VENTILATION DESIGN OF THE GULEMAN KEF CHROMIUM MINE

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Approval of the Graduate School of Natural and Applied Sciences.

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I certify that this thesis satisfies all the requirements as a thesis for the degree of Doctor of Philosophy.

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This is to certify that we have read this thesis and that in our opinion it is fully adequate, in scope and quality, as a thesis for the degree of Doctor of Philosophy.

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ABSTRACT

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The aim of this study is to develop solutions for reducing excessive dust concentration due to insufficient ventilation which causes interruptions in the mining operations of underground chromium mines at the Guleman district.

Natural ventilation system is currently being applied at the chromium mines in the district. The natural ventilation pressure leads to air circulation with variable air quantity. The air quantity obtained from natural ventilation is not uniform. Morever, the air quantities circulating in the mines which are resulted from seasonal differences are insufficient.

Dust concentration exceeds the maximum allowable dust concentration limits especially during production and after blasting operations. This situation consequently halts the mining activities and results in loss of economical values.

A laboratory equipped with extensive pressure and dust concentration measuring devices is established in the vicinity of the mine in an effort to support the field studies. The natural ventilation pressure quantities and the dust concentration values are measured continuously throughout one year. Some of these data are transferred to computer and evaluated by VnetPC2000 software.

Ventilation design for reducing dust concentration and supplying sufficient air quantity to provide a comfortable working environment is presented together with relevant parameters for the design in detail in the thesis. Morever, a suitable fan is selected for supplying required air quantity in the mine.

Finally, optimum values for insufficient ventilation and excessive dust concentration are determined and recommendations for better working environment are made.

Key Words: Natural Ventilation, VnetPC2000, Dust Concentration, Mine Ventilation Design, Fan.

ÖZ

GULEMAN KEF KROM MADENİNDE HAVALANDIRMA TASARIMI

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Bu araştırma, Guleman Bölgesi Yeraltı Krom İşletmelerinde, yetersiz havalandırmadan dolayı üretimin aksamasına neden olan aşırı toz konsantrasyonun azaltılmasına yönelik çözümler geliştirmeyi amaçlamıştır.

Bölgedeki işletmeler, doğal havalandırma yöntemi ile havalandırılmaktadır. Doğal havalandırma basıncı oluşan hava akım miktarı değişken olabilmektedir. Meydana gelen hava akım miktarını sürekli olarak kontrol etmek mümkün değildir. Mevsim değişikliklerine bağlı olarak oluşan doğal havalandırma miktarı genellikle yetersiz olmaktadır.

Havalandırma yetersizliğinden dolayı, üretim esnasında ve ateşlemeler sonucunda toz konsantrasyonu aşırı derecede artmaktadır. Bu koşullar üretimin uzun süre durmasına dolayısıyla büyük ekonomik kayıplara yol açmaktadır.

Deneysel çalışmalar için bölgede orta çaplı bir pilot laboratuar kurulmuştur. Laboratuarda doğal havalandırma basıncı ve işletmedeki toz konsantrasyonu bir yıl boyunca ölçülmüştür. İşletmede elde edilen verilerin bir kısmı bilgisayar ortamına aktarılarak, VnetPC2000 adlı havalandırmaya ilişkin paket programı ile değerlendirilmiştir.

Toz konsantrasyonun azaltılması ve rahat bir çalışma ortamı sağlamak için, işletmenin ihtiyacı olan hava miktarını gerçekleştirecek ocak havalandırma tasarımı yapılmıştır. Tasarım için gerekli parametreler tez içinde detaylı bir şekilde anlatılmıştır. Ayrıca, işletmenin ihtiyacı olan hava miktarını sağlayacak fan seçimi gerçekleştirilmiştir.

Sonuçta, sorun teşkil eden havalandırma yetersizliği ve toz konsantrasyonu için optimum değerler belirlenmiş ve daha iyi çalışma ortamını sağlayabilmek için, uygulama şansı büyük olan öneriler yapılmıştır.

Anahtar Kelimeler: Doğal Havalandırma, Toz Konsantrasyonu, VnetPC200, Ocak Havalandırma Tasarımı, Fan.

To all members of my family

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INTRODUCTION

1.1. General Remarks

In the Guleman district, the ventilation problem arises for all underground workings. The mine operation is interrupted due to insufficient ventilation. High dust concentration is formed during production and after blasting. A study must be carried out on this subject since; the interrupted time between two successive shifts of operation takes up to four hours. This waste time in a mining operation is very unfavorable and it must be kept in a tolerable range.

Guleman district is the most important chromium ore producing in the Turkey. There are twenty-six underground mines and two open pit mines operating in the district. In the earliest days of the mines, all underground mines worked in open pit mine. Then, they went on operation in underground.

Underground sublevel caving method is applied due to orebody and surrounding rock conditions. All main levels are in the form of adits and sublevels are connected with upper level by drifts. This situation leads to natural ventilation due to the difference in air density of mine air and atmospheric air. The amount of circulating air is not constant and can not be controlled. The air requirement of such mine activities may be higher or lower during production and after blasting than that of provided by natural means.

In order to evaluate the ventilation problem in the district, a mine was firstly chosen, named as Kef mine, to carry out the field study. The Kef mine comprises 5.000.000 tons of ore, which is 80% of the total ore in the Guleman district. The bottom and the top level of the Kef mine have an elevation of 1214-m and 1416-m, respectively. It is a continuing part of an outcrop mined before as an open pit, which had elevation of 1657-m. The orebody extends from 1657-m to 1214-m. without any interruption. There may be a very large deposit under the level of 1214-m. since, the orebody has geometry with an average thickness of 40-m and width of 185-m. There is not any prospective study on the ore deposit under this bottom level.

1.2. Objectives and Scope of the Thesis

The basic aim of this study is to design the ventilation system of the Kef underground mine by which the excessive dust concentration can be reduced to a tolerable level. For this purpose, a recently produced and most compatible computer software called VnetPC 2000 is used. It is intended to develop a reliability-based design procedure that can be used by the practicing miners at the Guleman district. Therefore, the advanced ventilation design practices as well as design procedures are adopted as the principal formats.

Based on extensive literature survey, data from research results that are relevant to the evaluation of ventilation and dust concentration are collected. Then, these data reprocessed according to the requirements of the ventilation design for reducing dust concentration in the mine are quantified. Field studies were therefore planned to be carried out in this underground mine.

A research laboratory equipped with extensive ventilation quantity and dust concentration surveying devices was initially established in the vicinity of the mine in an effort to support the field studies. The samples collected from the mine are analyzed and evaluated in this laboratory.

In Chapter 2 the basic concepts about ventilation are described, fan and natural ventilation literature are reviewed.

Chapter 3 contains the basic concepts about dust. Respirable dust hazard and control methods are given in detail.

The field studies on ventilation and dust surveys are presented in Chapter 4. The mine location, geology and owner are firstly introduced. Geochemical, mineralogical and petrographical properties of the ore is determined. The ventilation system used in the mine is only by the natural means. Therefore, the quantities of air circulating in the mine and natural ventilation pressure was measured throughout one year. Dust concentration was measured during the mining operation and after blasting throughout the year. The results of these surveys are presented in this chapter.

Chapter 5 contains computer application. Results of the air quantity and pressure surveys were used as input data to run VnetPC2000 computer software program for ventilation planning and design. All airways in the mine introduced to the program with their geometry, resistance, pressure loss and real country coordinates. The results of computer are very similar to that observed in the experimental study.

In Chapter 6, results of the experimental studies are evaluated and fan selection is presented. Finally, the conclusions and recommendations are summarized in Chapter 7.

CHAPTER II

FUNDAMENTAL ASPECTS OF VENTILATION

2.1. Introduction

Mine ventilation is essentially the application of the principles of fluid dynamics to the flow of air in mine openings. While air is a compressible fluid, airflow in underground mines is generally treated as steady state, turbulent and incompressible. Airflow is induced from the atmosphere, through the mine and back to the atmosphere by creating a differential pressures between the intake and return openings of the mine. This differential pressure is very small, of the order of 2 to 3% in most cases, when compared to the absolute pressure of the system. Therefore, volume and density changes are neglected without serious loss of accuracy or validity. However, in situations where the pressure, temperature and humidity changes are large and air conditioning processes are involved, calculations must incorporate compressibility considerations (Ramani, 1992).

The mine ventilation system consists of fans, airways, and openings to the surface and interconnections in the mine between the openings through the working area and control devices for air coursing. Quantity control in this system means achieving desired flow through the optimal selection of openings to surface; shape, size and number of airways, location of control devices, selection and location of fans. The term optimal as used here is global and not restricted to ventilation only. Very often, the airflow logistic problem is intertwined with mine and material transport as well as with many other safety and productivity considerations. The mine airflow distribution is completely defined by:

i. Physical parameters of the airways are shape, area, length and characteristics of the airway surface.

ii. Layout of the mine openings.

iii. Pressure sources in the system.

iv. Interconnections between the airways mine openings and pressure sources.

2.2. Density of Mine Air

Mine air is a mixture of gases and water vapor. The density or the specific weight of the air is the weight per unit of volume of the air-water vapor mixture. It is a function of the barometric pressure and dry-wet bulb temperature. An approximate formula for calculating density for dry air conditions is:

$$w = \frac{(B - 0.378f)}{0.287(273 + t_d)} \qquad (2.1)$$

Where;

w = Density of dry air in kg/m^3,

B = Barometric pressure in Pa,

t_d = Dry-bulb temperature in ^{0}C,

f = Vapor pressure at the dew-point temperature in kPa.

The density of air at 21.11 ^{0}C is the reference standard. This standard density value is 1.2014 kg/m^3.

2.3 Airflow Fundamentals

Airflow fundamentals consist of pressure difference, resistance and power consideration in an underground opening.

2.3.1. General Equation for Pressure Difference

The relationship of the total pressures or heads at two points between which airflow occurs is given by:

$$P_{T1} = P_{T2} + P_{L1\text{-}2} \tag{2.2}$$

Where;

P_{T1} = Total pressure at point 1,

P_{T2} = Total pressure at point 2,

$P_{L1\text{-}2}$ = Pressure loss due to flow between point1 and point 2.

The total pressure at a point (P_T) is the sum of the absolute static pressure (is the summation of atmospheric pressure P_a and the gage static pressure P_s), the velocity pressure P_v and the elevation head at the point z:

$$P_T = (P_a + P_s) + P_v + P_z$$

(2.3)

The velocity pressure at a point is given by:

$$P_v = \frac{wV^2}{2g} \tag{2.4}$$

Where;

w = Density of dry air in kg/m^3,

P_v = Velocity pressure in Pa,

V = Air velocity in m/s,

g = Gravity acceleration in m/s^2.

In equation (2.4), w/g is the mass density in kg/m^3. Air always flows from a point with higher total head to a point with a lower total head. Air velocity pressure and velocity pressure differences are very small. Therefore, only static pressures are used in many mine ventilation calculations. However, where velocities are high and where there are major changes in cross sectional areas, velocity pressures must be considered as well.

2.3.2. Resistance to Airflow

The resistance to the airflow in a mine opening arises from several sources:

i. Viscosity of the airflow (internal friction),
ii. Friction between the airflow and the airway internal surface,
iii. Changes in area and direction of airflow,
iv. Obstruction in the path of airflow.

The pressure loss incurred in overcoming the resistance from the first two sources is friction loss and that in overcoming the last two is shock loss. Friction losses account for over 70% of the pressure loss in mine ventilation and are by far, the most important. Friction loss due to viscosity of the air is generally neglected, as the airflow in mine airways is virtually always turbulent. Shock losses, which may account for as much as 30% of the pressure losses must be properly, considered in mine ventilation design.

The most widely used formula for friction pressure loss in mine airway is the Atkinson Formula:

$$P_f = \frac{KCLV^2}{A} \qquad (2.5)$$

Where;

P_f = Pressure loss in Pa,

L = Length of the airway in m,

C = Perimeter of the airway in m,

V = Velocity of airflow in m/s,

A = Cross sectional area of the airway in m^2,

K = Friction factor of the airway in kg/m^3.

Since L, C, A and K are constant for a given air friction pressure is proportional to velocity of air. Atkinson's equation is similar to the Darcy-Weisbach formula for calculating the head loss for fluid flow in pipes and conduits. The coefficient K in the Atkinson's equation is an empirical factor. It is value is determined by measuring the values in equation (2.5).

The value for K is a function of the type of flow in an airway. Therefore, it is a function of Reynold's number and in turn of velocity (V). In a mine airway, however, there are many minor variations in area, shape and velocity of the air. In view of this fact and that the flow in most cases is turbulent, no major error is committed by using Atkinson's equation for estimating pressure loss. Where more accurate calculations are needed or where the flow is not turbulent, the influence of Reynold's number on friction factors may be taken into considerations.

Shock losses arise from changes in direction changes (e.g., bends) in cross sectional areas (e.g., obstructions) or changes in both (e.g., junctions). Shock losses are independent of roughness of walls and therefore can not be computed directly as friction losses. However, shock losses bear a constant ratio to the velocity pressure corresponding to the mean velocity of flow. There are three methods for estimating shock losses:

i. Calculating the shock loss as a function of the velocity head:

$$P_x = XP_v \tag{2.6}$$

Where;

P_x = Pressure loss due to shock in Pa,

P_v = Velocity pressure in Pa,

X = An empirical shock loss factor found by experiment.

ii. Account for shock losses by increasing the value of the friction factor K for that section of the airway where shock losses occur.

iii. Account for shock loss by expressing a shock loss condition as an additional length of a straight airway to be added to the given length of the airway.

This length is known as the equivalent length, L_e. With the equivalent length method, the head loss in an airway is obtained by including in the Atkinson equation the equivalent length:

$$P = \frac{K(L + L_e)CV^2}{A} \qquad (2.7)$$

Where;

L_e = Equivalent length in m.

This method is recommended for all routine mine ventilation calculations. Hartman and Mutmansky (1982) provide values for equivalent length for various shock conditions based on an airway with K equals to 0.01855 kg/m^3 and a hydraulic radius of 0.61. Table 2.1 illustrates equivalent lengths for various sources of shock loss.

2.3.3. Power Considerations

The total pressure loss is the sum of both frictional and shock losses in flow from one point to another. For flow to be continuous in the airway, the pressure lost by the air must be compensated for by a continuous input of pressure to the air.

Table 2.1 Equivalent lengths for various sources of shock loss (Hartman and Mutmansky, 1982)

Shock Loss Condition	**Equivalent Length (m)**
Bend, acute, round	1
Bend, acute, sharp	50
Bend, right, round	0.3
Bend, right, sharp	21
Bend, obtuse, round	0.15
Bend, obtuse, sharp	5
Doorway	21
Overcast	19
Entrance	1
Discharge	19
Contraction, gradual	0.3
Contraction, abrupt	3
Expansion, gradual	0.3
Expansion, abrupt	7
Splitting, straight branch	9
Splitting, deflected branch (90^0)	65
Junction, straight branch	18
Junction, deflected branch	9
Mine car or skip (20% roadway)	30
Mine car or skip (40% roadway)	152

In other words, the loss of power due to friction and shock must be compensated for by input of equivalent power. This power is known as the air horsepower and is given:

$$AHP = \frac{PQ}{1000} \tag{2.8}$$

Where;

AHP = Air horsepower in kW,

P = Pressure in Pa,

Q = Airflow quantity in m^3/s.

2.4. Control Devices

Two distinct airflows, intake and return, are generally recognized in mine ventilation. The fresh airflow or intake air is the air form the surface that is conducted directly to the workings and other desired locations to provide fresh air to the miners and to dilute and carry away gasses and dust from the workings. After this air has served this purpose, it becomes return air to be conducted back to the surface.

In general, intake and return airflows must not allowed to mix. Control devices in mine ventilation serve to separate the intake and return airflows in adjacent airways. They allow the crossing of the intake and return airflows without mixing. Control devices regulate the flow of air through the various airways in the desired manner when the quantity has to be split between the airways.

2.4.1. Stoppings

Stoppings are physical barriers erected between intake and return airways to prevent the air flowing in them from mixing with each other. Stopping can be temporary or permanent. Temporary stoppings are often constructed of fire resistant jute fabric, plastic, rough lumber covered with plastic or even various types of metal sections. These are extensively used in areas where frequent adjustments to air directions are necessary, such as in the working panels.

