown in Figure 100. (Mine Mesh, 2022)

14.10 The Monitoring of Waste Dump

To drive development openings, the waste material must be extracted. The extracted waste material is dumped into the waste dump. In the waste dump, slope failure can cause serious injuries. Therefore, the slope stability of the waste dump must be monitored. To monitor slope stability, IBIS - FL georadar is used. It is shown in Figure 103.



Figure 103 IBIS-FL georadar (IDS GeoRadar, 2022)

15.ELECTRIC SUPPLY

Electricity is needed in all mining activities, regardless of surface facilities (main buildings, road illumination and launder) or underground (drilling, hoisting, ventilation and underground illumination). Electricity must be supplied regularly and adequately for the continuity of operation. The electricity line will be provided by establishing a new line from the nearest village to the mine site.

15.1 Power Requirements

The equipment that uses electricity determined the power requirements. Table 31 contains the equipment list as well as the daily electricity generation. Other facilities that operate on the surface should be considered when calculating electricity consumption.

Table 31 General Power Consumption

Equipment	Number	Power(kW)	Working Hours(hr)	Daily Power(kWh/day)
Ventilation Shaft Fan	1	400	24	9600
Main Pump	11	-	6	8230
Sump Pump	4	-	24	1980
Shearer	1	1250	20	25000
Belt	-	-	-	5000
Illumination	-	-	24	1000
Surface Facility	-	-	-	3000
Total				53810

15.2 Illumination

Lighting in underground mining must be done in accordance with the regulation because the risk of occupational health and worker accident should be reduced and a safe working environment must be created. All mine site ramps, stopes, and drifts have illumination provided 24 hours a day. Since we will include 3 different seams in the operation at different times, we do not have to light the other 2 seams. Lighting intensities according to working places in underground mining enterprises are given in Table 32.

Table 32 Lighting intensity values according to working places in Mining Operations (Çınar and Şensöğüt, 2017)

Locations	Average Luminous Power, Lux		
	General Lighting	General Lighting with Additional Lighting	
		General Lighting	Additional Lighting
Workshop	120	60	250
Well excavation site, Crushing facilities, Winch room, Warehouses, Pump room, Gallery face, Switch/Transformer room, Redresor room	80	40	150
Main Well Floorheads, Stations	40	-	-
Conveyor transfer points, Filling machines, Large tanks, Loading areas	30	-	-
Gallery intersections, Staff stations, Shears	15	-	-

16.COMPRESSED AIR

In underground mines, air compressors are used as auxiliary equipment. The compressors will primarily power pneumatic manual rock drills. Even though many mining machines have their own air compressors and air compressors are required in mining workshops for tasks such as engine cleaning and wheel repair. When inflating the wheels of mining equipment, high pressure is required.

Compressed air tank capacity is selected according to pneumatic manual rock drills used. According to the Toyo TY24C catalog, each TY24C needs a maximum air pressure of 0.63 MPa. 5 pieces of TY24C will be used. Therefore, a compressed air tank needs to supply this equipment.

Table 33 Compressed air tank catalog (W.W. Grainger, 2022)

Horizontal Stationary Air Tanks



Horizontal stationary air tanks have a low profile and can fit under work spaces. They are permanently installed where they can store air for an air compressor or expand the capacity of an existing system. These air tanks can hold high capacities of air without the risk of tipping.

↑Tank Capacity	Maximum Operating Pressure	Overall Length	Overall Width	Overall Height	Standards	Brand
2 gal	175 psi	19 in	6 in	8 in	——	SPEEDAIRE
3 gal	175 psi	25 in	6 in	8 in	——	SPEEDAIRE
6 gal	175 psi	20 in	10 in	12 in	ASME U Stamp	SPEEDAIRE
10 gal	175 psi	24.5 in	12 in	14 in	ASME U Stamp	SPEEDAIRE
15 gal	175 psi	34 in	12 in	14 in	ASME U Stamp	SPEEDAIRE
20 gal	175 psi	32.5 in	14 in	16 in	ASME U Stamp	SPEEDAIRE
30 gal	175 psi	37.5 in	16 in	18 in	ASME U Stamp	SPEEDAIRE
30 gal	200 psi	39.37 in	18 in	23 in	ASME U Stamp	SPEEDAIRE
30 gal	200 psi	41.1 in	16 in	21.7 in	ASME U Stamp; ASME UM Stamp	SPEEDAIRE
30 gal	200 psi	41.5 in	16 in	20.46 in	ASME U Stamp	SPEEDAIRE
60 gal	200 psi	46.75 in	20 in	27.05 in	ASME U Stamp	SPEEDAIRE
60 gal	200 psi	48 in	20 in	27.6 in	ASME U Stamp; ASME UM Stamp	SPEEDAIRE
80 gal	200 psi	61.75 in	20 in	25.65 in	ASME U Stamp	SPEEDAIRE
80 gal	200 psi	63 in	20 in	27.4 in	ASME U Stamp; ASME UM Stamp	SPEEDAIRE
120 gal	200 psi	67.5 in	24 in	30.52 in	ASME U Stamp	SPEEDAIRE
120 gal	200 psi	68.4 in	24 in	31.9 in	ASME U Stamp; ASME UM Stamp	SPEEDAIRE

17.MINE LAYOUT

The mine is designed by D | D Co. The mine has 3 underground entrance, main ramp, conveyor belt ramp, and shaft, waste dump, cafeteria, administration building, ore dump, explosive storehouse, cement plant, dorm, and workshop.

The explosives for blasting is stored in the explosive storehouse. Since explosives are highly dangerous, they are stored away from the surface facilities.

The waste material will only come out in development work. The amount of waste material that will come out from development is about 750,000 m^3. Based on this the waste dump is designed. The waste Dump has 50 meters' height and 30° slope angle.

The mine layout is shown in Figure 104.