Permanent stoppings are installed in places where a permanent or long-term control of flow is needed, such as between the main intakes and returns. They should be substantially built so that they are airtight and if they are functioning as seals or bulkheads, they must resist the disruptive forces of explosions. Shown in Figures 2.1 and 2.2 are two types of such constructions. The one in Figure 2.2 is able to withstand greater pressures than the stopping in Figure 2.1. The greater distance between the two masonry walls, the more effective will be the stopping. The stopping is usually keyed into the roof, and sides.

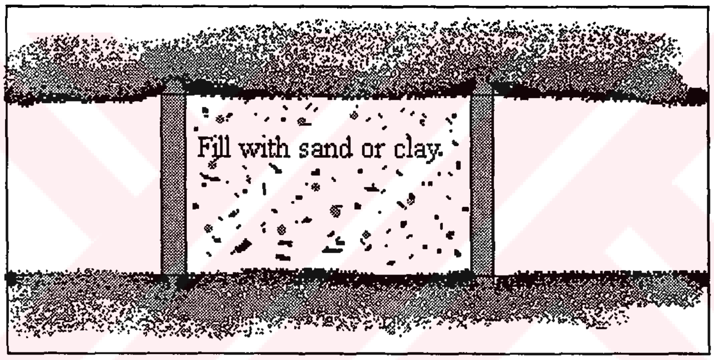


Fig.2.1 Well constructed explosion proof stopping

It is common practice to provide a fire resistant stringer or cushion across the top to prevent cracking due to strata pressure. Since stoppings must be accessible for inspection and repair, it is advisable that they be built where the roof and sides are secured. Additionally, gobing rock against the stopping must be avoided as this makes both inspection and maintenance difficult.

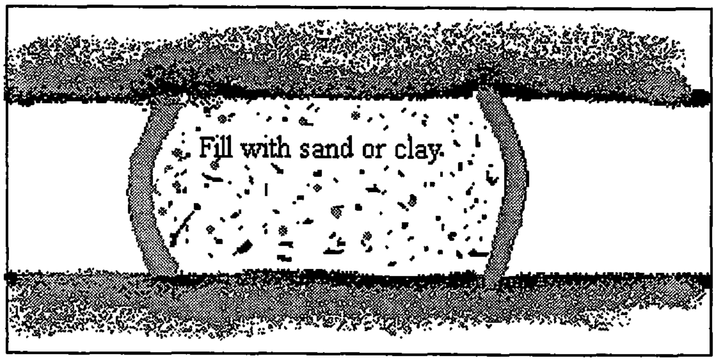


Fig.2.2 Excellent construction for explosion proof stopping

A door is simply a hinged or movable partition within a stopping designed to permit the passage of men and equipment. Mine doors may be constructed of metal or of lumber covered with tarpaper, plastic or other sealant material. They serve the same function as stopping and are frequently used in haulageways. To avoid short-circuiting between the return and intakes, doors should always be arranged in pairs to provide an airlock one door will always be closed while other is open. Automatic self-opening doors are especially useful along the haulageways do not have sills, so that conveyor belting or brattice clothe should be attached to the bottom of the door.

2.4.2. Overcasts

Overcasts or crossings are air bridges that allow the intake and return airways to cross one another without mixing. Undercasts are less frequently used since these are below the level of surrounding openings and water tends to accumulate in them. Figure 2.3 shows a very purely constructed overcast.

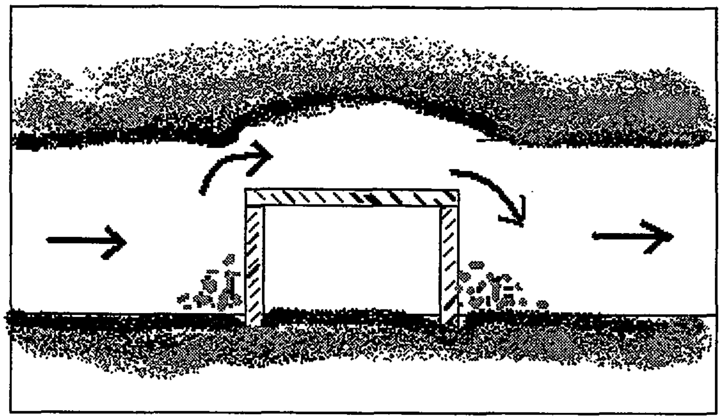


Figure 2.3 Poor construction of overcast

An ideal overcast with long sweeping curves at both the approach and discharge sides is shown in Figure 2.4. The cross sectional area of the overcast should be one half the area of the approaching airway, if the approach is gradually contracted and the discharge is gradually expanded, as shown in Figure 2.4.

The overcast should be properly built with high quality material. In the area where an overcast is required, the roof is raised to the necessary height (Stefanko, 1983). In constructing the overcast, solid blocks set in mortar should be used. The two wing walls must be anchored firmly in the ribs. Steel beams are then placed on the wing walls over which the roof is built. Two walls are then built from this roof to the heightened roofline.

An inexpensive but good quality overcast, most suitable for temporary use. Here a solid wall is built on either side of the airway and a sufficient number of pipes of large diameter are laid on the top of the walls to carry the air cross the main airway. It is possible to transport up to 12 m^3/s air with two pipes of 0.9 m in diameter. The ends of the pipes should be embedded in the stopping and made airtight.

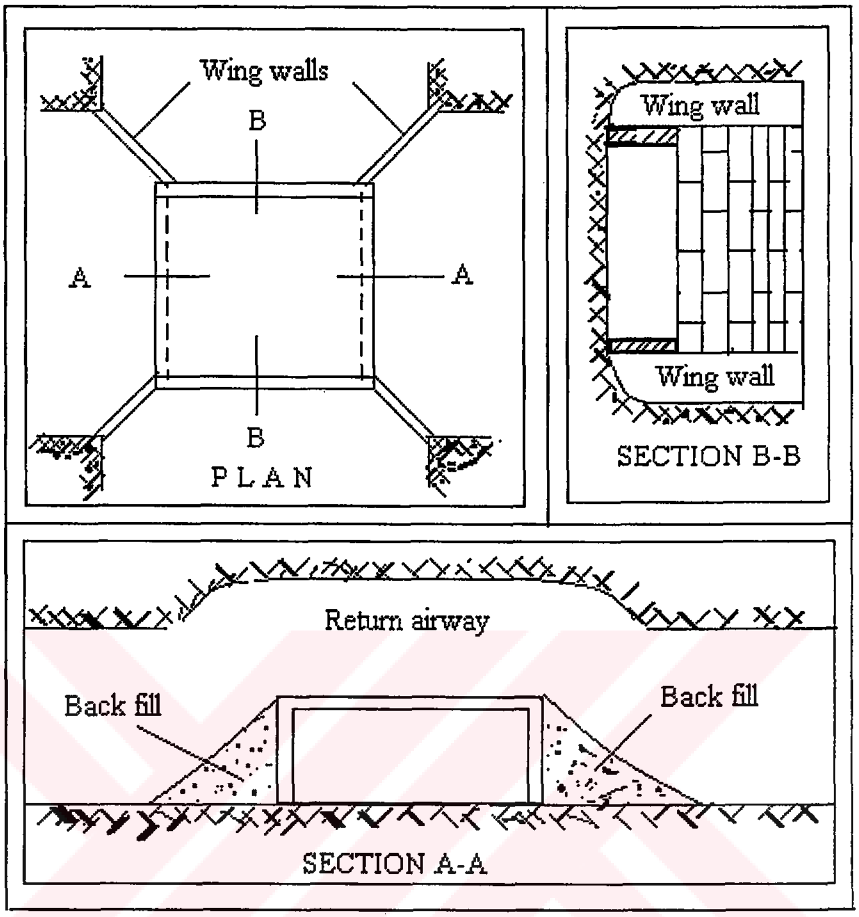


Figure 2.4. Excellent construction for overcast

2.4.3. Regulators

Regulators are used to control and redistribute the quantity of flow in each split of air. The regulator is an opening in a stopping and may be equipped with an adjustable sliding door. Regulators should be located in places where roof and rib conditions are good. They should be kept clear of debris.

Box types are widely used. Figure 2.5 shows a box type regulator. The ideal location for regulators are in return, near the beginning of ventilation split so as not to a dead set with haulage and materials transport. In this location, it can be moved for the life of the working section and it is possible to use it even after the section is mined out.

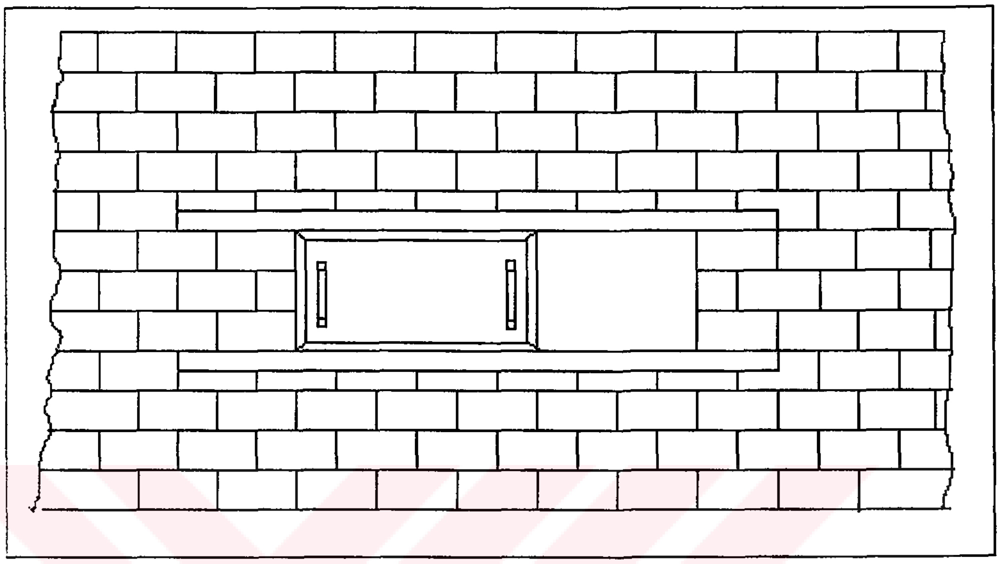


Figure 2.5 Box type regulator

2.5. Leakage Consideration

Unintended losses of air directly to the return from the intake are known as leakage. Leakage occurs through stoppings, overcasts, and doors, crushed pillars and improperly packed gob. Leakage phenomena are complex and amount of leakage that occurs is a function of the pressure differential across the control devices, the construction and condition of the devices and the area exposed.

In the past, on the average, not more than 20% of the air quantity measured at the fans reached the working places. Conditions today are slightly better in that, this figure is anywhere from 50 to 65%. Leakage through stoppings, doors and regulators depends not only upon the pressure across the control devices but also on the condition of the device itself.

Ground pressures destroy control devices and fugitive air losses become so great that it is almost impossible to provide adequate air at the face. Generally, leakage is most severe through old stoppings in the out portion of the circuit. Therefore, the circuit air quantity diminishes at a decreasing rate as the air progress from out to in the circuit. Painting and plastering a stopping decrease leakage, on the average, to around 2-3% of the leakage through a mortar-laid, unplastered and unpainted stopping.

In ventilation planning, leakage has been handled by a system of rough allowances since they often can not be measured accurately. There is very little information on leakage through overcasts, doors and airlocks. The leaking air does not help in the ventilation of the workings. Thus leakage is doubly disadvantageous; it does not improve health and safety conditions and it increases the cost of getting the quantity of air needed into the main split (Ramani, 1992).

2.6. Airflow in Mines

Two basic circuits or combinations of airways, series or parallel are used to distribute air through the mine. A mine ventilation network is a combination of several series and parallel airways.

2.6.1. Series Flow

In series combinations the airways are connected end to end, and the same quantity flows through each of the airways. For a system of N airways in series, the following relationships hold;

$$Q = Q_1 = Q_2 = Q_3 = Q_4 = \ldots\ldots\ldots\ldots = Q_N$$

$$P = P_1 + P_2 + P_3 + P_4 + \ldots\ldots\ldots\ldots\ldots\ldots + P_N$$

$$R = R_1 + R_2 + R_3 + R_4 + \ldots\ldots\ldots\ldots\ldots\ldots + R_N \qquad (2.9)$$

Where, Q_i, P_i and R_i are quantity, pressure and resistance of the ith airway and Q, P and R are the quantity, pressure and resistance of the system, respectively.

2.6.2. Parallel Flow

In parallel flow, all airways start at the same point. Therefore, the pressure difference between the ends of each airway is the same. For a system of N airways in parallel, the following relationships hold;

$$Q = Q_1 + Q_2 + Q_3 + Q_4 + \ldots\ldots\ldots\ldots + Q_N$$

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} + \frac{1}{\sqrt{R_4}} + \ldots\ldots + \frac{1}{\sqrt{R_N}}$$

$$P = P_1 = P_2 = P_3 = P_4 = \ldots\ldots\ldots\ldots\ldots\ldots = P_N \qquad (2.10)$$

Also the following relationship for quantity distribution in the parallel airways can be developed. This quantity distribution is called natural splitting.

$$Q_i = Q\left(\frac{R}{R_i}\right)^2 \qquad (2.11)$$

In natural splitting, the highest quantity flows in the lowest resistance split. Quantity flow in a split with higher resistance is lower than in a lower resistance split.

2.6.3. Controlled Splitting

When airways are connected in parallel and the quantity required in each airway is specified, natural splitting generally will not result in the required distribution. In controlled splitting the airway with highest head loss is known as the free split. The regulators are, in effect, resistances added in series to the airway to increase the head loss. A number of formulas are available to calculate the size of regulators. A simple but approximate formula, known as the equivalent orifice formula, is as follows.

$$A = \frac{1.2Q}{\sqrt{P_x}} \tag{2.12}$$

Where,

A = Area of regulator in m^2,

P_x = Shock loss to be dissipated in the regulator in Pa,

Q = Quantity in the airway in m^3/s.

In practice a box type regulator is placed in a stopping in the airway and opening is adjusted until the desired quantity flow is obtained.

2.6.4. Mine Ventilation Network

A mine ventilation network is defined as a simple network if the series and parallel airways can be combined through the equivalent resistance formulas into one airway with a resistance equal to the network resistance. Otherwise, the network is complex. Solutions for the flow and head for large ventilation networks, whether simple or complex, can be obtained through the use of computer programs that apply the mathematical theory of networks and physical laws of mass and energy conservation to solve the pressure-quantity distribution problem.

2.7. Mine Heads and Mine Characteristic Curve

Three mine pressure heads are recognized: the static head P_s, the velocity head P_v and total head P. The total head is the sum of the static head and velocity heads of the mine. The plotting of the mine head vs. the quantity results in curves called mine characteristic curve.

An example to a mine characteristic curve is shown in Fig. 2.6. It is useful to visualize the solutions to ventilation problems and to facilitate the selection of fans for mine ventilation systems. If one point on the characteristic is known, the other points on the characteristic can be calculated by means of head loss formula;

$$P = RQ^2 \tag{2.13}$$

Where,

R = Resistance of mine in N s^2/m^8,

P = Pressure of mine in Pa,

Q = Quantity in the airway in m^3/s.

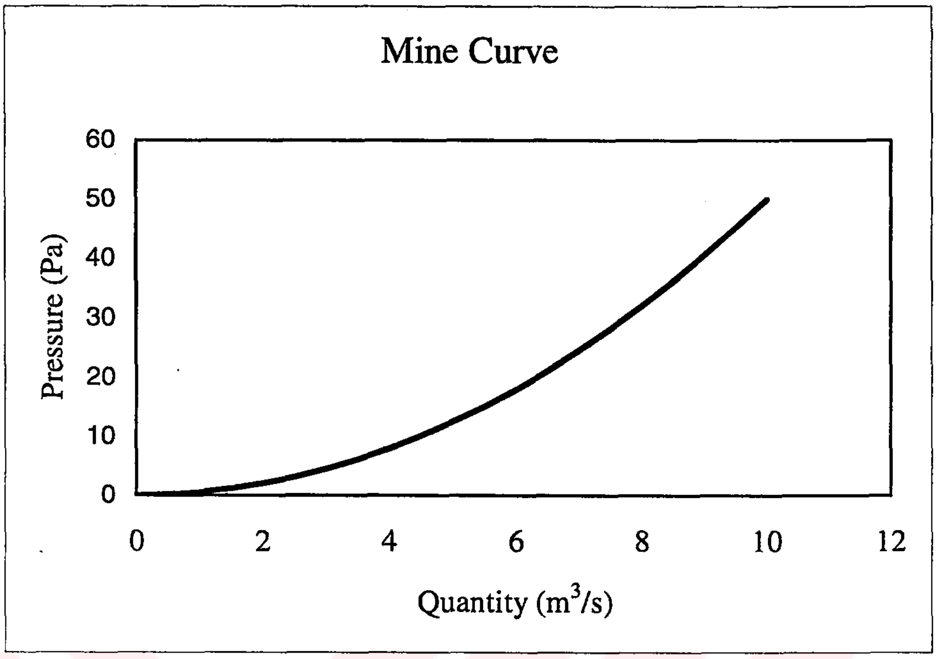


Figure 2.6 A typical mine characteristic curve

2.8. Mine Fan

Fans provide energy required overcoming the resistance to flow. They are mechanical devices, which convert mechanical energy to kinetic energy. By increasing the pressure of the air flowing through it, a fan creates a pressure difference in the system that causes the air to flow and the flowing air to overcome the pressure losses in the system.