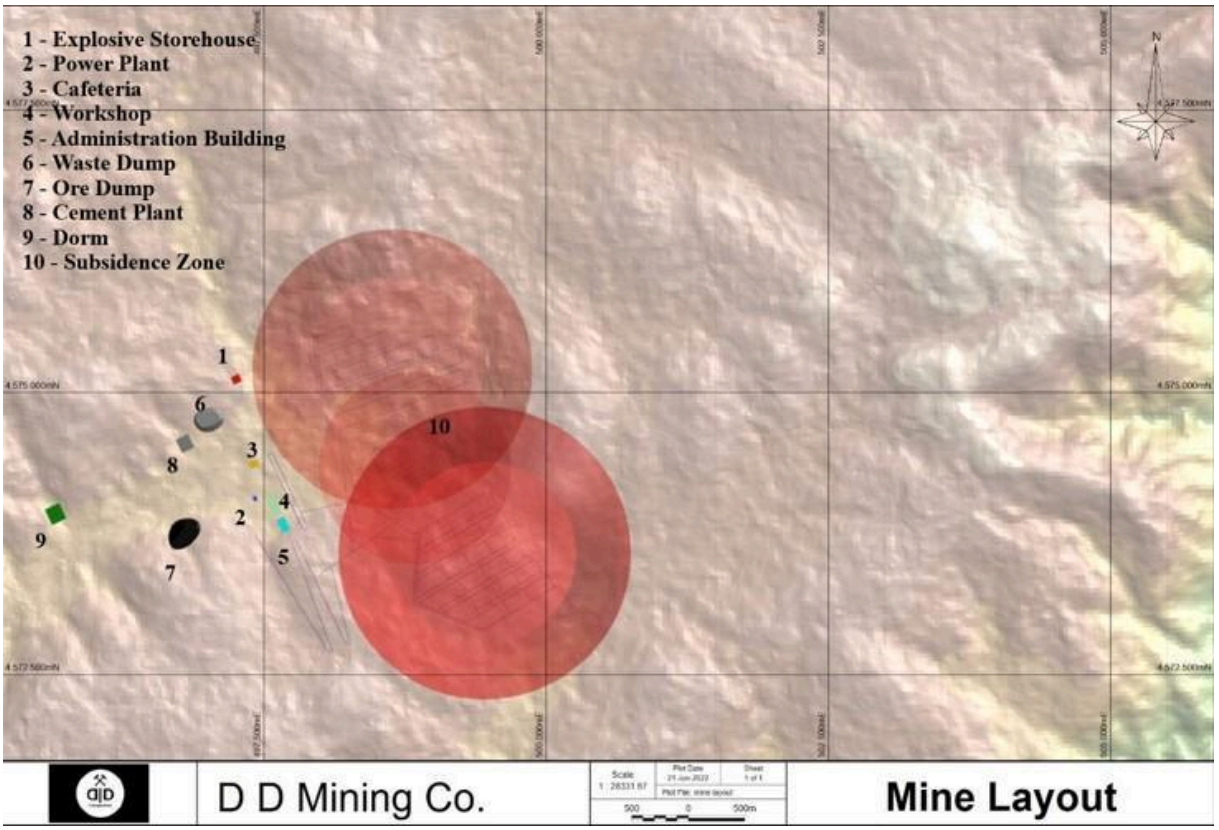


Figure 104.Mine Layout

18.MINE CLOSURE AND RECLAMATION PLANNING

The basic purpose of reclamation is to recover impacted areas to their original economic and ecological value as much as feasible. Closing planning, dismantling of the mine site, regeneration, and continuing monitoring are all part of the mine closure process. Reclamation processes are mainly carried out after the mine is closed. however, the planning and development work for reclamation starts from the opening of the mine. Therefore, progressive mine closing will be applied.

Sealing of Underground Openings

Mine entrance sealing is used as a closure method in underground mines to prevent and reduce mine drainage, as well as to reduce disulphide oxidation. Sealing abandoned mine workings may help prevent the release of harmful gases. Mines are frequently sealed for safety reasons to prevent illegal access to potentially dangerous open voids. (The Pennsylvania Department of Environmental Protection, 1998) During the rehabilitation process, it is important that the region be surrounded by a suitable material such as wire mesh. Moreover, in areas where humans and wildlife may be endangered so, warning signs placement should be done. Additionally, because there is a possibility of someone falling through or entering these openings, they must be closed. Mine shaft is very dangerous since it is a giant hole with a length of about 900 meters and a diameter of 5 meters. Therefore, shaft must be closed to prevent subsidence, reduce the dangers of living, working, or playing near a mine shaft. Also, when necessary, to protect cultural resources. Filters with non-acid forming, unrestricted materials must be used to fill shaft.

Removing the Surface Facilities

To begin land development activities, some surface utilities must be removed. After certain amenities that can be used while undertaking reclamation operations provided, other facilities must be removed from the mining site. After contacting the local people, the buildings used in the project such as offices and dorms can be left to the local people. If buildings are unusable or not needed by local people, demolition will take place. In other words, if the buildings are suitable for use and the area is not dangerous, they can be allocated to the local people.

Reclamation

The area affected by underground mining is not as much as surface mining. However, there are many issues to be considered in the reclamation of underground mining. Therefore, the reclamation process will be carried out considering the EIA report prepared and taking the opinion of the local people. The reclamation process will begin once all underground entrances are sealed, and the area is fully secured. As stated in the previous sections, the mining method used in the project is LTCC. In this method, surface subsidence is expected. However, this settlement is not at a level to be dangerous. Nevertheless, the places where the most settlement is observed will be backfilled and the area will be afforested with the tree species found in the area.



Figure 105. Reclaimed Area (Xiao, Hu, & Fu, 2014)

Environmental Laws and Regulations

Every situation that will affect the health of the public and nature is regulated by the Turkish government with regulations. Environment Law (1983) and Regulations about Mine Closure and Reclamation (2011) are the most significant regulations. Other regulations are as follows:

Mining Law (Law No: 6592) (2015)

Occupational Health and Safety (Law No: 6331) (2012)

Regulation on Labor Hygiene Measurement, Test and Analysis Laboratories (Law No: 28717) (2013)

Regulation on The Prevention and Reducing Effectiveness of Great Industrial Injuries (Law No: 28867) (2013)

Regulation on the Use of Personal Protective Equipment in Workplaces (Law No: 28695) (2013)

Regulation on Vocational Trainings to be operated at the Hazardous and High Dangerous Class (Law No: 28706) (2013)

Regulation on Health and Safety Signs (Law No: 28762) (2013)

Regulation of Struggle with Dust (Law No: 28812) (2013)

Regulation on the Protection of Risks to the Noise of Employees (Law No:28721) (2013)

Regulation on the Protection of Explosive Environments from Hazardous Environments (Law No: 28633) (2013)

Regulation on the Protection of the Employees Relating to Vibration (Law No:28743) (2013)

Environmental Impact Assessment Regulation (2005)

Regulation on Control of Air Pollution from Industry (Law No: 25606) (2004)

Regulation on Control of Soil Pollution (Law No: 25831) (2005)

Regulation on Control of Solid Waste (Law No: 20814) (1991)

Regulation on Control of Hazardous Waste (Law No:25755) (1983)

Regulation on Control of Water Pollution (Law No: 25687) (2004)

Assessment of Management of Environmental Noise (Law No: 25862) (2005)

Regulation on Wildlife Protection and Improvement (Law No:25637) (2004)

Regulation on Waste Management (Law No: 29314) (2015)

Regulation of Natural Resources with Mining Activities Regarding Nature (Law No:27471) (2010)

Principles for Establishing a Balance between Mining and Environment

The main factor that establishes the balance between the environmental and mining concepts is the Environmental Impact Assessment (EIA). A positive EIA report has been reviewed by government officials and shows that environmental damage has been minimized. This creates the bridge between the environment and mining. A well-planned mine closure phase is important for both nature and the local community and will be done in accordance with all regulations. In the mine closure plans to be applied, all social, environmental, and economic factors will be taken into consideration.

Independent Environmental Management

To have more detailed information about the environment and to reveal possible potential hazards, an Environmental Baseline Study is conducted in the environment before the project. This study includes climatic conditions in the region, development of the business plan, preliminary investigation, investigation of strategy, field investigation, selection of soil and groundwater quality standards and reporting. In this way, information such as wildlife, weather, soil samples, geological data, water resources in the region are obtained, thus minimizing potential risks, and providing environmental management.