There are two main kinds of fans; centrifugal and axial flow fans. While centrifugal fans are still used in some mines, the majority of modern mine fans are axial flow fans. Airflow established by fans is termed as mechanical ventilation.

Measures of fan performance include the range of head and quantity delivered, power requirements, efficiency, noise characteristics, ease of changing operating points etc. While most important among these is the head-quantity relationship, efficiency and noise characteristics are gaining importance from the economic and environmental points of view (Wang, 1982).

As with mine heads, three kinds of fan heads are defined. The total head P_{TF} of a fan is the difference between the total head at fan outlet and the total head at the fan inlet. The velocity head P_{VF} of the fan is the head due to the velocity at the discharge of the fan. The static head P_{SF} of the fan is the difference between the fan total head and fan velocity head. Mathematically, fan total head is the sum of fan velocity head and fan static head.

Fan efficiency η_f is the ratio of the total power output of the fan to the power input supplied to the motor driving the fan:

$$\eta_f = \frac{\text{Power output of the fan}}{\text{Power input supplied to the fan}} \tag{2.14}$$

The graphical method is the most useful way of presenting the performance curves of a fan. These curves are provided by the manufacturers and are for standard density conditions. The design type, diameter and speed of the fan for which the characteristics valid are specified. Fig. 2.7 illustrates a fan characteristic curve. It is customary to plot the quantity as the independent variable with head, power and efficiency as dependent variables.

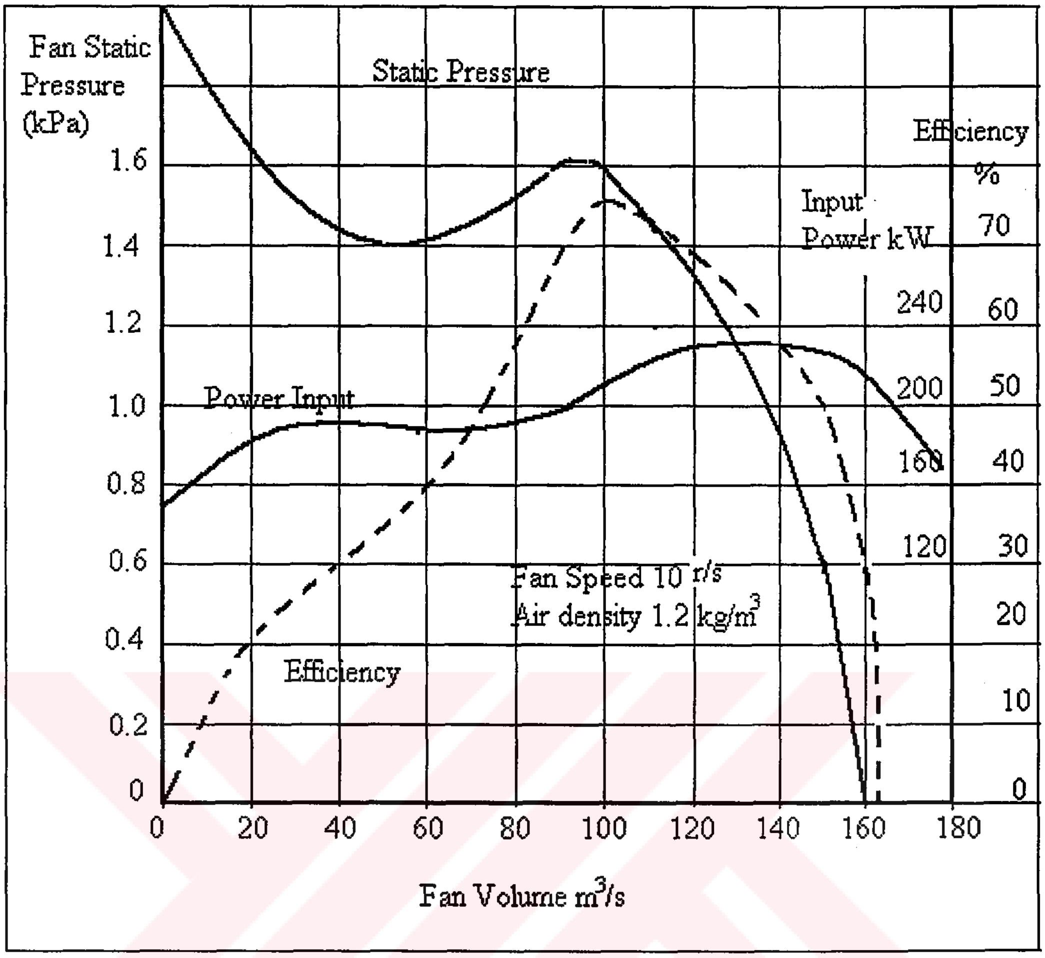


Figure 2.7 A typical fan characteristic curve (Güyagüler and Güngör, 1999)

In most mine ventilation calculations, the static head characteristics of fans are used. As long as the fan velocity head is small, this is acceptable. However, for precise calculations, total head characteristics must be used.

2.8.1. Fan Laws

Fan rotational speed (n), fan size (D), and air density (w) affect the performance of a fan. For geometrically similar fans and for a particular point of operation on the head quantity characteristic, the following relationships are valid:

$$Q \alpha nD^3$$

$$P \alpha n^2 D^2 w$$

$$Power \alpha n^3 D^5 w \qquad (2.15)$$

The fan laws are derived from the above equations and are useful to predict the performance of the fans under new conditions of speed, size and density.

2.8.2. Fan Selection

Over the life of the mine, the demands for head and quantity varies. When the mine is fully developed. and all the units are in full operation, the demand for head and quantity may stabilize somewhat with only small variations due to mine extensions, opening and closing of units.

As the workings move away from the main fan shafts, the duty on the fan may dramatically increase due to new operating sections or decrease due to addition of new shafts and fans. Fan selection procedure must consider these varying duties so that a fan with a range of heads and quantities at high efficiencies is selected (Ramani, 1992). Figure 2.8 represents a fan characteristic with different blade settings.

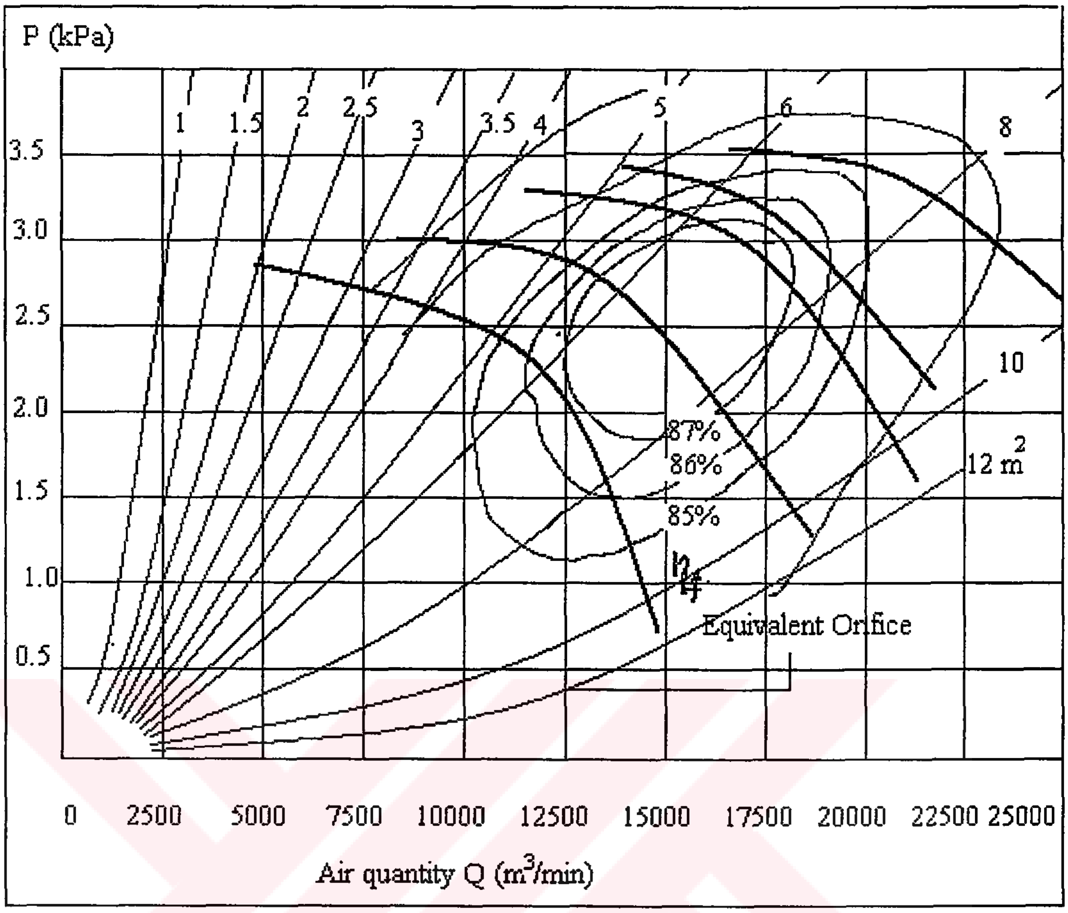


Figure 2.8 Fan characteristic curve according to different blade pitch angle

Some manufacturers provide simplified fan selection charts to aid in the fan selection of fans for given or anticipated mine conditions (Güyagüler and Güngör, 1999).These charts are useful for the initial selection of fans and for initial selection of fan curves for computer software inputs.

2.8.3. Fan Rating

An infinite number of combinations of pressure and volume can be obtained for a fan. Fan rating is a statement of fan performance at one point of operation. Specifically, it is the head, quantity, power and efficiency to be expected when a fan is operating at peak mechanical efficiency. During this operation fan diameter, speed and air remaining constant.

Since the static head developed by a fan can not be mathematically calculated, the static head-quantity curve for a fan must be developed by running fan tests. A complete fan test provides the data for plotting its total static head quantity curves and mechanical efficiency and horsepower.

2.8.4. Fan and Mine Characteristics

When fan and mine characteristics are plotted on the same graph, they intersect, and their point of intersection is known as the operating point. Simply stated, when a fan is introduced into a mine circuit, the air quantity flowing through the mine and the head generated by the fan are completely determined by the fan and mine characteristics.

The mine characteristic and fan characteristic superimposed on it is illustrated in Fig. 2.9. These two characteristics intersect at the operating point O. It should be noted that the true operating point is the point of intersection of the total head characteristic of the fan and the mine. The intersection of the two static head curves gives operating point only if the mine and fan velocity heads are the same or if their difference is very small.

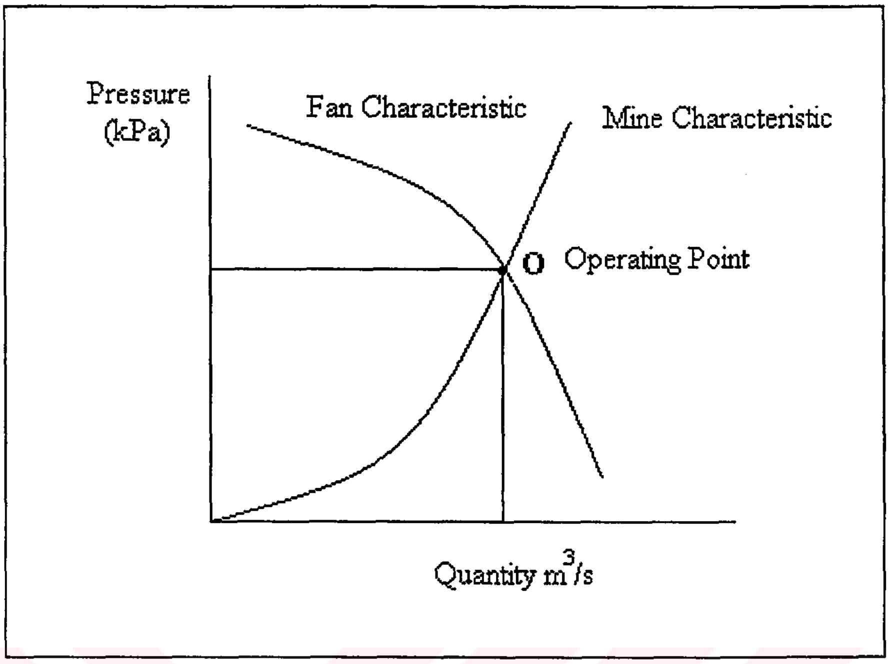


Figure 2.9 A typical fan and mine characteristics

2.8.5. Fan Applications

There are three distinct applications of fans in mine ventilation; main fans, booster fans, and auxiliary fans (Turcic and Banfield, 1982). Main fans are the principal fans used to produce the general ventilation air current in mines. They are normally permanent in nature and large in capacity. Booster fans are usually installed to ventilate parts of the mine where the main fans may be inadequate. Booster fans are used extensively in metal mines. Auxiliary fans are usually of small size, portable and temporarily installed to ventilate working places through which the general ventilating air may not otherwise pass such as dead end openings. It is common practice to conduct the air from the auxiliary fan to the working place by a ventilation tube or duct.

Main fans can be installed on the surface and underground. The primary advantages of locating a fan on the surface are as follows:

i. Accessibility.

ii. Safety.

iii. Ease of installation

When a fan installed on the surface, there can a considerable leakage of air from the outlet side of the fan to the inlet side of the fan unless care is taken. As much as 20% of the air quantity handled by the fan may be air leakage through the surface air lock, fan-housing etc., in which case only 80% circulating through the mine. The placing of main fans underground is more common in metal mines. In underground metal mines generally, separate fans are used for different underground levels.

Depending on the location of the mine circuit, the fan in the mine circuit, the fan static and total pressures generated by it relative to the atmosphere can be positive or negative or both. When the fan is placed on the top of the intake shaft, it is known as a forcing fan and pressures in the mine are above atmospheric. When it is placed on the top of return shaft, it is known as exhausting fan and pressures in the mine are below atmospheric. When it is placed in an underground location it is known as a booster fan. In the booster location the mine pressure is negative form the intake shaft to the fan and is positive from the fan to the exhaust shaft.

2.8.6. Pressure Gradient for Fan System

The total pressure is always zero at the entrance to a system and it is always positive and is equal to the velocity pressure at discharge. In the forcing system the fan inlet is open to the atmosphere and the fan outlet to the mine intake shaft. The top of the intake shaft is equipped with an airlock. The air increases in velocity and static head as it flow through the fan. The absolute pressure measured at any point in the mine is higher than that of atmosphere. The pressure gradient for a simple mine layout is shown in Figure 2.10. In the forcing systems, there is no shock loss at the entrance to the system as the fan absorbs this loss.