Communication

More than one responsible party should work together and be in harmony to use the mine site and area in a safe and healthy way and to avoid any problems in operations. In healthy communication, the government, D & D Corporation, and community will always be united in communication and will produce solutions together for the problems that may arise. In this way, stakeholders and the company will be more transparent, and trust will be built in the region. These factors are also necessary to start mining activity.

18. ECONOMIC ANALYSIS

In this coal project, economic analysis is divided the 6 submodels. The submodel gave below;

1. Production Sizing
2. Equipment and Construction
3. Manpower
4. Supplies, Materials, and Power
5. Investment Requirement
6. Financial Analysis

This economic analysis has been prepared according to the coal mining cost model handbook.

18.1 Production Sizing

In D | D coal mine project, the coal deposit is 130 million tonnes. However, it is not possible to get all of them due to the method we used. At the end of the project, 14.3 million tons of coal will be extracted.

$$\frac{14.3\ million\ tons\ of\ Coal}{12{,}5\ Years} = 1.144\ million\ tons\ per\ year$$

Coal production in the project will take 12.5 years. In the above equation, annual coal production is seen. In this project, it is argued that production will be made 350 days a year.

$$\frac{1.144\ million\ tons\ per\ year}{330\ working\ day\ per\ year} = 3500\ tons\ per\ day$$

The work will be 3 shifts. The planned production in each shift is given below.

$$\frac{3500\ tons\ per\ day}{3\ Shift\ per\ day} = 1170\ tons\ per\ shift$$

18.2 Equipment and Construction

The material requirement of our coal project is determined according to the needs of production capacities and geological conditions. In part 10, The equipment planned to be used as mentioned. The total cost of equipment is about 60 million dollars. The cost of this equipment and its quantities can be seen in Table 34.

Table 34 Equipment List and Their Cost

Equipment	Brand	Quantity	Cost
Load-Haul-Dump	Epiroc Scooptram ST2D	2	$ 750,000
Truck	CAT AD63 mining truck	4 (2)	$ 720,000
Jumbo	Epiroc Boomer L2	2	$ 2,000,000
Road Header	CTR300A	3 (1)	$ 10,000,000
Pneumatic Manual Rock Drill	Toyo TY24C	5	$ 20,000
Shearer	CAT EL2000	1	$ 5,000,000
Hydraulic Roof Support	CAT AEHQ6801	80	$ 28,000,000
Armored Face Conveyor	CAT PF3	1	$ 1,500,000
Conveyor	Pylon Plus 1000/5	5250 m	$ 200.000
Roof Sealing Machine	SCALER S320	2	$ 1,000,000
Mixer	COGUARO 4	4	$ 1,500,000
Shotcrete	Dingo from CITA	2 (1)	$ 400,000

Concrete Station	Mobil Mix 30	1	$ 100,000
Bolting	Sandvik DD422iE	2	$ 1,000,000
Dewatering Pumps	Standard Pump SKM - K 100 /6	11	$ 120,000
Fire Truck	Mercedes AROCS Fire Truck	2	$ 300,000
Ambulance	Mercedes Sprinter Ambulance	2	$ 150,000
Personal Vehicles	Mercedes Sprinter Service	10	$ 500,000
Off-road Cars	Toyota Hilux	8	$ 700,000

A budget of approximately 6 million dollars has been allocated for the construction works to be carried out on the surface of the D&D Coal Project. This budget will be used for the construction of structures such as administrative buildings, changing rooms, and guesthouses.

18.3 Manpower

D | D Coal Project's labor requirements are divided into 8 categories. The following tables show the positions and headcounts. The labor requirements near mines are used to determine headcounts.

Mining staff requirements can be seen in Table 35. The total number of mining staff requirements is 373.

Table 35. Mining Staff Requirement

POSITION	HEADCOUNT
MINING OPERATORS	60
MINE WORKERS	240
MINE MANAGER	1
MINE SUPERINTENDENT	1
MINE ENGINEER	24
GEOLOGY	6
GEOTECHNICAL ENGINEER	6
SURVEYOR	3
TECHNICIAN	30

ASSISTANT	2

The maintenance staff requirement can be seen in Table 36. The total number of maintenance staff requirements is 86.

Table 36. Maintenance Staff Requirement

POSITION	HEADCOUNT
MAINTENANCE MANAGER	1
MAINTENANCE SUPERINTENDENT	1
PLANNER	6
FORMEN	18
MECHANIC	40
ELECTRICIAN	20

General and administrative staff requirements can be seen in Table 37. The total number of general and administrative staff requirements is 11.

Table 37. General and Administrative Staff Requirement

POSITION	HEADCOUNT
GENERAL MANAGER	1
OPERATION MANAGER	1
FINANCE DIRECTOR	1
EXTERNAL RELATION DIRECTOR	1
ADMINISTRATION MANAGER	1
OFFICE ASSISTANT	3
TECHNICAL ASSISTANT	3
ACCOUNTING STAFF	3

Human resource staff requirements can be seen in Table 38. The total number of human resource staff requirements is 8.

Table 38. Human Resource Staff Requirement

POSITION	HEADCOUNT
HUMAN RESOURCE DIRECTOR	1
HUMAN RESOURCE SUPERVISOR	1
TRAINING SUPERVISOR	3

TRAINERS	3

Finance and accounting staff requirements can be seen in Table 39. The total number of finance and accounting staff requirements is 3.

Table 39. Finance and Accounting Staff Requirement

POSITON	HEADCOUNT
ACCOUNTING	2
PAYROLL	1

Purchasing and warehouse staff requirements can be seen in Table 40. The total number of purchasing and warehouse staff requirements is 15.

Table 40. Purchasing and Warehouse Staff Requirement

POSITON	HEADCOUNT
MATERIALS & PURCHASING SUPERVISOR	1
WAREHOUSE SUPERVISOR	1
WAREHOUSE	8
PURCHASING SUPERVISOR	1
BUYER	4

Occupational health and safety staff requirements can be seen in Table 41. The total number of occupational health and safety staff requirements is 11.

Table 41. Occupational Health and Safety Staff Requirement

POSITON	HEADCOUNT
OHS DIRECTOR	1
HSE MANAGER	1
OHS SUPERVISOR	1
OHS TECHNICIAN	4
DOCTOR	1
NURSE	3

Environmental staff requirements can be seen in Table 42. The total number of environmental staff requirements is 6.

Table 42. Environmental Staff Requirement

POSITON	HEADCOUNT
ENVIRONMENTAL DIRECTOR	1
PERMITTING ENGINEER	1
ENVIRONMENTAL ENGINEER	1
ENVIRONMENTAL TECHNICIAN	3

The entire labor required, as shown in the tables above, is around 513 workers. This number can be arranged based on the rate of operation.

18.4. Supplies, Material, and Power

Chapter 15 calculates the energy requirements of the D&D Coal Project. According to such estimations, the daily energy demand is estimated to be 53810 kW. This will set you back approximately $2,000,000.

18.5. Investment Requirement

An initial investment in the project is determined by using the cost of equipment that is discussed in Chapter 17.2. In addition to this cost, there will some other costs such as equipment installation, piping, electrical, instrumentation, etc. These costs are related to the cost of equipment and their cost can be seen in Table 43 (Gölbaşı 2020). According to these table total required investment for mining operation is around 150,000,000 $.