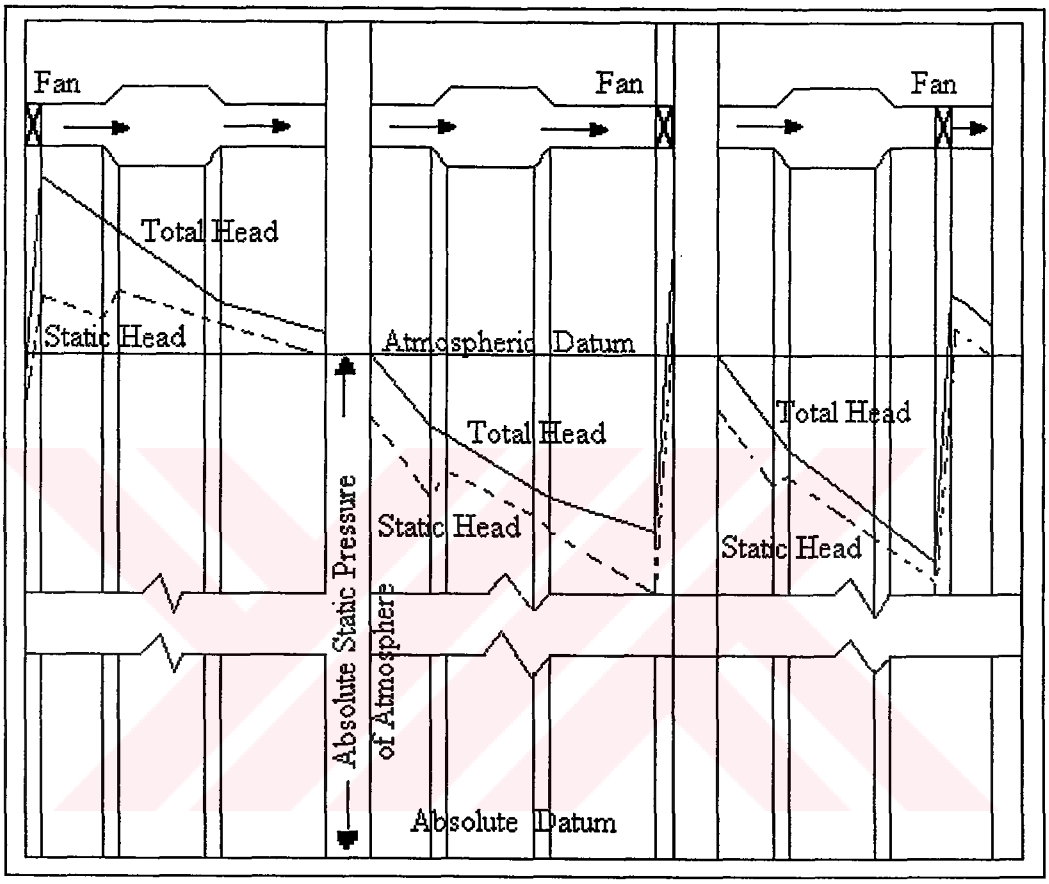


Figure 2.10 Pressure gradient for fan-mine systems

In the exhausting system, the fan inlet is open to the atmosphere, usually through an evase, a gradually expanding exhaust duct, that recovers velocity head. The top of the return shaft has a weak wall or an airlock. The absolute pressure on the inlet side of the fan is lower than that of the atmosphere. Also there is reduced shock loss at the discharge of the system. The pressure gradient for a booster system is a combination of the gradients of the exhausting and forcing system. The pressure gradient on the inlet side of the fan is similar to that of the exhausting system. After fan discharge, the pressure gradient is similar to that of the forcing system.

2.8.7. Multi-fan Arrangements

Increases in head, quantity or both can be obtained by operating more than one fan. Multiple fans can be operated in series, parallel or in combination as shown in Figure 2.11.

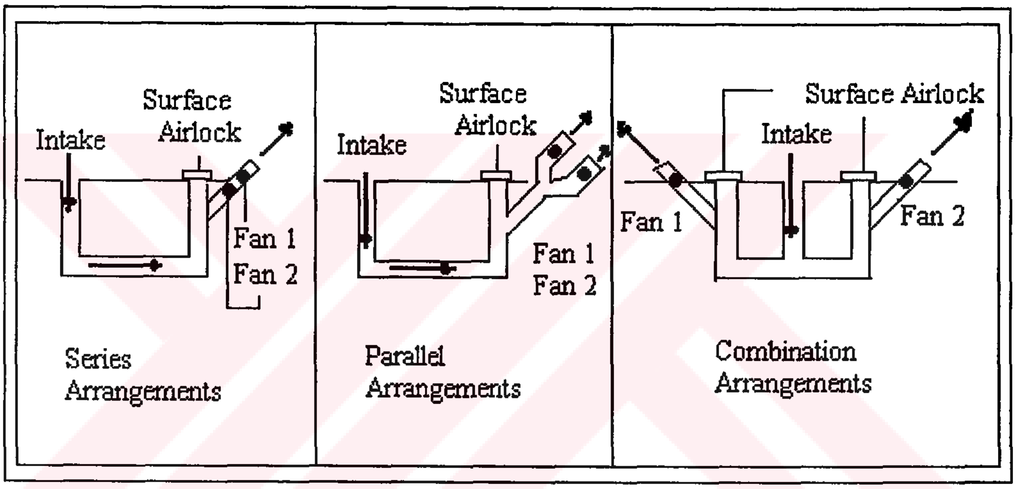


Figure 2.11 Various arrangements for multiple fans

Fans operating in series are on the same flow circuit, each handling the same airflow and each generating a part of the total pressure required. Series operation of fans may be necessary when there is a need for increasing the pressure considerably without increasing the air quantity by the same proportion. Such conditions arise in high resistance mines. The combined characteristic for series operation is illustrated in Figure. 2.12

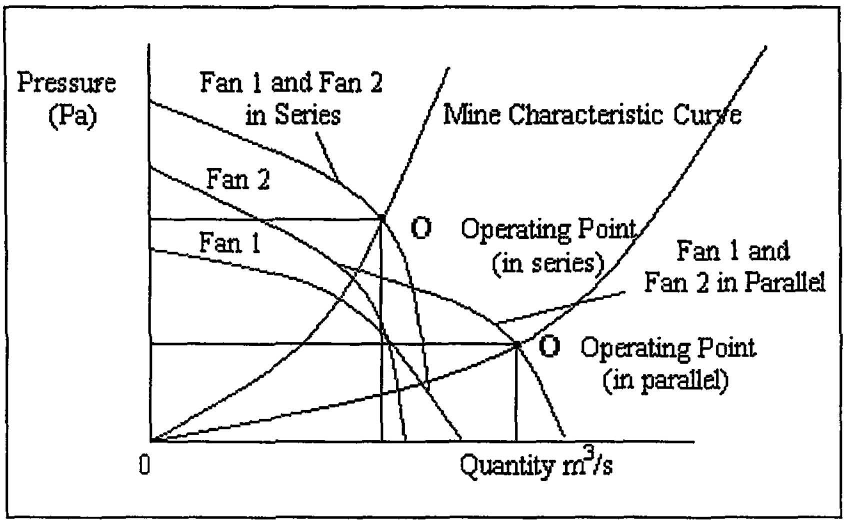


Figure 2.12 Arrangement of two fans in series and parallel

When two fans are installed in separate openings in such a way that they are draw air from the same common point and deliver it to the same destination, they are said to be operating in parallel. Mine fans may be operated in parallel to increase the quantity of air flowing through the system when the pressure capacity of each fan is adequate. These conditions may arise in low resistance mines. The combined characteristic for parallel operation is illustrated in Figure. 2.12.

Many fans are operated in combination rather than in true series or parallel operation. Two possible applications are fans in semi-parallel and fans in semi-series. In these arrangements, each fan shares a portion of the total mine resistance load, but retains its own specific zone of influence. Thus the operation of each fan is influenced by the others in the system. The mathematical analysis of multiple fan systems can become complex and is usually performed more rapidly and accurately with the use of computer oriented mine ventilation network solution programs.

2.9. Natural Ventilation

Natural ventilation is a result of the differences in elevation between mine openings and the addition of heat energy to the air as it passes through the workings. Air as it flows through a mine, gains in heat due to the reasons such as the geothermal gradient, autocompression and oxidation. Air may also be cooled due to contact with flowing water in the shafts or due to rock temperatures being colder than outside air. As a result, the displacement of hot air columns by colder air columns ensues.

When the difference in the elevations of the openings is very small so that the columns of air do not have significant differences in density, flow does not occur naturally. Even if the elevations of the openings are the same, if local weather conditions cause a significant difference in the surface air temperatures above the openings, a flow of air can ensure (Johnson and Ramani, 1992).

2.9.1. Natural Ventilation Pressure

The rate of displacement of one column of air by another is proportional to the difference in the temperatures of the two columns. The differences in temperatures account for the density difference and the pressure heat generated. This pressure head because it arises due to natural conditions is known as natural ventilation pressure (NVP). However, the ventilation pressure provided by natural means is often inadequate, subject to variations and difficult to control, ventilation by mechanical means is absolutely essential for most mines. Natural ventilation affects the performance of the mechanical ventilator, so it is necessary to understand its effects on mechanical ventilation.

The magnitude and direction of NVP are not controllable. NVP is independent of the quantity flowing, depending only on temperature difference and the height of the air column. The difference in temperatures of the air in the intake and return air columns leads to a difference in densities of the air columns. Calculations of the natural ventilating pressures using the densities of the columns of equal heights lead to results that are reasonably accurate for routine mine ventilation calculations.

Natural ventilation pressure changes frequently in magnitude. Its direction also changes, though this change can be seasonal or daily depending on the frequency and amount of temperature variations on the surface. In fact, due to the continual changes in temperature during a day, the NVP also continually changes.

To quantify more accurately the natural ventilation pressure at a mine, a continuous monitoring system consists of measuring dry-wet bulb temperature of a mine air as a function of time and surface temperature should be installed (Loomis and Wallace, 1993). For estimates of NVP, the following approximate formulas can be employed.

$$P_n = \frac{P_B L}{0.287}\left(\frac{1}{T_1} - \frac{1}{T_2}\right) \tag{2.16}$$

Where;

P_n = Natural ventilation pressure in Pa,

P_B = Average absolute pressure in Pa,

L = Vertical height of the air columns in m,

T_1 = Average absolute temperatures of the upcast columns in ^0C.

T_2 = Average absolute temperatures of the downcast columns in ^0C.

The approximate average absolute pressure is obtained by direct measurement near the center of the columns or from a measurement at any elevation.

2.9.2. Quantity and Air Horsepower due to NVP

NVP is independent of the quantity of circulating air and the quantity that will circulate is determined by the resistance only. However, Atkinson's equation (2.5) and air horsepower equation (2.8) still holds.

2.9.3. Natural Ventilation Pressure and Fans

Since NVP acts on the same circuit as the fan, it can be treated as acting in series with the fan. However, it can act in the some direction as the fan and aid the fan or act in the opposite direction and hinder its performance. Fig.2.13 shows the fan, NVP and mine characteristics. Three effects can be demonstrated with the aid of fan NVP characteristics and mine.

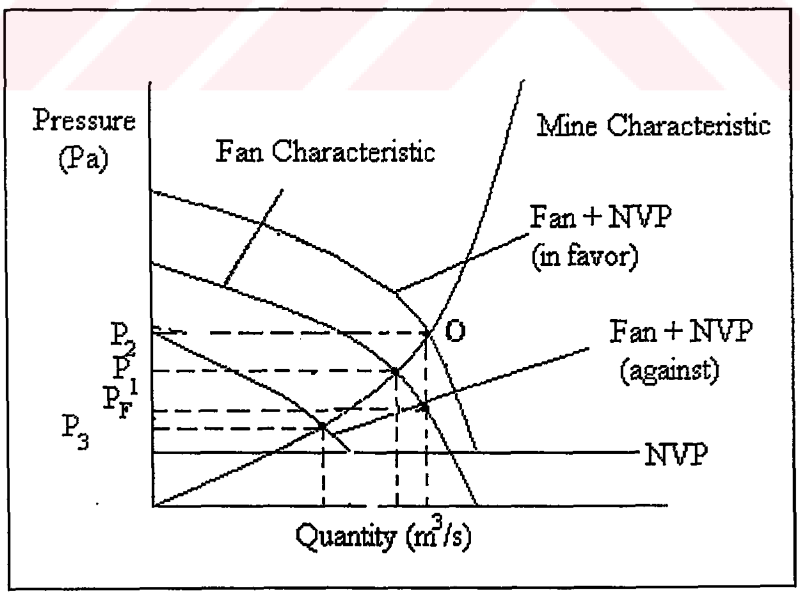


Figure 2.13 Mine and fan characteristics with NVP

When NVP is acting in the same direction as the fan, the combined characteristic is obtained by adding the NVP to the fan pressure at each quantity. The quantity circulating in the system increases but pressure developed by the fan is lower than P_2. When NVP is acting in the opposite direction to the fan, the combined characteristic is obtained by subtracting the NVP from the fan pressure at each quantity. The quantity circulating decreases. However the head developed by fan is higher than P_3. If NVP can significantly affect the operation of the fan, the magnitude must be known (Güyagüler and Güngör, 1999).

CHAPTER III

FUNDAMENTAL ASPECTS OF DUST

3.1. Introduction

The term dust is applied to suspensions of finely solid particles in the air. Instability, coagulation and precipitation are their special characteristics. Dust includes a wide range of particle sizes from over 1 mm. to under 1 μm. The size of the particles considerably influences their behaviour. Dust particles are usually divided into three classes (Sengupta, 1992):

i. Particles greater than 10 μm, which settle according to the laws of gravity, with increasing velocity in still air.

ii. Particles between 0.1 to 10 μm, which in still air settle at a constant velocity, the uniform settling velocity depends on particle size and density, viscosity of the medium and acceleration due to gravity.

iii. Particles between 0.01 and 0.1 μm, which do not settle but diffuse in the air and remain in a colloidal state.

A dust cloud consisting of particle 5 μm in size is not visible to the naked eye. It cloud not be seen in the strongest beam of light underground, but would appear as motes in a bear at sunlight.

3.2. Classification and Harmfulness of Dust

Dust can be classified according to their harmful physiological effects. The following list different types of dusts in order of decreasing harm in each category:

i. Fibrogenic dusts (harmful to the respiratory system)

ii. Carcinogenic dusts (cancer – causing)

iii. Toxic dusts (poisonous to human organs)

iv. Radioactive dusts

v. Nuisance dusts (few adverse effects)

Any dust, if present in excessive quantities and inhaled for a sufficiently long time, can cause physiological damage. Factors, which determine the harmfulness of any dust are its composition, particle size, concentration in mine air, exposure time and susceptibility of the exposed individual.

Mineralogical and chemical composition are important. Free silica is more damaging than combined silica. The surface energy of the particles can be determining factor and solubility is a very important variable in toxic dusts.

Dust concentration in mine air is expressed either in terms of the number of particles per unit volume of air as particles per cubic centimeter (ppcc) or on the basis of weight per unit volume of air as milligram per cubic meter (mg/m^3).

In assessing the pulmonary effect of dusts the concept of respirable dust is important. At present two criterias are accepted for defining respirable dust. Both criteria approximate dust deposition in the noniliated portions of the lung. The first resulting from work performed by the U.S. Atomic Energy Commission (AEC) and the other recommended by British Medical Research Council (BMRC) are defined by the sampling efficiency curve labeled in Figure 3.1.

Particle sizes refer to equivalent diameter which is defined as the diameter of a spherical particle of unit density having the same falling velocity as the particle in question.

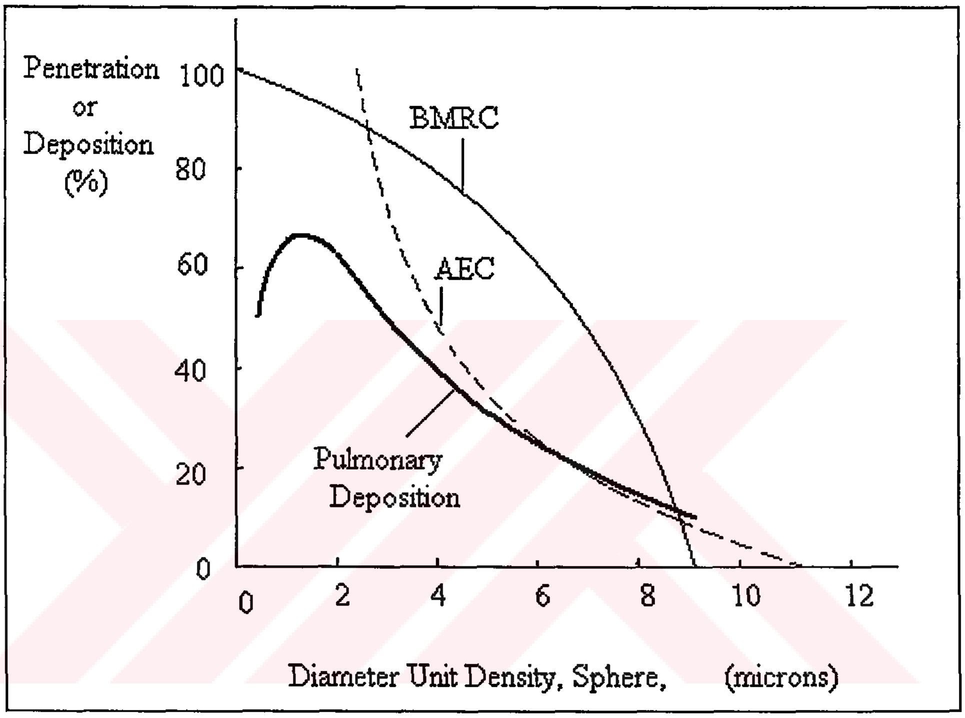


Figure 3.1 Pulmonary deposition curve and comparison of respirable dust criteria (Morse et al., 1970).