Table 43. Total Initial Costs for Mining Operation

CAPITAL COST ITEM	COST
Purchased Cost	$ 60,000,000
Equipment Installation	$ 12,000,000
Piping	$ 600,000

Electrical	$ 12,000,000
Instrumentation	$ 4,800,000
Building	$ 6,000,000
Services	$ 6,000,000
Site Development	$ 6,000,000
Construction Management	$ 6,000,000
Engineering and Construction	$ 18,000,000
Fixed Capital Cost	$ 131,400,000
Contingency	$ 19,000,000
Total Fixed Capital Cost	**$ 150,000,000**

17.6. Financial Analysis

The D | D Corporation Coal project is main parameter is yearly production for financial analysis. The annual production was determined as 1.144 million tons as a result of geotechnical studies. The other parameter which affects directly is coal price. The coal price is taken as 155 $/metric ton. Near-mine reports are used to calculate operating costs. According to the Mining Law of Turkey royalty is taken as 3.75%. According to Turkish tax law, the tax rate is set at 22%. The straight-line method is used to determine depreciation amounts. The capital cost of the project is about 150,000,000 $. This amount will be provided by bank loan with an interest rate of 5%. First 4 years, no payment will be made. The remaining years will be paid equally and at $22 million. All these calculation and net cash flow for each can be seen in Table 61 .

Table SEQ Table * ARABIC61. Yearly Costs, Revenue, Profits and Net Cash Flows

Year	-	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	Total
Investment Cost ($)					$ -	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 22.000.000	$ 286.000.000
Coal production (tons)		-	-	-	-	576.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	1.144.000	14.304.000
Waste Production (tons)		303.000	303.000	303.000	303.000	150.000	-	-	-	-	-	-	-	-	-	-	-	-	1.362.000
Revenue ($/year)		$ -	$ -	$ -	$ -	$ 89.280.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 177.320.000	$ 2.217.120.000
G&A cost ($/year)		$ 606.000	$ 606.000	$ 606.000	$ 606.000	$ 1.452.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 2.288.000	$ 31.332.000
Waste mining cost ($/year)		$ 9.090.000	$ 9.090.000	$ 9.090.000	$ 9.090.000	$ 4.500.000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 40.860.000
Coal mining cost ($/year)		$ -	$ -	$ -	$ -	$ 8.640.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 17.160.000	$ 214.560.000
Rehabilitation cost ($/year)		$ 45.450	$ 45.450	$ 45.450	$ 45.450	$ 108.900	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 171.600	$ 2.349.900
Total cost ($/year)		$ 9.741.450	$ 9.741.450	$ 9.741.450	$ 9.741.450	$ 36.700.900	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 41.619.600	$ 575.101.900
Royalty ($/year)		$ -	$ -	$ -	$ -	$ 21.600	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 42.900	$ 536.400
Depreciation ($/year)		$ 5.294.118	$ 5.294.118	$ 5.294.118	$ 6.901.261	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 8.632.030	$ 135.000.000
Profit ($/year)		$ 9.741.450	$ 9.741.450	$ 9.741.450	$ 9.741.450	$ 52.579.100	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 135.700.400	$ 1.642.018.100
Taxable income ($/year)		$ -	$ -	$ -	$ -	$ 43.947.070	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 127.068.370	$ 1.568.767.513
Tax ($/year)		$ -	$ -	$ -	$ -	$ 9.668.355	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 27.955.041	$ 345.128.853
NCF ($/year)		$ 9.741.450	$ 9.741.450	$ 9.741.450	$ 9.741.450	$ 42.910.745	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 107.745.359	$ 1.296.889.247

After all these calculation, net cash flow diagram is formed by using the net cash values from Table 61. Net cash flow diagram can be seen Figure 106.

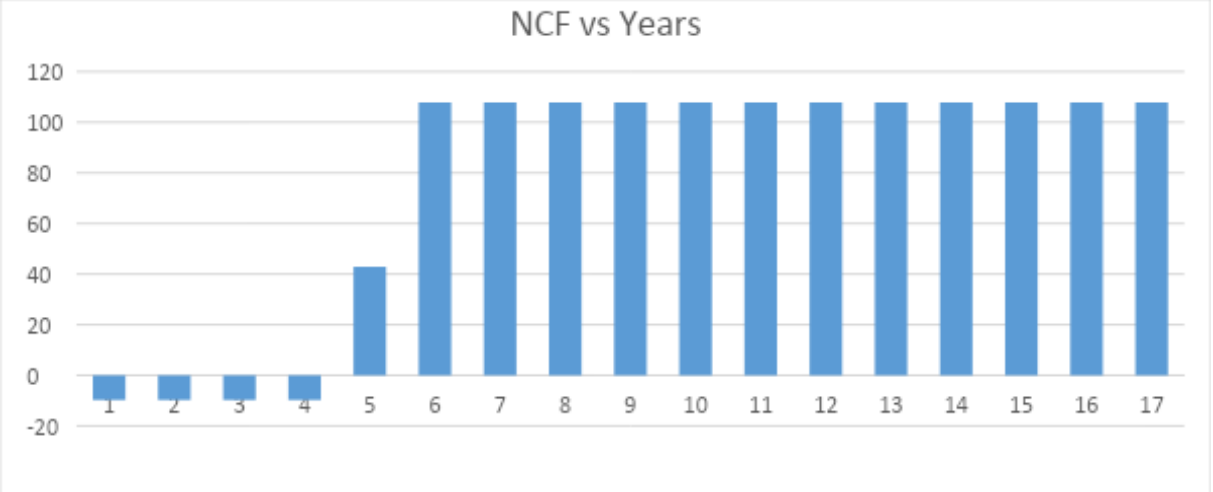


Figure 106. Net Cash Flow Diagram

Payback period analysis is performed using cumulative net cash flows to determine the project's payback period. The project's payback period is roughly 5 years, as shown in Figure 107.