3.3 Generation of Dust

Dust is generated and dispersed into the mine air through rock breakage, rock loading, transportation and unloading, and through the flow of ventilating air. The quantity of potentially airborne dust produced is related to the quantity of rock broken. There are two phases involved, cutting the chip and removing it from the rock face. In the cutting process the production of potentially airborne dust is inversely related to size of the rock chips (Knight, 1980).

In order to reduce the formation of such dust, the intensity of the cutting force should be applied a small area with a greater depth of cut. The design of cutting tools achieve this, is complex. The minimum specific energy consumption in rock breaking and the minimum specific dust generation are usually achieved simultaneously. The chip-removal process in drilling can generate airborne dust. Insufficient clearance or not enough flushing fluid can produce further breakage of chips, decrease cutting speed and energy loss in the drill rod with increased specific dust generation.

The handling of the broken rock has a significant effect on the dispersion of the fine dust in mine air. The quantity of dust dispersed in handling process is determined by the amount of fine dust created in breaking process. The extent to which the material is disturbed in handling, the energy expended to create new breakage, the kinetic energy imparted to the freshly broken rock particles by the stress relief in the material broken and ambient air velocities.

3.4 Measurements of Dust Generation in Underground Mines

Underground hard rock mining consists of breaking the ore into small pieces, loading, transportation and handling, drilling and ancillary operations. Measurements and laboratory analyses were performed to determine total respirable dust in these operations. Sampling stations were located in the airways both upstream and downstream of operation. The sampling station locations were chosen to give the most uniform mixture of dust and air possible.

The total amount of dust produced at each operation was determined by multiplying the dust concentration at each return station by the total airflow in the airway and subtracting the same figures obtained in the intake airway. This figure was then divided by the unit of production, tonnes of ore, length of holes drilled or other units to give specific values per unit of production.

Three main categories of instruments are used in underground mines to measure concentration of respirable dust in mine atmosphere. These are gravimetric dust samplers, short-term dust monitors and instrument for particle size measurements.

3.4.1 Personal Dust Sampler

The personal sampler consists of an air pump, filter assembly and a small cyclone that acts as a size selector to remove nonrespirable dust particles. In the cyclone, air enters a cylindrical chamber tangentially and passes inward through a spiral path with increasing velocity. Because of centrifugal force, large particles are removed from the airstream and collected at the bottom of the cyclone chamber. Respirable dust penetrating through the cyclone is collected on a filter paper. The filter is weighted to determine the mass of the respirable dust.

The preweighted capsulated filter is weighed after sample measurement. The weight gain due to dust is determined by extracting initial weight from final weight. Dust concentrations are expressed in milligrams of dust per cubic meter of air sampled. The calculation is as follow;

$$\text{Dust Concentration} = \frac{\text{Weight Gain}}{\text{TR}} \ \text{mg/m}^3 \tag{3.1}$$

Where;

T = Time of sampling in s,

R = Rate of sampling in m^3/s,

Weight gain in mg.

Personal gravimetric samples have several advantages over other sampling methods. A miner can wear the sampler, permitting sampling of the dust cloud to which he is exposed. The sampler provides an estimate of the amount of respirable dust in the mine air for periods of up to 8 hours.

3.4.2 MRE Gravimetric Dust Sampler

The MRE is a portable gravimetric dust sampler that continuously samples air in a working mine and separates the airborne dust into respirable and nonrespirable fractions. This separation occurs as the larger particles settle to the floor of the horizontal elutriator (size-selector) sections and are prevented from reaching the filter smaller particles having slower settling rates pass through the elutriator and are deposited on the filter. The instrument consists of an elutriator, a filter, and an electrically driven pump.

Dust-laden air enters the precision-made four-channel horizontal elutriator at an adjustable flow rate. The rear of each channel floor plate is lipped to help prevent nonrespirable dust from being accidentally dislodged from the plates onto the filter.

A nose restrictor is fitted at the entrance to minimize the effects of air cross flow. A diaphragm pump, fitted with flap values of aluminum-coated polyester film, induces airflow. An adjustable crank varies the pump output, and the unit is driven through spurges at half speed by a governed electric motor. A flowmeter is connected to the output side of the pump through a diaphragm-type-smoothing device.

There are many other types of dust sampling instruments. These are GCA Dust Monitor, TSI Piezobalance, Sibata Digital Dust Monitor, The Ram-1, Konimetry, Midget Impinger, Impinger-Microprojector-Coulter Counter, Thermal Precipitator, and Thermal Precipitator-Diffraction Size Frequency Analyzer.

3.5 Dust Control Methods

Dust control methods in mine environments include efficient ventilation systems, use of water sprays and use of dust collectors and scrubbers.

3.5.1 Ventilation

Ventilation of the working place is the most important measure for dust control. Settling rate of airborne dust particulate less than 10μm in diameter show that dust in the respirable size range behaves in the same manner as gaseous contaminants and may be transported for great distances by air currents. Respirable dust generated at the working face can be transported by the same airstream, which is used to remove other airborne contaminants such as mine gases.

Ventilation techniques used to control respirable dust can be divided into three general types: conventional face ventilation, secondary blowing ventilation and secondary exhaust ventilation. Conventional face ventilation uses ducting and sometimes brattice cloth to direct the intake airstream toward the face or exhaust the dust-lades airstream away from the face. Secondary blowing ventilation uses machine mounted diffuser fans to push the airborne dust away from the workers. Secondary exhaust ventilation vacuums the dust-laden air from the face through a machine-mounted fan.

3.5.1.1 Conventional Face Ventilation

Face ventilation can be of the blowing or exhaust type. Exhaust ventilation exhausts away the dust and other contaminants through a duct or a brattice line. The dust laden air is confined within the duct. The exhaust ventilation, in order to be effective, should have its duct or brattice line as close as possible to the face.

The blowing system forces a stream of air into the working area through a duct and fan or by a brattice line. The range of influence of a blowing system is considerably higher than it is for an exhaust system, because in the blowing system the air retains its directional effect over a longer distance than in an exhaust system. For a blowing system 10% at the face velocity is reached at a distance of 30 diameters from the blowing opening, whereas in an exhaust system the same velocity is reached only at a one-meter distance from the exhaust opening. Extensible duct and brattices are often used. The movement of air into an opening such as an exhaust duct requires a pressure difference to accelerate the air to the required velocity and to overcome some turbulence loss at the duct opening.

3.5.1.2. Secondary Ventilation

In secondary ventilation, the newly formed dust is vacuumed from the vicinity of the face and then either discharged into the return via a duct or passed through a machine mounted dust collector which discharges the effluent air from the back of the machine. Secondary ventilation may face difficulty because space constraints on mining machines often severely limit the configuration of the air ducting, the location of air intakes, and the installation of the fan and dust collection on the machine. Secondary airflow must be matched to the airflow into the entry in order to avoid local recirculation problems.

Total dust reduction efficiency of a secondary ventilation system involves the dust capture efficiency of the air intake and dust collector and the interaction between the secondary and face airflow systems. A secondary ventilation system tends to be more expensive than the use of water sprays (Sengupta, 1992).

3.5.1.3 Ventilation Air Quantities

Main and auxiliary ventilation systems are used to dilute and control dust concentrations in mine air. Air quantities needed to dilute the dust concentration is given by the following equation (Hartman and Mutmansky, 1982):

$$e^{-\left(\frac{Q}{V}\right)^{-T}} = \frac{(G+BQ)-XQ}{(G+BQ)-X_0Q} \qquad (3.2)$$

Where;

G = Dust generation rate in mg/min.,

Q = Quantity of ventilating air in m^3/s,

V = Volume of space in m^3

T = Time in min

B = Dust concentration in normal air in mg/m^3,

X = Dust concentration in the mixture in mg/m^3,

X_O = Dust concentration in the intake air in mg/m^3.

The above equation can be simplified as time T approach infinity ($T \to \infty$) for weigh;:

$$Q = \frac{G}{TLV - B} \qquad (3.3)$$

Where TLV is allowable dust concentration

3.5.2. Dust Supression with Water Spray

Water is used in mining to wet the dust and to bond the sedimentary dust in order to avoid the development and whirling-up the respirable dust. Water immobilizes the dust and prevents it from being airborne and collect airborne dust. This is done by water infusion into drill holes (wet drilling) and by spraying water onto broken ore during loading and transport. To reduce whirling-up tendency of the dust, the low adhesive forces at dry dust particles must be increased substantially. The best effect can be expected when the voids between individual particles are just about filled with water because the capillary forces developed hold the particles firmly together. However, these forces disappear when the water content is too high.

The surface tension of the water then present is no longer sufficient to bind the dust particles firmly. Slurry is formed and under high air velocity the dust-laden water can be atomized with the result that airborne dust develop again. The volume of water necessary to obtain the optimum effect depends on the size distribution and type of dust and on the properties of water. The effect of water can be improved by use of wetting agents.

The capture efficiency of water droplets for the sedimentation of dust particles in the air current is a function of the relative velocity between the water droplets and the dust particles, and the ratio of the diameter of the dust and water particles. For dust particles between 1 and 5μm, the water droplet size should be about 0.05 to 0.3 mm (Chanchi et al., 1996). The capture efficiency diminishes rapidly with the decreasing size of dust particles and droplets. Fine water particles in the size range of mists have hardly any effect on the sedimentation of dust particles. The reason is the coagulation of small particles, which hardly contributes to an increase of efficiency. Therefore, the number of dust particles and water droplets is often too low and time available is too short (Cheng, 1973).

The low relative velocity between the water droplets and dust particles contributes to the low degree of efficiency of water sprays. In order to improve the efficiency of water spraying, energy must be applied to increase the relative speed between water droplets and dust particles (such as venturi tube, compressed air et etc.) and to increase the falling speed of the particles (as in the centrifugal separators). Dust cloud with high concentration of fine dust particles should be exhausted with suction devices.

3.5.2.1. Mathematical Models

A spray water drop collides with a dust particle as shown in Figure 3.2 and the particle drop agglomerate drops to the floor or is collected by other processes. For uncharged dust particles greater than 0.5 μm in diameter, inertial impacting is regarded as the major collection mechanism. The overall collection mechanism of the spray is given by:

$$E_0 = 1 - \exp\left(- \frac{3\eta L Q}{2 D Q_g} \right) \tag{3.4}$$

Where;

E_0 = Overall collection,

η = Capture efficiency %,

L = Characteristic length for the total capture process in mm.,

D = Mean drop diameter in mm.,

Q = Volume flow rate of water component of dust cloud in m^3/s,

Q_g = Volume flow rate of air component of dust cloud in m^3/s.

Optimum collection efficiency for drops' failing at gravitational terminal velocity is obtained with drops about 1000μm in diameter and collection efficiency increases with particle size.

A water spray ejected from a high-pressure nozzle usually has high collection efficiency because the drops have a high initial velocity and a long transverse distance before air resistance slows them to their terminal velocity.

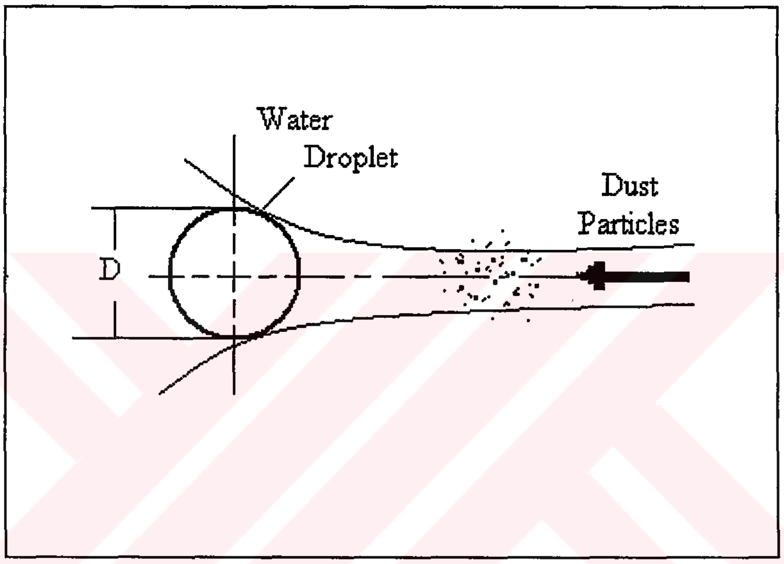


Figure 3.2 Impaction of dust particles with a droplet

3.5.2.2 Open Water Spray

The concept of mean interdrop and inter particle spacing is important in dealing with interaction between two species of particle clouds, water drops, and dust.

The mean interdrop length (S) is given by:

$$S = \left(\frac{\pi}{6\Phi} \right)^{\frac{1}{3}} D \tag{3.5}$$

Where;

D = Mean diameter of the drop in mm.,

Φ = Volume fraction of spray,

S = Interdrop length in mm.

If Q is the volume of water to be atomized by a spray nozzle to fill a volume with a differential length, ℓ, and a cross sectional area, A, then; the number of drop layers, n, along the length can be written as:

$$n = \left(\frac{6Q\ell}{\pi A D^3} \right)^{\frac{1}{3}} \ell^{\frac{2}{3}} \tag{3.6}$$

Where;

D = Mean diameter of the drop in mm.,

Q = Volume of water in m^3/s

A = Cross sectional area in m^2

n = Number of drop layers

ℓ = Spray nozzle length in mm,

Certain important principles should be emphasized when designing water spray systems for dust control. These are:

i. The maximum allowable quantity of water that can be sprayed for dust suppression purposes should be made available.

ii. The dust should be attached as close as possible to its source of origin. Once the dust gets airborne, it becomes difficult to suppress it with water sprays.

iii. The water sprays should be oriented in the direction of the mine ventilation wherever possible to avoid turbulence and recirculation.

iv. An optimum water spray system should consume the minimum quantity of water.

v. Clog-free water circuits and an efficient filtration system require minimal maintenance are essential.

Design of a water spray system involves selecting the type, number, location, and orientation of spray nozzles; an efficient water filtration system and an adequate water supply system with pipes, hoses, and pumps to supply the water requirement at the desired pressure (Sengupta, 1992).

3.5.3. Dust Collectors

A dust collector is a mechanical device used to separate dust particles from a contaminated airstream. The forces or mechanism utilized for dust collection may be classified as; gravity settling, inertial and diffusion depositions, flow-line interception, electrostatic deposition, thermal precipitation, and sonic agglomeration. Dust collectors are available in four types. The performance of a dust collector, commonly termed collection efficiency is defined as η:

$$\eta = \frac{\text{Dust Collected}}{\text{Dust Entered}} \tag{3.7}$$

Performance for a dust collector is given by:

$$\gamma = \frac{\ln\left[\frac{1}{(1-\eta}\right]}{\Delta P_i} \tag{3.8}$$

Where;

η = Collection efficiency,

ΔP_i= Pressure drop expended.

γ = Performance of dust collector

3.5.3.1. Dry Centrifugal Collectors

As the name implies, centrifugal force is the principle used in this type of collector. The centrifugal force applied to a particle varies as the square of the inlet velocity and inversely as the radius of cyclone. This relationship is expressed as:

$$SF = \frac{V}{rg} \tag{3.9}$$

Where;

SF = Separation factor,

r = Radius in mm,

g = Gravity constant.

V =. Inlet velocity in m/s.

Cyclone collectors are the major appliances in this category. In a cyclone separator, the dust-laden air enters the cylindrical or the conical chamber tangentially at one or more points and leaves through a central opening.

The dust particles, by virtue of their inertia, tend to move toward the outside separator wall, from which they are led into a receiver. Dimension of different parts of the cyclone influence efficiency of the cyclone. Cyclone separator proportions are shown in Figure 3.3.

3.5.3.2. Fabric Collectors

Fabric collectors are referred to as bag houses, bag collectors, or clothe collectors. They can be large, on the job-erected installations or small, assembled collectors shipped on the job size. A schematic of a bag collector is shown in Figure 3.4. The air to clothe ratio or meters per second velocity employed through the cloth media is a function of the dust-loading application and design of the cloth collector.