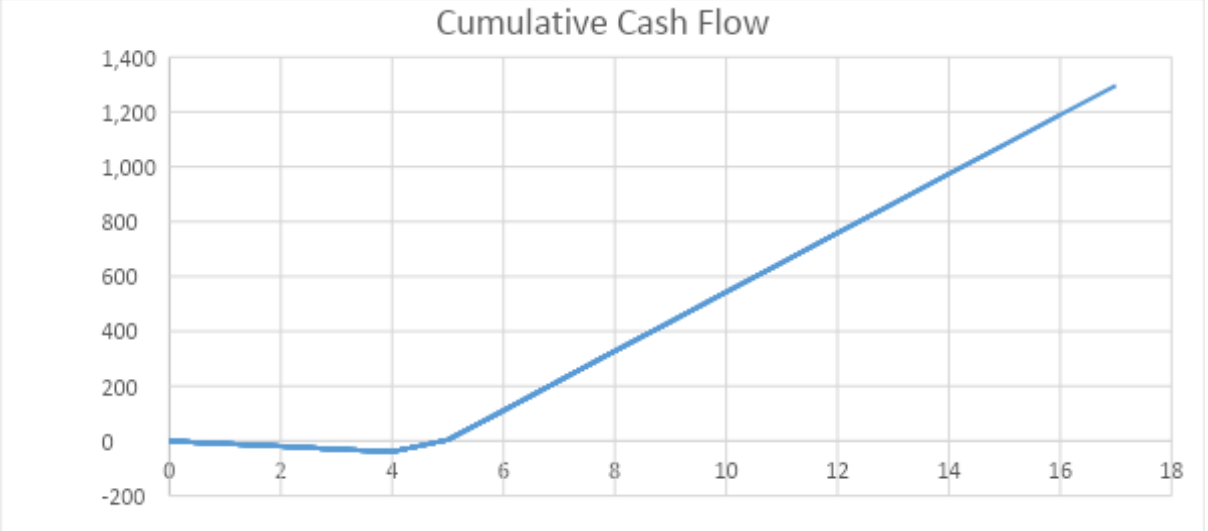


Figure 107. Payback Period Analysis

Following these calculations and analyses, the project's net present value (NPV) is computed using a 3% discount rate. The project's NPV is 925,951,219 dollars, according to this computation. The project's net cash flows are also used to calculate IRR, which is set at 74%.

Appendix A

In Table 45, the costs and dilution is listed for calculation of MASR.

Table 45. Table of Costs and Dilution

Cost Type and Other Parameters	Rate
Stripping Cost ($/m^3)	3.5
Open Pit Mining Cost ($/tonne)	5
Underground Mining Cost ($/tonne)	15
Dilution	%10
Administration Cost ($/tonne)	2

$$MASR = \frac{15\ \$/tonne - 5\ \$/tonne}{3.5\ \$/m^3} = 2.87 \frac{m^3}{tonne}$$

Since Stripping ratio of different open cast models is greater than the MASR value, underground mine should be applied.

Appendix B

Taylor's Mine Life Rule;

N – Expected Mine Life, year **T** – Tonnage [M Tonne] (Proven + Probable Reserve)

$$N = (1 \pm 0.2) * (6.5) * \sqrt[4]{T\ [M\ Tons]} =>$$

$$0.8 * 6.5 * \sqrt[4]{14.9} < N < 1.2 * 6.5 * \sqrt[4]{14.9}$$

$$11\ years < N < 16\ years$$

Appendix C

$S = t \times a$ (Wiley, 1978) Where,

S= subsidence value (m)

t= seam thickness (m)

a= subsidence factor

Since this is a longwall top coal mining method and the depth is 930 m, the subsidence factor can be taken as 0,35. When the calculations are done, the subsidence value is 3.5 m from the equation.

$$S = 10 * 0,35 = 3.5\,m$$

$$W = H * 2 * tanA'$$

$$W = 1030 * 2 * tan30 = 1189\,m$$

Maximum subsidence will be seen when (W) meters of width panel excavated that located (H) meters below the surface.

Appendix D

According to Rock Mass Parameters, C_r=1.76 MPa, C_p=1.76 MPa, φ_r=32.6°, and φ_p=32.6° for Coal. Also, C_r=2.44 MPa, C_p=2.44 MPa, φ_r=32°, and φ_p=32° for Limestone. Therefore, the Bearing capacity of Coal is,

$$q_f = 2 * 1.76 * ta(45 + 16.3) + 2 * 1.76 * ta(45 + 16.3) * (45 + 16.3) = 27.88\,MP$$

The Bearing capacity of Limestone is,

$$q_f = 2 * 2.44 * ta(45 + 16) + 2 * 2.44 * ta(45 + 16) * (45 + 16) = 37.46\,MPa$$

Appendix E

Table 46 Blasting Design Parameters for development

Empty Hole Diameter (d) (mm)	102.00
Charged Hole Diameter (dc) (mm)	48.00
Hole Depth (H) (m)	3.20
Tunnel Excavation Area (m^2)	21.28
Tunnel Width (at bottom) (m)	5.00
Tunnel Width (at mid-height) (m)	5.00
Tunnel Height (m)	5.00

As can be seen from the above Table 46 the blasting design parameters are;

- Empty Hole Diameter = 102 mm
- Charged Hole Diameter= 48 mm
- Tunnel Excavation Area=21.28 m^2
- Drill hole Length (H)= 3.20 m

With an 95% efficiency, the advance will be;

Advance = H x 0.95 =3.04 ~3.00 m

Number of cartridges per meter = 1000 mm / 400 mm = 2.5

Bottom charge = l_b = 545 g x 2.5 = 1362.5 g/m =1.3625 kg/m

Column Charge Concentration is 50% of l_b and it is 680g/m.

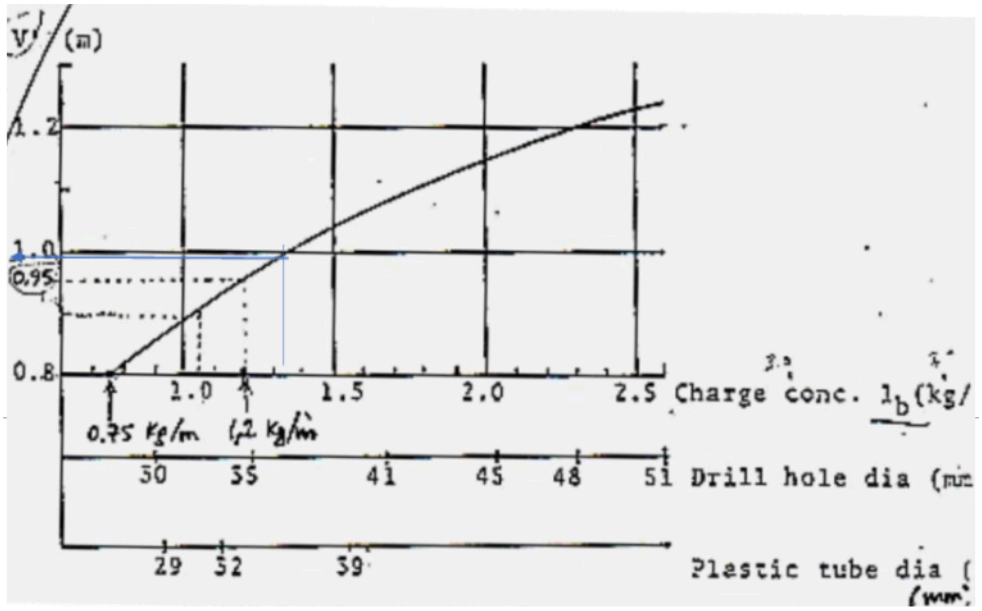


Figure 108. Charge concentration graph (Karpuz, 2012)

According to the Figure 108 V' is about 1m.

V' =1 m

- V=0.7 x d (d stands for the empty hole diameter , V is burden)
- $B^2 = (\frac{d}{2} + V)^2 \times 2$

V_1= 0.7 x 102.00 =71.40 mm

B_1=173.10 mm

V_2= 0.7 x 173.10 =12.17 mm

B_2=293.76 mm

V_3= 0.7 x 293.76 =205.63 mm

B_3 = 498.53 mm

V_4 =0.7 x498.53 =348.97 mm

B_4 =846.03 mm

V_5= 0.7 x 846.03=592.22 mm

B_5=1435.76 mm

V_6= 0.7 x 1435.76 =1005.03 mm

B_6= 2436.56 mm

V_7=0.7 x 2436.56 =1705.59 mm

V_7 is higher than the V' value. Therefore, the cut is completed with 6 squares.

Table 47 Burden Distance Table

Square Number	1	2	3	4	5	6
V (mm)	71.4	121.17	205.63	348.97	592.22	1005.03
B (mm)	173.1	293.76	498.53	846.03	1435.76	2436.56

Table 48 Charge Concentration Table

Square No	Height of Bottom (m)	Column Charge Concentration (kg/m)	Stemming Lenght (m)
1	0.15	0.68	0
2	0.15	0.68	0.0605
3	0.6	0.68	0.1025
4	0.99	0.68	0.1745
5	0.99	0.68	0.2960
6	0.99	0.68	0.5025

The column charge concentration is 0.68 kg/m. The freezing risk charge point is 0.75 kg/m, the 0.68 kg/m is lower. The charge concentration value does not present any risk for freezing.

In order to create a smooth surface after blasting, Orica - Nitro Trimex is used in wall and roof blast holes. It gives homogenous charge conveyance along the edge openings and guarantees a smooth segment with least overbreak or underbreak.

Table 49 Charge Concentration of all blasting agents of Orica

Blasting Diameters (mm)	Length (mm)	Weight (g)	Charge Concentration
27	225.00	155	0.34
30	225.00	191	0.42
32	200.00	193	0.48
34	225.00	245	0.54
34	400.00	436	0.55
36	225.00	275	0.61
40	400.00	485	0.61
38	225.00	306	0.68

38	400.00	545	0.68

Orica Nitro Powergel Magnum 365 is a suitable type for development blasting with chosen charge concentration. The Table 49 shows the other dimensions of the blasting agent to check the freezing points.

Table 50 Burden – Bottom Charge Height – Column Charge – Stemming and Spacing Calculations

Hole type	**Burden (V x V') (m)**	**Spacing (E x V) (m)**	**Bottom Charge Height, (hb x H) (m)**	**Column Charge Conc. (Ip x Ib) (kg/m)**	**Stemming (ho x V) (m)**
Floor	1.0 x 1.0 = 1	1.10 x 1.00 = 1.10	1/3 x 3 = 1	1.0 x 1.5 = 1.5	0.2 x 1 = 0.2
Wall	0.9 x 1.0 = 0.9	1.20 x 0.90 = 1.08	1/6 x 3 = 0.5	0.4 x 1.5 = 0.6	0.5 x 0.9 = 0.45
Roof	0.9 x 1.0 = 0.9	1.20 x 0.90 = 1.08	1/6 x 3 = 0.5	0.3 x 1.5 = 0.5	0.5 x 0.9 = 0.35
Stoping (h&u)	1.0 x 1.0 = 1	1.10 x 1.00 = 1.10	1/3 x 3 = 1	0.5 x 1.5 = 0.75	0.5 x 1 = 0.5
Stoping (d)	1.0 x 1.0 = 1	1.20 x 1.00 = 1.20	1/3 x 3 = 1	0.5 x 1.5 = 0.75	0.5 x 1 = 0.5

The Table 50 refers to the burden, spacing, bottom charge height, column charge concentrations and stemming points of floor, wall, roof and stoping.

Table 51 Bottom Charge Explosives

Type of Hole	hb (m)	Ip (kg/m)	Type of Explosive	Size (mm)	Pieces
Cut 1	0,05 x 3 =0,15	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	0,15 / 0,4 = 0,375
Cut 2	0,05 x 3 =0,15	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	0,15 / 0,4 = 0,375
Cut 3	0,20 x 3 = 0,6	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	0,6 / 0,4 = 2,375
Cut 4	0,33 x 3 =0,99	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	0,99 / 0,4 = 3,375
Cut 5	0,33 x 3 =0,99	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	0,99 / 0,4 = 3,375
Cut 6	0,33 x 3 =0,99	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	0,99 / 0,4 = 3,375
Roof	1/6 x 3 = 0.5	1,40	Orica - Nitro Trimex	19 x 735	0,5 / 0,735 = 0,68
Wall	1/6 x 3 = 0.5	1,40	Orica - Nitro Trimex	19 x 735	0,5 / 0,735 = 0,68
Stoping	1/3 x 3 = 1	1,40	Orica – Nitro Powergel Magnum 365	38 x 400	1 / 0,4 = 2,5
Floor	1/3 x 3 = 1	1,40	Orica - Nitro Powergel Magnum 365	38 x 400	1 / 0,4 = 2,5

The Table 51 shows to the bottom charge explosives and pieces of dynamite for cuts, roof, wall, stoping and floor.

Table 52 Column Charge Explosives

Type of Hole	hp (m)	Ip (kg/m)	Type of Explosive	Size (mm)	Pieces
Cut 1	3- 0.2 - 0 = 2.80	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	2.8 / 0.4 = 7
Cut 2	3- 0.5 - 0.161 = 2.34	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	2.34 / 0.4 = 5.85
Cut 3	3- 0.5 - 0.1 = 2,4	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	2.4 / 0.4 = 6
Cut 4	3 - 0,5 - 0,18 = 2,32	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	2.32 / 0.4 = 5.8
Cut 5	3 - 0.5 - 0.3 = 2.2	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	2.2 / 0.4 = 5.5
Cut 6	3 - 0.5 - 0.5 = 2	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	2.0 / 0.4 = 5
Roof	3 - 0.5 - 0.5 = 2	0,15	Orica - Nitro Trimex	19 x 735	2.0 / 0.735 = 2.72
Wall	3 - 0.5 - 0.5 = 2	0,15	Orica - Nitro Trimex	19 x 735	2.0 / 0.735 = 2.72
Stoping	3 - 0.5 - 1 = 1.5	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	1.5 / 0.4 = 3.75
Floor	3 - 1 - 0.2 = 1.8	0,68	Orica - Nitro Powergel Magnum 365	38 x 400	1.8 / 0.4 = 4.5

The Table 52 refers to the column charge explosives and pieces of blasting agents for cuts, roof, wall, stoping and floor.

Table 53 Total Amount of Explosives

Type of Hole	Number of Drill Holes	Total Amount of Explosive (kg)
Cut Holes	24	72
Roof	7	5
Wall	4	3
Stoping	10	27
Floor	5	16
TOTAL	50	123

The required total amount of explosives can be seen from Table 53 for development roads in horseshoe shape 5x5 m.

The powder factor for development can be calculated as;

$$\text{Powder factor} = \frac{Total\ Amount\ of\ Explosive\ Used\ (kg)}{Volume\ of\ Rock\ Blasted(m3)}$$

$$\text{Powder factor} = \frac{123.00}{63.85} = 1.93\ kg/m^3$$

The underground development blasting part is below.

Table 54 Delay Numbers and Time Intervals

Delay Number and Time Ranges			
Number	**Delay (ms)**	**Number**	**Delay (ms)**
0	0	13	1600
1	100	14	1800
2	200	15	2000
3	300	16	2400
4	400	17	2800
5	500	18	3200
6	600	19	3600
7	800	20	4000
8	900	21	4400
9	1000	22	4800
10	1100	23	5200
11	1200	24	5600
12	1400	25	6000

Firing starts from the cut holes to the wall holes in case of not to give a harm to side rocks. Table 54 shows delay numbers and time intervals.