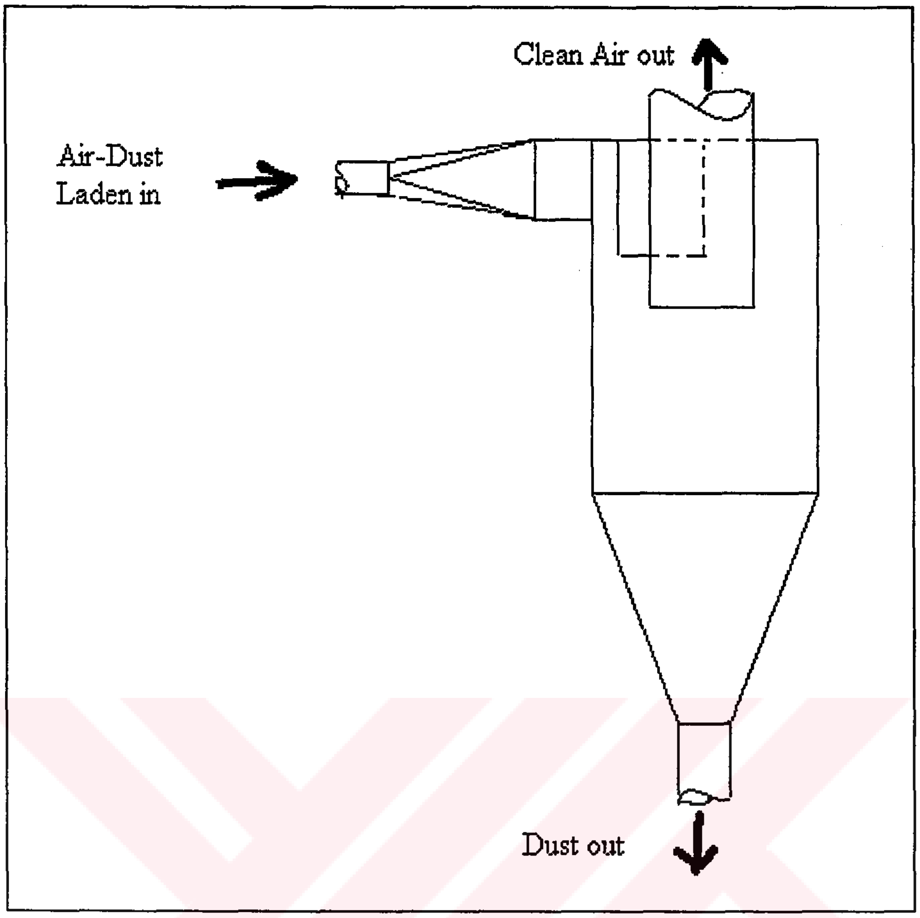


Figure 3.3. Cyclone dust separator

The fabric is usually woven to provide a tight weave, although under close examination the weave reveals wide openings; felted fabric has fewer openings in the fabric structure than the cloth fabric does. The most common fabric is cotton, wool, nylon, orlon, dacron, and siliconized glass fiber.

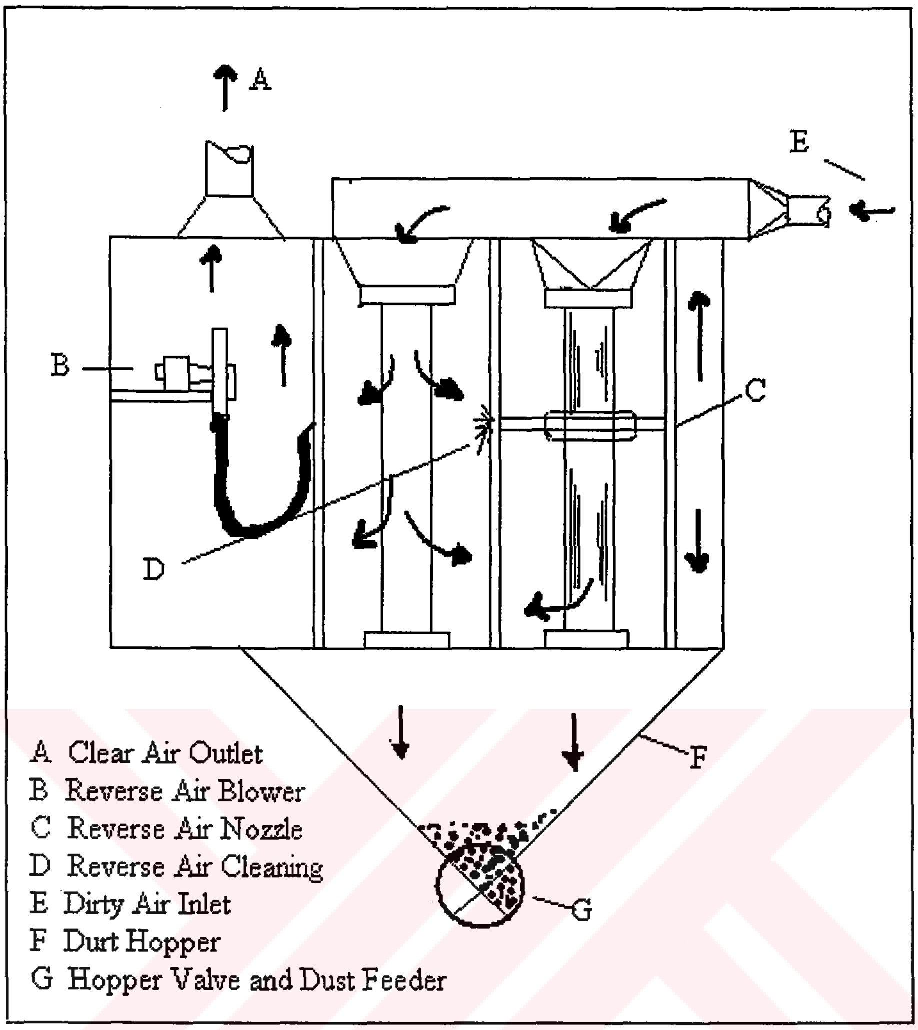


Figure 3.4 Bag filter

The dust collection on the fabric initially starts by the sieving or straining action. Subsequently, diffusion, impingement, interception, and gravitational and electrostatic forces play major roles in achieving dust collection efficiency.

The size and shape of the filter element is a function of the cleaning employed, the consumption on the filter media, and the maximum cloth area in the fixed space available. Five commonly used methods to clean the fabric are shaking, reverse jet, pulsejet, simple collapse, and reverse air backflow.

3.5.3.3. Electrostatic Precipitator

Dust particles are given an electrical charge, positive or negative. Under the influence of the electrical field, the charged particles migrate toward the electrode with an opposite charge. The collecting electrode is often a grounded plate with the ability to collect the dust and discharge it into a hopper.

There are two general classes of electrical precipitators: single stage and two stages. In single stage precipitators, ionization and collection are combined; in two-stage equipment, ionization is achieved in one portion of the equipment, followed by collection in another. A schematic of an electrostatic precipitator is given in Fig 3.5.

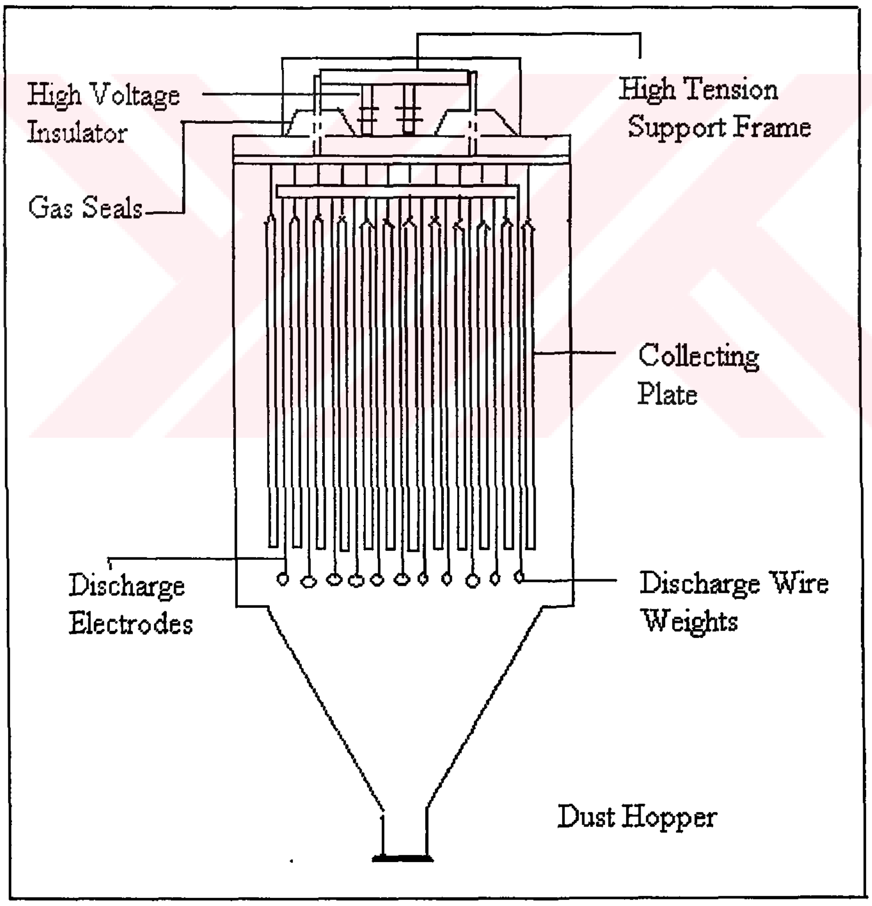


Figure 3.5 Electrostatic filter

3.5.3.4. Wet Collectors

Various types of wet collectors or scrubbers are available for use with mining machines because of their small size, low cost, simplicity, and safety. The height and width of the scrubbers should be the minimum possible so that the scrubber can be mounted on the mining machine for dust extraction. Flooded fibrous bed-type, small diameter cyclone, wetted fan and wetted brush scrubbers are specifically suitable for underground use with mining machines.

The basic scrubber, as shown in Figure 3.6, consist of a fan, short duct transition at fan inlet, and a rectangular housing that holds a knitted wire mesh bed.

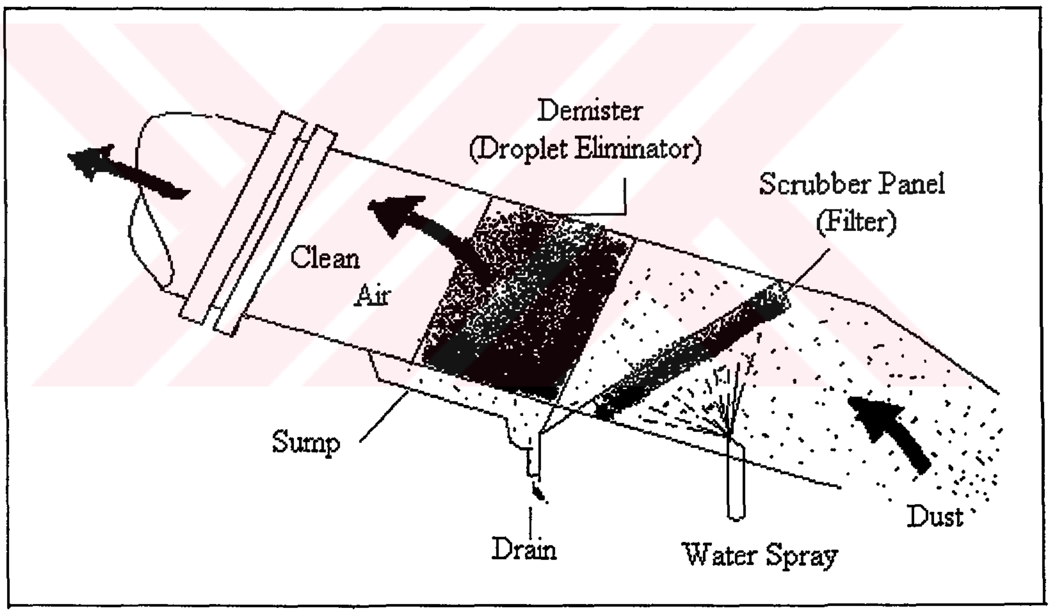


Figure 3.6. Flooded bed scrubber

Dust air is drawn through the wire mesh bed spray nozzles are installed in the duct upstream of the mesh to insure reasonably complete and uniform wetting of the mesh. Slurry drain slots with appropriate pipe connections are installed on both bottom sides of the wire mesh housing (Chironis, 1977).

The small diameter cyclone scrubber consists of a bank or panel of side by side cylindrical cyclone tubes. Each tube has an air volume capacity depending on the tube diameter and the air pressure differential across the tube. Stationary vanes in each tube give a spinning action to the airstream. Particulate matter is thrown to the side of the tube where it can either be bled off in a separate airstream or impact on the wetted vanes or sidewalls of the tube. Water sprays are placed upstream of the panel to insure complete wetting. The wetted fan scrubber consists of a heavy-duty fan, of a centrifugal or vane axial type, with a water spray into the fan inlet.

A bed of wire or plastic brushes is added to the wetted fan scrubber in order to improve the efficiency by utilizing the dust impact benefits of the small diameter wire of the flooded fibrous bed scrubber (Sengupta, 1992).

3.6. Dust Behavior in Mine Air

An airborne dust cloud is a complex system containing particles of different sizes, shapes, densities and states of aggregation. The behaviour of the dust cloud is dependent on several complex mechanisms. The net interaction determines the ambient concentration of dust in mine air.

3.6.1 Settlement of Dust in Mine Air

The rate of settlement of dust in air is dependent on its size and density, the velocity of air, and upon other factors such as Brownian motion and coagulation. In the higher velocity air normally encountered underground, momentum also plays a part.

Brownian motion is caused by bombardment of small particles by air molecules. These impacts do not cancel out but give the dust particles a resultant motion in some direction. The rate of fall is given by the Stokes-Cunningham

Equation:

$$V = \frac{2wgr^2}{9\gamma}\left(1 + A\frac{e}{r}\right) \qquad (3.10)$$

Where,

V = Velocity of fall in m/s,

W = Density of particle kg/m^3,

g = Acceleration due to gravity in m/s^2,

γ = Viscosity of medium,

r = Radius of particle in mm,

e = Mean free path of molecules in air in mm,

A = Constant.

3.6.2. Dust Deposition

Dust deposition takes place due to the gravitational, electrostatic, thermal, sedimentation, Brownian and eddy diffusion, and inertial impaction effects. In conditions of turbulent airflow, sedimentation, inertial impaction, eddy diffusion, and Brownian diffusion are the major mechanisms of deposition. Inertial impaction is important in bends in mine airways. Convective diffusion is the dominant mechanism in the 0-to-1 μm range and inertial deposition to be the most important for particle sizes greater than 1 μm.

3.6.2.1. Deposition by Convective Diffusion

The Friedlander model describes turbulent diffusion of particles by a one-dimensional deposition flux equation. The model assumes that the particle deposition on the walls of a pipe can be described by the diffusion of particles to the boundary layer followed by a final free flight. The equation of the diffusion flux is;

$$N = (D_p + \varepsilon)\frac{dc}{dy} \tag{3.11}$$

Where;

N = Deposition rate,

D_p = Brownian diffusivity,

ε = Eddy diffusivity,

C = Concentration at center of duct,

y = Distance from source of deposition.

3.6.2.2. Turbulent Diffusional Deposition

With particles larger than 1 μm, this is the dominant mechanism of deposition. Real-life systems of particles are polydispersed and can be specified by a function of the type;

$$N_0 = \int_0^\infty n(r)dr \tag{3.12}$$

Where;

N_0 = Bulk concentration,

n(r) = Number of particles of radius r.

In the case of distributed particle sizes where the distribution is given in the form of a graph of proportion by weight of particles below each value of particle diameter, discretization of the size distribution is needed. The size spectrum is divided into small classes, with a representative particle diameter (d) and weight (W_d) for each class. The total deposition rate is given by,

$$N = N_0 V_x \sum_{d_1}^{d_{max}} W_d V_d \tag{3.13}$$

Where;

V_d = Dimensionless deposition velocity for diameter d,

V_x = Friction velocity.

3.6.2.3. Inertial Deposition on the Floor

The effect of gravity is also considered for particle deposition on the floor. The rate of deposition due to gravity depends on the terminal velocity. The deposition velocity due to gravity is equal to the terminal velocity.