Number of cartridges per meter = 1000 mm / 200 mm = 5

Bottom charge = l_b = 186 g x 5 = 930 g/m = 0.93 kg/m

Column Charge Concentration is 50% of l_b and it is 465 g/m

Table 55. Charge Concentration for Underground

Square No	Height of Bottom (m)	Column Charge Concentration (kg/m)	Stemming Lenght (m)
1	0,15	0,47	0,00
2	0,15	0,47	0,06
3	0,60	0,47	0,10
4	0,99	0,47	0,17
5	0,99	0,47	0,30
6	0,99	0,47	0,50

The Table 55 shows the column charge concentration and stemming length for underground development part.

Table 56.Burden – Bottom Charge Height – Column Charge – Stemming and Spacing

Hole type	Burden (V x V') (m)	Spacing (E x V) (m)	Bottom Charge Height (hbxH) (m)	Column Charge Conc. (Ip x Ib) (kg/m)	Stemming (ho x V) (m)
Floor	1.0 x 1.0 = 1	1.10 x 1.00 = 1.10	1/3 x 3 = 1	1.0 x 0.93 = 0.93	0.2 x 1 = 0.2
Wall	0.9 x 1.0 = 0.9	1.20 x 0.90 = 1.08	1/6 x 3 = 0.5	0.4 x 0.93 = 0.37	0.5 x 0.9 = 0.45
Roof	0.9 x 1.0 = 0.9	1.20 x 0.90 = 1.08	1/6 x 3 = 0.5	0.3 x 0.93 = 0.28	0.5 x 0.9 = 0.35
Stoping (h&u)	1.0 x 1.0 = 1	1.10 x 1.00 = 1.10	1/3 x 3 = 1	0.5 x 0.93 = 0.47	0.5 x 1 = 0.5
Stoping (d)	1.0 x 1.0 = 1	1.20 x 1.00 = 1.20	1/3 x 3 = 1	0.5 x 0.93 = 0.47	0.5 x 1 = 0.5

The Table 56 refers to the burden, bottom charge height, column charge, stemming and spacing of the underground development part.

The Table 57 shows the bottom charge explosives and pieces of the blasting agents.

Table 57. Bottom Charge Explosives

Type of Hole	hb (m)	Ip (kg/m)	Type of Explosive	Size (mm)	Pieces
Cut 1	0,05 x 3 = 0,15	0.93	Orica - Nitro Super Ajax	32 x 200	0,15 / 0,20 = 0,75

Cut 2	0,05 x 3 = 0,15	0.93	Orica - Nitro Super Ajax	32 x 200	0,15 / 0,20 = 0,75
Cut 3	0,20 x 3 = 0,6	0.93	Orica - Nitro Super Ajax	32 x 200	0,60 / 0,20 = 3,00
Cut 4	0,33 x 3 =0,99	0.93	Orica - Nitro Super Ajax	32 x 200	0,99 / 0,20 = 5,00
Cut 5	0,33 x 3 =0,99	0.93	Orica - Nitro Super Ajax	32 x 200	0,99 / 0,20 = 5,00
Cut 6	0,33 x 3 =0,99	0.93	Orica - Nitro Super Ajax	32 x 200	0,99 / 0,20 = 5,00
Roof	1/6 x 3 = 0.5	0.93	Orica - Nitro Super Ajax	32 x 200	0,50 / 0,20 = 2,50
Wall	1/6 x 3 = 0.5	0.93	Orica - Nitro Super Ajax	32 x 200	0,50 / 0,20 = 2,50
Stoping	1/3 x 3 = 1	0.93	Orica - Nitro Super Ajax	32 x 200	1,00 / 0,20 = 5,00
Floor	1/3 x 3 = 1	0.93	Orica - Nitro Super Ajax	32 x 200	1,00 / 0,20 = 5,00

The Table 58 refers to the column charge explosive and pieces of the blasting agent for the underground development part.

Table 58. Column Charge Explosives

Type of Hole	**hp (m)**	**Ip (kg/m)**	**Type of Explosive**	**Size (mm)**	**Pieces**
Cut 1	3 - 0.2 - 0 = 2.80	0,47	Orica - Nitro Super Ajax	32 x 200	3,00 / 0,20 = 15,00
Cut 2	3 - 0.5 - 0.161 = 2.34	0,47	Orica - Nitro Super Ajax	32 x 200	2.54 / 0.2 = 12,70
Cut 3	3 - 0.5 - 0.1 = 2,4	0,47	Orica - Nitro Super Ajax	32 x 200	2,60 / 0,20 = 13,00
Cut 4	3 - 0,5 - 0,18 = 2,32	0,47	Orica - Nitro Super Ajax	32 x 200	2.52 / 0.20 = 12,60
Cut 5	3 - 0.5 - 0.3 = 2.2	0,47	Orica - Nitro Super Ajax	32 x 200	2,40 / 0,20 = 12,00
Cut 6	3 - 0.5 - 0.5 = 2	0,47	Orica - Nitro Super Ajax	32 x 200	2,20 / 0,20 = 11,00
Roof	3 - 0.5 - 0.5 = 2	0,47	Orica - Nitro Super Ajax	32 x 200	2,20 / 0,20 = 11,00
Wall	3 - 0.5 - 0.5 = 2	0,47	Orica - Nitro Super Ajax	32 x 200	2,20 / 0,20 = 11,00
Stoping	3 - 0.5 - 1 = 1.5	0,47	Orica - Nitro Super Ajax	32 x 200	1,70 / 0,2 = 8,50
Floor	3 - 1 - 0.2 = 1.8	0,47	Orica - Nitro Super Ajax	32 x 200	2,00 / 0,20 = 10,00

The Table 59 shows the total amount of drills and explosives for Orica-Nitro Super Ajax blasting agent.

Table 59. Total Amount of Explosives

Type of Hole	Number of Drill Holes	Total Amount of Explosive (kg)
Cut Holes	24	62
Roof	7	9
Wall	4	5
Stoping	10	13
Floor	5	8
TOTAL	50	97

The required total amount of explosives can be seen from Table 59 for development blasting in coal in horseshoe shape 5x5 m.

The powder factor for development can be calculated as;

$$\text{Powder factor} = \frac{Total\ Amount\ of\ Explosive\ Used\ (kg)}{Volume\ of\ Rock\ Blasted(m3)}$$

$$\text{Powder factor} = \frac{97.00}{63.85} = 1.52 \text{ kg/m}$$

The shaft design and calculations can be shown below.

- Shaft Diameter (D) = 5m => 16.4 ft
- B = Burden (ft)
- SG_e = Density of Explosive (g/cm^3) = 1.2 g/cm^3
- SG_r = Density of Rock (g/cm^3) = 1.7 g/cm^3
- D_e = Diameter of Explosive (in) = 1.5 in
- N_R = Number of Rings
- R_{SH} = Shaft Radius (ft) = 8.2 ft = 2.5m
- B_A = Burden Actual
- S = Spacing
- N_H = Number of Holes/Ring
- R_R = Ring Radius = 6.2

- S_A = Spacing Actual
- L = Advance (ft)
- J = Subdrill
- T = Stemming
- H = Hole Depth (ft) = 4.92 ft x 0.95 efficiency = 4.67 ft =>1.42m

$$B = \left(\frac{2SG_e}{SG_R} + 1.5\right)D_e => \left(\frac{2x1.2}{1.7} + 1.5\right)x(1.5) = 4.37\,ft = 1.3m$$

$$N_R = \left(\frac{R_{SH} - \frac{B}{2}}{B}\right) + 1 = \left(\frac{8.2 - \frac{4.37}{2}}{4.37}\right) + 1 = 3$$

$$B_A = \frac{2R_{SH}}{2N_R - 1} = \frac{2\ x\ 6.2}{2\ x\ 3 - 1} = 2.48\,ft = 0.75m$$

$$S = B = 4.37\,ft = 1.3m$$

$$N_H = \frac{2R_R\pi}{S} = \frac{2\ x\ 6.2\ x\ 3.14}{4.37} = 8.91 => 9$$

$$S = \frac{2R_R\pi}{N_H} => \frac{(2\ x\ 6.2\ x\ 3.14)}{9} = 4.32\,ft = 1.3\text{m}$$

$$L = 2B => 2\,x\,2.48ft = 4.96\,ft => 1.5m$$

$$J = 0.3B => 0.3\,x\,4.37 = 1.3\,ft => 0.4\,m$$

$$T = 0.5B => 0.5\,x\,4.37 = 2.2\text{ ft} = 0.67 => 0.7\text{ m}$$

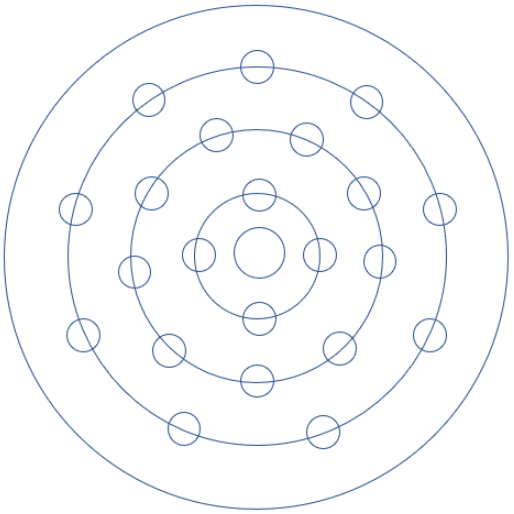


Figure 109. The drilling pattern design of shaft

The minimum delay time for shaft blasting is 100ms. In this design the delay time is taken as 120ms. The chosen blasting agent is Powergel Magnum 365 and it has 38mm x 225mm diameter and length.

Number of Cartridge per 1000m

$$\frac{1000}{225} = 4.5 \Longrightarrow 4.5\ x\ 306g \Rightarrow \frac{1380g}{m} \Rightarrow l_b$$

Column Charge Concentration

$$\%50\ of\ Lb \Rightarrow \frac{\frac{1380g}{m}}{2} = 690g/m$$

$$Charge\ Weight = 1.5\ x\ 1.38\ kg = 2.07\ kg$$

$$Target\ powder\ factor\ is\ 1.8.$$

$$Round\ Volume = Face\ Area\ x\ Round\ Depth = \frac{3.14\ x\ 5\ x\ 5}{4} x\ 1.3 = 25.5\ m^3$$

$$Total\ Explosive\ Weight = 1.8\ x\ 25.5 = 45.9\ kg$$

$$Total\ drillhole\ =\ 22$$

$$Powder\ Factor\ =\frac{45.9\ kg}{25.5m^3}\ =\ 1.8$$

Appendix F

Eqipments	Quantities	Power (kW)	Total kW
Roof Scaler	1	129	129
Coguaro 4 Concrete Mixer	1	121	121
Pc158 Concrete Shooter	1	22	22
Cat Underground Mine Truck	3	550	1650

Roadheader	1	550	550
R1700G Underground Loader	2	250	500
Total	9	1622	2972

Table 60 Amount of Air for Diesel Engines for Production

Utilization rates are assumed as 0.6 because not all the equipment. Works at the same time. Also the safety factor 1.2 is calculated and added to the needed air quantity.

$$0.6\ x\ 2972\ =\ 1783.2\ kW$$

$$Total\ Q\ =\ (8\ x\ 1783.2\ kW)x\frac{1.2}{100}\ =\ 171,2\frac{m^3}{sec}\ needed\ air\ quantity$$

Equipment	**Quantities**	**Power (kW)**	**Total kW**
Roof Scaler	1	129	129
Coguaro 4 Concrete Mixer	1	121	121
Pc158 Concrete Shooter	1	22	22
Cat Underground Mine Truck	2	550	1100
R1700G Underground Loader	2	250	500
Total	7	1072	1872

Table 61 Pre-Production Stage Ventilation

Utilization rates are assumed as 0.6.

$$0.6\,x\,1872\;=\;1123.2\,kW$$

$$Total\,Q\;=(8\,x\,1123.2\,kW)x\frac{1.2}{100}\;=\;107.8\frac{m^3}{sec}\;needed\;air\;quantity$$

Amount of Air for Workers

Assumed 100 workers will work actively in underground. For very active personals 6 m^3/min. oxygen amount is assumed.

0.2095 x Q – 6 = 0.20Q

Q = 631.6 lt/min for an active worker

For 100 person 631.6 x 100 = 63160 lt/min 63.16 m^3/min 1.1 m^3/sec

Amount of CO2 produced by workers

0,0003 x Q + 6 = 0.005Q

Q = 1276.6 lt/min

Total Carbon Dioxide quantity = 127660 lt/min about 2.1 m^3/sec

Amount of Air to Dilute the Dust

The threshold limit for dust is 2 mg/m^3. The dust production value G is 0.5 mg/min. The intake air value B is 0.5 mg/m^3. The required air can be calculated with the formula;

$$Q = \frac{G}{TLV-B} => \frac{0.5}{2-0.5} = 0.33\ m^3/s$$

Amount of Air to Dilute Harmful Gases

The threshold limit of CH4 is %1 and the threshold limit of CO2 is %0.5. For safety working conditions, these harmful gases needed to dilute.

E Harmful gas inflow (m^3/sec)

X Gas concentration in the intake air

Q Amount of fresh air needed (m^3/sec)

MAC Maximum allowable concentration

$$Q = \frac{E}{(MAC-X)} - E = 9.9\frac{m^3}{min} => 0.2\frac{m^3}{sec}$$

Amount of Air to Dilute for Blasting

t: Ventilation time (sec.) (30 min.)

F: Amount of used explosive (kg) (123kg)

V: Ventilated area (m^3) (63.84 m^3)

The blasting cause harmful gases and dust problems in underground mining operations. To prevent the problem needed air quantity should be calculated.

$$Q = \frac{0.14}{t} x \left(FV^2\right)^{1/2} = \frac{0.14}{1800sec.}\ x\ \left(123\ kg\ x\ 63.84^2 m^3\right)^{\frac{1}{2}} = 0.06\frac{m^3}{s}$$

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