3.6.3. Coagulation

Coagulation of dust particles occurs when two particles collide to form a larger particle. The shape, volume, and orientation of the colliding particles determine the volume and shape of the resultant particle. A model for the rating the changes of the particle number in any size range is given by:

$$\frac{d \cdot n_k(t)}{dt} = \frac{1}{2} \sum_{i=1,(i+j=k)}^{k-1} K_{ij} n_i n_j - n_k \sum_{i=1}^{\infty} K_{ik} n_i \tag{3.14}$$

Where;

$n_k(t)$ = Number of particles of size k at time t,

K_{ij} = Collision frequency function between particles of sizes i and j.

The first term on the right side of the equation represents the gain in size k from collisions between particles i and j, and the second term represents loss from the k^{th} class by collisions between particles i and j. The factor one half is introduced because each collision is counted twice in summation.

The model assumes that when two particles collide the probability of sticking together is 10. The efficiency of sticking together is dependent on shape, size, and nature of the particle surface as well as electrostatic and Vander Walls Forces.

3.6.4. Dispersion of Dust

According to Skubnov's determinations, the coefficients of turbulent transfer are similar, irrespective of the nature of the flowing fluid or the diffusing coefficient. The empirical formula for longitudinal dispersion is stated as;

$$E_x = 15.8\,UDSc_i^{-0.6}Sc_t\sqrt{\frac{\lambda}{\lambda r}} \tag{3.15}$$

The value of the turbulent Schmidt number is 1.1 for dust. The value of Sci. the molecular Schmidt number is 0.72. These values relate to workings with a steady, uniform velocity profile in the transverse direction. Where the velocity profile is nonuniform the coefficient of longitudinal diffusion may be larger than with the uniform velocity profile.

3.6.5. Modeling of Dust Flows

Hwang has developed a model for the calculation of dust concentration in underground mines. For dust source in the airway the concentration equation is given as:

$$c_2 = c_1 e^{-BL} + \frac{Q}{B}\left(1 - e^{-BL}\right) \tag{2.16}$$

Where;

c = Dust concentration,

B = A factor that incorporates flow velocity, terminal velocity and dimensions of the airway,

Q = Dust production related term,

L = Length of airway under consideration.

3.6.5.1. Numerical Modeling of Dust Flow

The transport and disposition of dust can be modeled following Hidy's statistical equation for turbulent diffusion. In the turbulence of mine air, significant variation in dust concentrations across the cross sectional of the mine airway is not expected. However, in mine airways, the concentration of dust varies to some extent in the vertical direction. The dust concentration in the upper half of an airway is generally higher than that in the lower half. A one-dimensional population balance equation of the convective diffusion form for flow in x direction is presented as:

$$\frac{\partial c}{\partial t} = E_x \frac{\partial^2 c}{\partial x^2} - v\frac{\partial c}{\partial x} + r - s \tag{3.17}$$

This equation requires to be solved for different particle size ranges, using representative particle diameters for each size range. The behavior of the size range is summed up to give the general behavior of the dust cloud.

3.6.5.2. Initial and Boundary Conditions

The initial condition to solve the equation is,

$$c(x,t) = 0 \quad \text{for } t = 0 \qquad 0 < x < L \tag{3.18}$$

Where;

L = Length of the airway.

It is assumed that there is no dust concentration in the mine air initially. The boundary condition can be determined by assuming that at the point of mixing where sectional airways meet with the high velocity exhaust airways, the concentration gradient does not vary with distance. Therefore, it can be assumed(Sengupta, 1992):

$$\frac{dc}{dt} = 0 \tag{3.19}$$

CHAPTER IV

FIELD STUDIES ON VENTILATION AND DUST SURVEYS

4.1. Introduction

Eti Holding Eastern Chrome Establishment in Guleman lies 76 km to the East of Elazığ province and is within Southeast Anatolian thrust belt, Turkey. Fig.4.1 shows the location of the mine area.

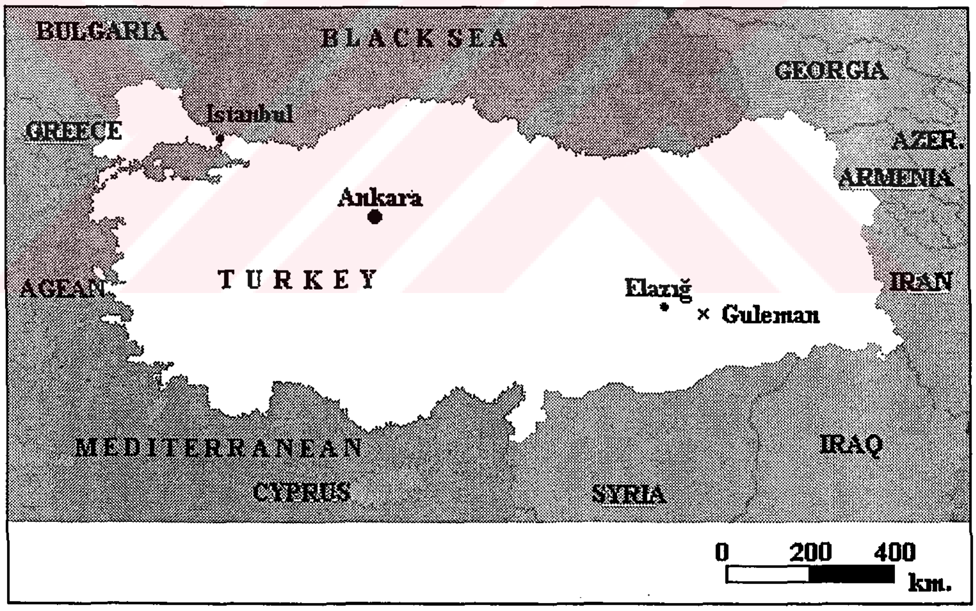


Figure 4.1 Location of Eti Holding Eastern Chrome Establishment

Eti Holding Eastern Chrome Establishment is the most important chromite ore-producing mine in Turkey. Abdullah Guleman, a mine engineer, discovered the chromite deposits in 1935. After prospecting and exploration work by Mineral Research and Exploration Institute (MTA), the actual mining operation in the

Guleman started in 1953. The establishment has twenty-six underground mines and two open pit mines in operation. It has also concentrating and processing plants. The mine area comprises 200 km^2 and has East-West elongation (Arıkal et al., 1989). In this study, the Kef district of the mine, which has a 6 km^2 was chosen for field studies described in this thesis. The main rock units of the Guleman are shown in Figure 4.2. They can be divided into three groups as:

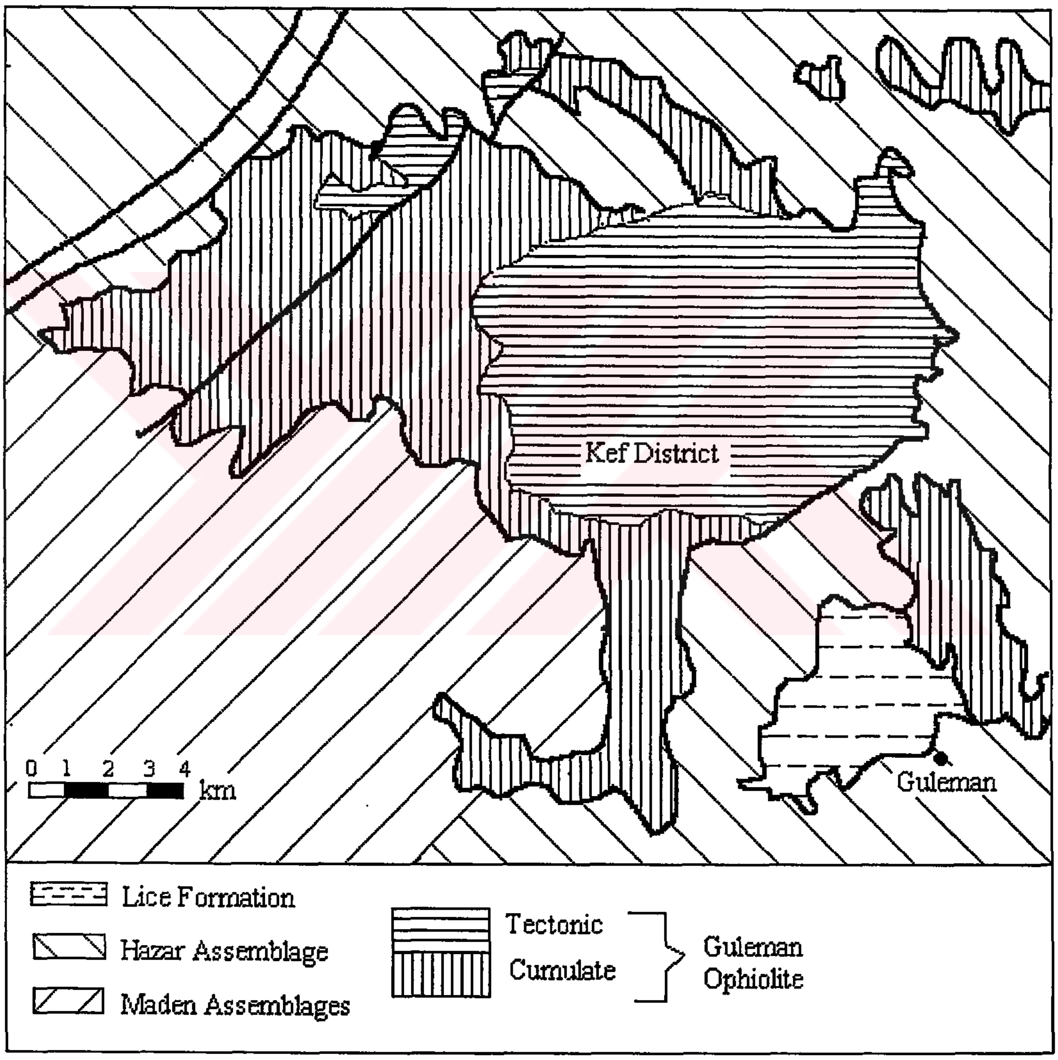


Figure 4.2. Geological map of the Guleman district (Engin and Sümer, 1987)

i. Autochthonous Arabian Platform rocks, Lice formation of Miocene age.

ii. Allochthonous rocks, Guleman ophiolite of lower cretaceous-upper Jurassic age.

iii. Neoautochthonous rocks, the Nacaran limestone of upper Miocene age and alluvium.

In the Guleman district about 10 million tons of chromite ore were produced by the end of 2000. The district has about 6 million tons of unmined chromium reserve. The Kef district contains 80% of the total ore deposit explored in the region. Figure 4.3 shows geological cross-section of the study area. It is mainly composed of harzburgite and minor dunite.

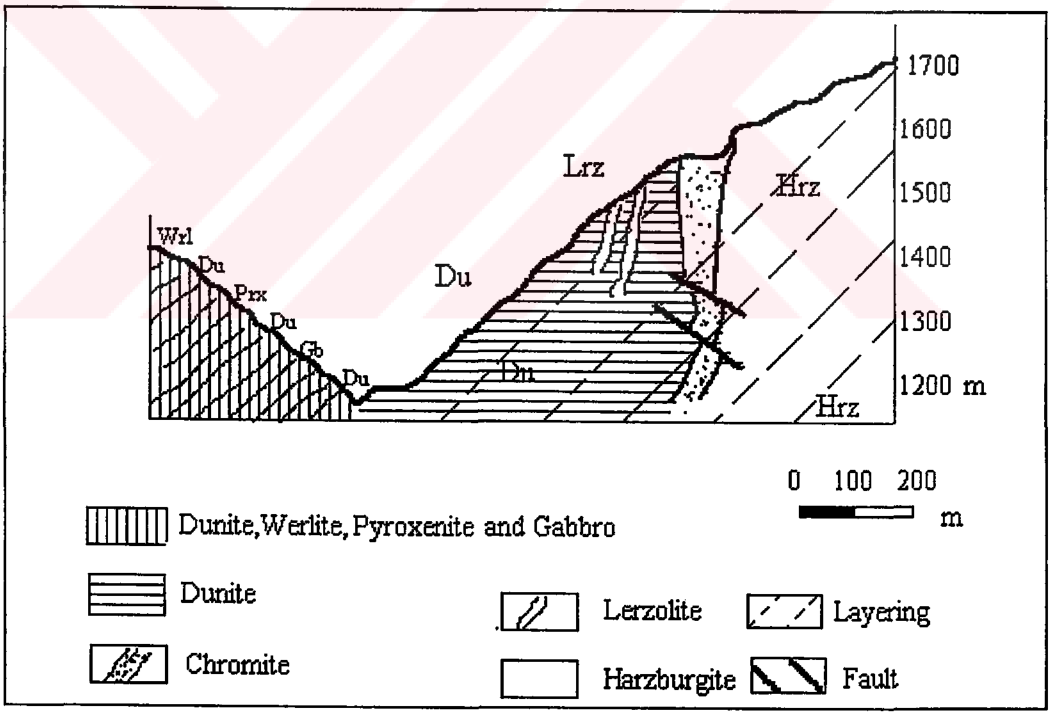


Figure 4.3. Geological cross-section of the Kef deposit (Engin and Sümer, 1987)

It also includes chromites that are generally present as lenses. Granulation, crystal kicking, foliation and compositional layering are their main characteristics. The dunites around the Kef reveal textural properties different from those associated with harzburgites. Cumulates form magmatic series and start with dunites in which chromite layers are frequently present at various levels (Düzgören, 1987).

4.1.1. Geochemistry of Chromite

Chromium is one of the petrogenic or rock-forming elements of the earth crust. It is a typical transition element, so it has a variety of oxidation states. It is twentieth in order of abundance of the elements in igneous rocks and tenth in meteorites. The geochemistry of chromium is characterized by its strongly chalcophile character in the sulfide-metal systems of iron meteorites and somewhat more strongly lithophile character in the lithosphere of the earth.

The mineral chromite crystallizes in cubic system is the only commercial source of chromium. The term chromite is used for all chromium bearing spinels that contain more than 15% Cr_2O_3 and appear to have crystallized as a primary mineral with the associated silicates. Thayer (1970) has pointed out that chromite composition conforms to the general formula AB_2X_4 that may be modified to R^{+2} $R_2^{+3}O_4$ or ROR_2O_3.RO consists principally of MgO and FeO, but includes also MnO, Nio and ZnO. R_2O_3 consists of Cr_2O_3, Al_2O_3, Fe_2O_3 and V_2O_3. Theoretically, the RO and R_2O_3 constituents should balance in primary magnetic chromites.

Chromite deposits are of two general types: podiform and stratiform. All stratiform complexes of the earth show similar structure and lithology with remarkably similar lateral and stratigraphic change in chromite composition. In most podiform chromite deposits, although the bulk ore may differ widely in composition, the composition of cleaned chromite concentrates does not vary much, so the podiform chromite deposits ore petrologically unique. The Guleman chromite ore deposits are podiform types.

4.1.2. Mineralogy and Petrography of the Guleman Chromite

The isotropic character of the Guleman chromites has relatively low reflectivity and considerably high Vicker hardness number, with grayish color. They have subhedral form cataclastic and pull apart textures with small surrounded silicate. Grain size ranges from few microns to over one hundred microns. By increasing deformation, cataclastic and pull apart textures developed; pull apart cracks are opened within the chromite grains and are filled by gangue minerals.

Serpentinization of gangue minerals deforms the grain boundaries of chromites and decreases the size of grains. The Guleman chromites have generally rounded edges; therefore, partial dissolution of chromite most probably took place after the deposition of the previously corroded chromite grains.

Serpentine, olivine and pyroxene were observed as gangue minerals and serpentinization is a common alteration process in the district. The Guleman chromites also contain calcite and minerals that are suspected to be termolite and kammererite.

Texture of the chromite is classified as primary textures related to crystal settling and secondary textures related to magmatic deformation. The primary texture of the chromite is nodular and orbicular. It exhibits well-developed pull apart texture and lenticular cavities within the nodules, which are filled by gangue minerals. Secondary textures are marked by tendency to show layering and lineation due to thinning of chromite and gangue minerals, by showing cataclastic textures, and decreasing grain size, pull apart textures due to stretching of the mineral and especially of nodules. Chemical composition of the Guleman chromites and their structural elements are given in Table 4.1.

Table 4.1. Chemical composition and structural elements of the Guleman chromites

Oxides	%	Structural Elements	%
Cr_2O_3	58.06	Cr	39.72
Al_2O_3	14.56	Al	7.71
MgO	14.12	Mg	8.51
FeO^T	12.87	Fe^{-3}	2.77
SiO_2	4.02	Fe^{-2}	7.23
Fe_2O_3	3.96	Fe^T	10.00
CaO	0.15	Ni	0.16
TiO_2	0.15	Mn	0.07
		Co	0.07
		Zn	0.06
		V	0.01
		Cr/Fe^T	3.97
		Cr/Cr+Al	0.81
		$Cr/Cr+Al+Fe^{+3}$	0.76
		$Al/Cr+Al+Fe^{+3}$	0.18
		$Fe^{+3}/Cr+Al+Fe^{+3}$	0.06
		Fe^{+2}/Mg	0.85
		$Fe^{+2}/Mg+Fe^{+2}$	0.46

The Vickers hardness number of the chromite varies from 1225 to 1416 kg/mm^2 with an average of 1297 ±72 kg/mm^2. The hardness increases with increasing Al_2O_3 and decreases with increasing MgO content. The reflectivity of the chromite was measured in air at 589-nm wavelength, which corresponds, to yellow light. The reflectivity ranges between 12.41 to 13.24%. The reflectivity increases with increasing Cr_2O_3. Table 4.2 lists the composition of the Guleman chromites after normative olivine.

Table 4.2. Composition of the Guleman chromites in normative olivine.

Oxides	**%**
Cr_2O_3	49.69
Al_2O_3	12.47
MgO	12.09
FeO^T	11.02
Fe_2O_3	3.40
FeO	7.96

4.2. Air Quantity Surveys

In order to determine the performance of the mine ventilation system comprehensive air quantity surveys are necessary. These surveys are also essential to the improvement of the current ventilation system if the performance is found to be unsatisfactory

Three dimensional view of the Kef chromium mine is shown in Figure 4.4. A ventilation network, which is a graphical representation of the ventilation system of the Kef mine is shown in Figure 4.5. It consists of junctions and interconnecting lines that denote major airflow routes. There are five main levels and seven sublevels in the mine. The actual country coordinates elevations and numbers of attached branches of all the junctions in the mine were determined. The data are illustrated in table 4.3.

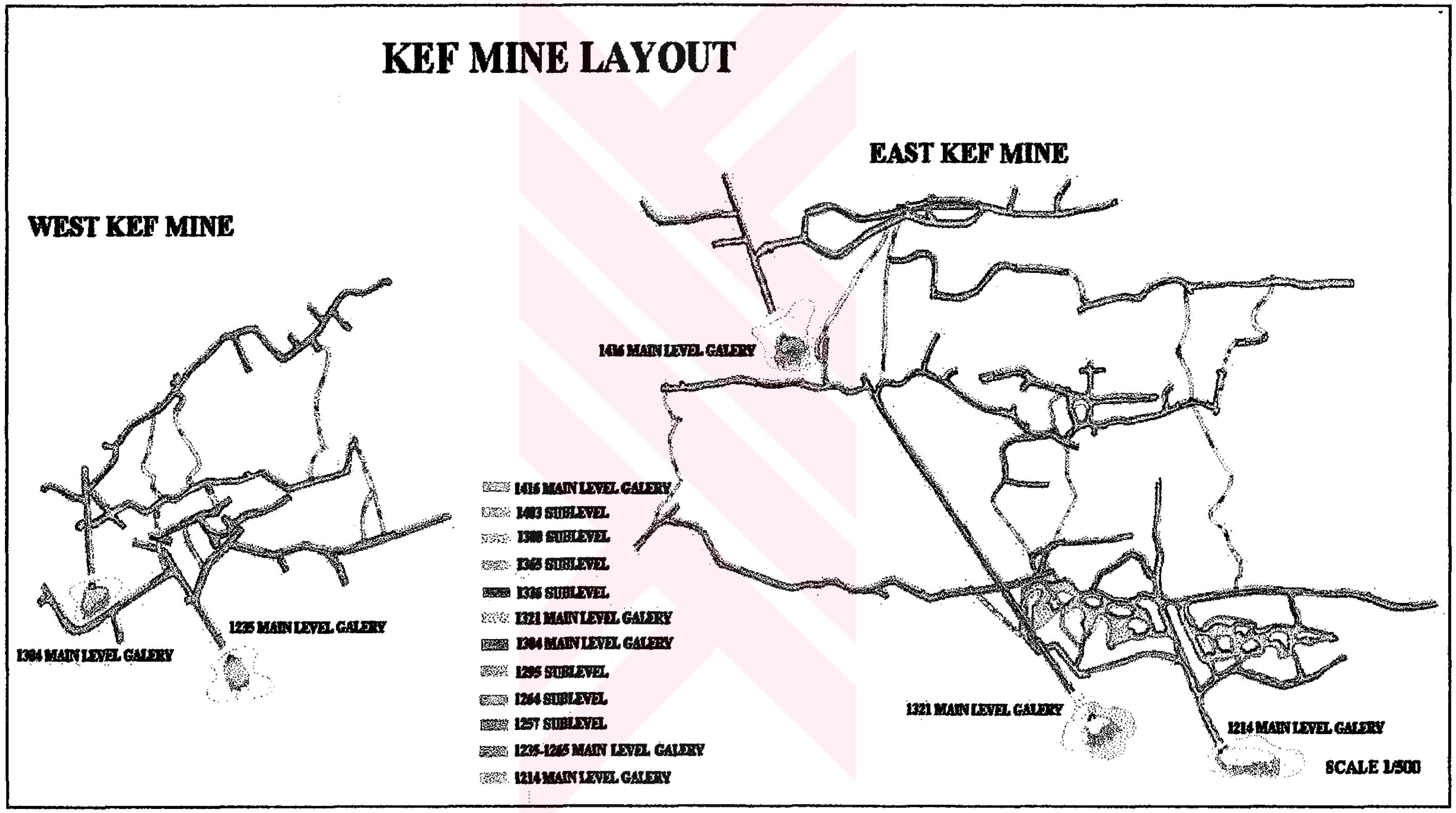


Figure 4.4 Three dimensional view of the Kef Chromium mine

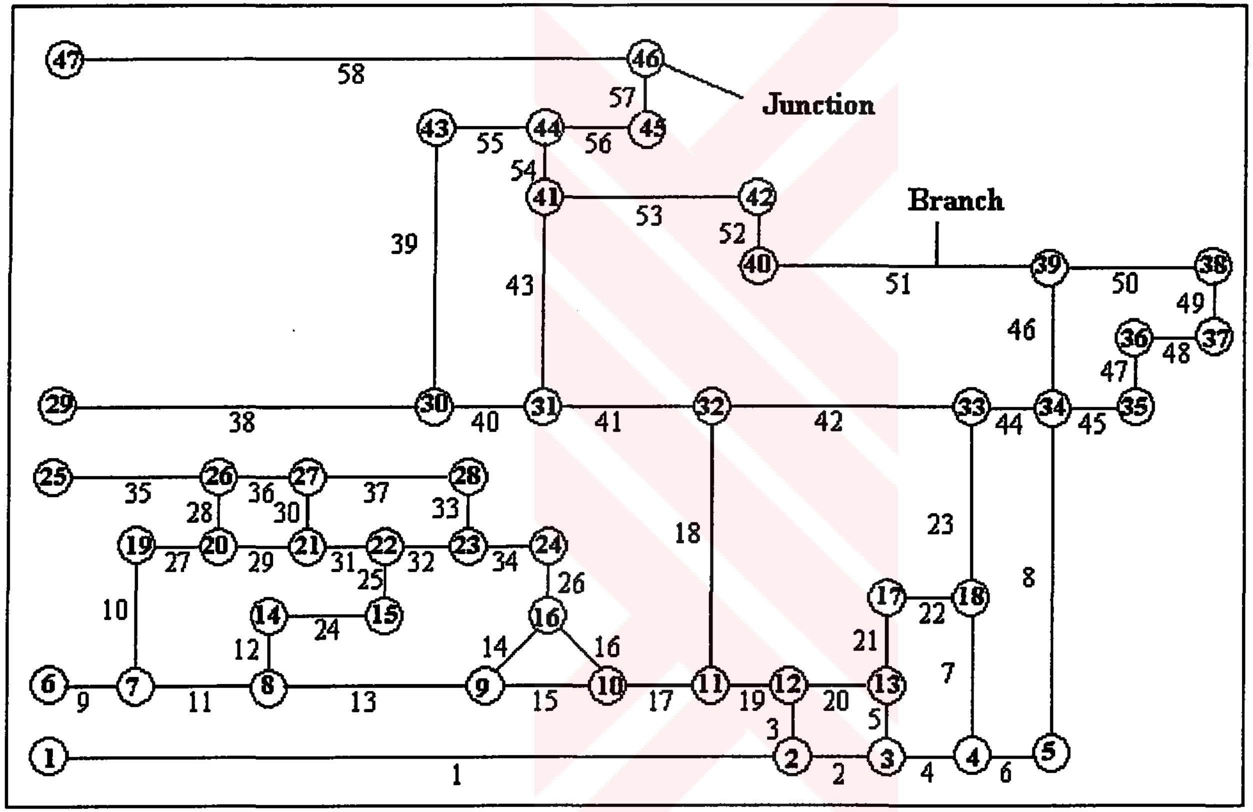


Figure 4.5 Ventilation network of the Kef mine

Table 4.3. The Coordinates of the junctions given in Figure 4.4.

Junction No.	X Coordinate	Y Coordinate	Z Coordinate	Level	Branches Attached
1	66104	62099	1214	1	1
2	65832	62176	1217	1	3
3	65493	62225	1218	1	3
4	65528	62668	1218	1	3
5	65970	62570	1219	1	2
6	65244	61954	1235	2	1
7	65180	62410	1241	2	3
8	65210	62420	1242	2	3
9	65908	63199	1256	2	3
10	65310	62435	1257	2	3
11	66264	61914	1263	2	3
12	65650	62462	1264	2	3
13	66062	62615	1265	2	3
14	65200	62400	1257	3	2
15	65401	62809	1258	3	2
16	65967	62567	1269	3	3
17	65820	62520	1295	4	2
18	66913	62518	1295	4	3
19	65175	62400	1264	5	2
20	65190	62410	1264	5	3
21	64942	62895	1264	5	3
22	65583	63293	1265	5	3
23	66263	62475	1266	5	3
24	65827	62093	1267	5	2
25	65150	62390	1304	6	1
26	65482	63336	1308	6	3
27	66095	63492	1309	6	3
28	66336	62782	1310	6	2
29	66030	62150	1321	7	1
30	65729	61642	1324	7	3

Table 4.3. The Coordinates of the junctions given in Figure 4.4. (continued)

Junction No.	**X Coordinate**	**Y Coordinate**	**Z Coordinate**	**Level**	**Branches Attached**
31	64978	62056	1325	7	3
32	65202	62481	1325	7	3
33	65563	63056	1326	7	3
34	66182	63223	1326	7	4
35	66776	62830	1327	7	2
36	65012	62418	1336	8	2
37	65430	62410	1336	8	2
38	66322	62664	1365	9	2
39	65932	62938	1365	9	3
40	65687	62503	1367	9	2
41	65770	62510	1380	10	3
42	66357	62519	1380	10	2
43	65312	62378	1403	11	2
44	65797	62793	1407	11	3
45	66143	6207	1408	11	2
46	65986	62458	1417	12	2
47	65466	62345	1416	12	1

There are four major areas under the general heading of ventilation surveys. These are quantity, pressure, temperature and air quality surveys. Air quantity is calculated from velocity and area measurements at selected cross section of the mine airway.

4.2.1. Velocity Measurements

The accuracy in finding air quantity depends on the measuring technique and the type of velocity measuring instrument used. Different instruments are used to measure airflow velocity such as; smoke tube, velometers, manometer-pitot tube, katathermometer and anemometers.

Anemometers are the instruments which are widely used in air velocity measurements. There are two main types, the cup and the vane type. The cup type anemometers are mostly used in meteorological work to measure wind speed ranging from 1.0 to 20.0 m/s. The moving air strikes to four hemispherical cups and forces them to rotate. At the same time a counter notes this rotations.

Vane type anemometers are more common in the mine ventilation measurement. Considering the velocity of airflow, anemometers are classified as low, medium and high-speed anemometers. Another classification can be done according to their measuring techniques. They can either be continuous measuring (digital anemometer) or stationary measuring (conventional anemometer) instruments (Güyagüler and Güngör, 1999).

In this study three types of conventional anemometers which were low, medium and high speed anemometers for stationary measuring, and one digital anemometer for continuous measurements were used. For each airflow velocity measurements at least nine spot readings and one continuous reading were done at one station. Figure 4.6 shows the continuous and spot reading procedures.

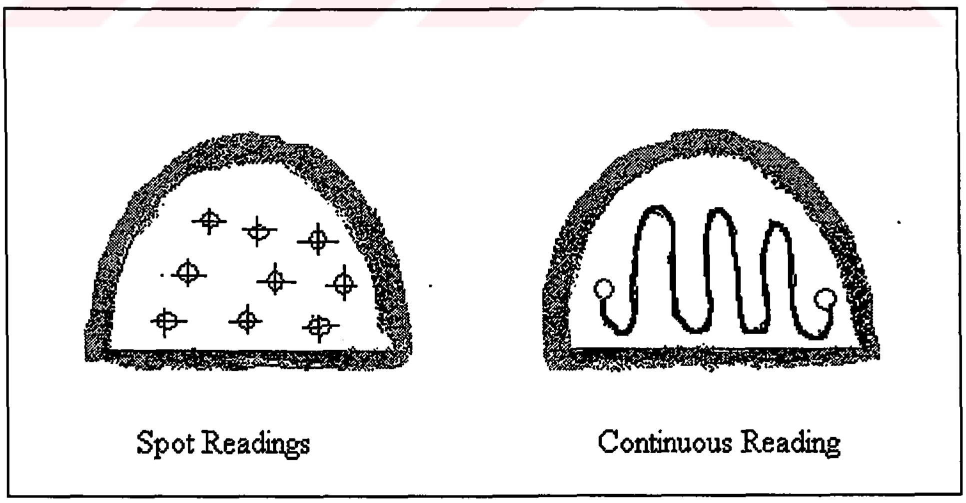


Figure 4.6 Spot and continuous reading procedures

The digital anemometers were well suited to obtain the average velocity in the sections by traversing. When using digital anemometer, integrated or cumulative readings in 15, 30 and 60 seconds were obtained.

The selection of the suitable measuring station is very important in the airflow velocity measurements. In this study, the stations are selected in such a way that, they are as straight as possible. Moreover floor of the mine is clear and the station is not behind any equipment and it is away from the curves. The readings were recorded after reliable results are obtained in which the deviation between individual measurements less than 3 %.

Then, these velocity measurement readings were analysed and an average airflow velocity for each main gallery was determined. Table 4.4 shows average airflow velocity of the main galleries obtained from readings of period one year. Positive values in the table indicate the right airflow direction and negative values indicate airflow in reverse direction.

Measurements of air velocities were done daily at least once or more at each main intake airways, main return airways and working places throughout the year. If airflow direction changes, the measurements of air velocity were repeated three times a day. This situation was faced especially during spring and fall seasons since, the direction of ventilation was very dynamic at these seasons. It was possible to measure airway velocity within a day in the same airway. That is, airflow direction was sometime upward, sometime downward or there was not any air circulation at all.

Table 4.4. Airflow velocity in the main galleries

Months	Average Airflow Velocity (m/s) (+ : in, - : out)				
	1214 Gallery	**1235 Gallery**	**1304 Gallery**	**1321 Gallery**	**1416 Gallery**
January	+0,89	+0,87	-0,30	-0,45	-1,27
February	+0,60	+0,55	-0,21	-0,28	-0,85
March	+0,16	+0,15	-0,02	-0,05	-0,30
April	-0,23	-0,05	0,00	+0,07	+0,27
May	-0,48	-0,13	+0,05	+0,16	+0,54
June	-0,90	-0,42	+0,24	+0,41	+1,02
July	-1,14	-0,67	+0,36	+0,59	+1,18
August	-1,26	-0,88	+0,48	+0,75	+1,27
September	-0,89	-0,36	+0,01	+0,13	+0,95
October	-0,33	-0,09	+0,01	+0,13	+0,38
November	+0,08	+0,07	-0,01	-0,02	-0,15
December	+0,75	+0,71	-0,25	-0,38	-1,06

4.2.2. Area Measurements

Cross sectional area of an airway is important parameter for calculating the air quantity passing through the gallery. Air quantity is found by the multiplication of velocity and the cross-sectional area of the measuring station. The airways in the mine have not regular geometry. Therefore, it must be divided to proper subareas. In the study, cross sectional area calculations were conducted at the stations where the airflow velocity measurements were carried out.

The cross sectional areas were divided into ten subareas as shown in Figure 4.7. The areas of all subareas were calculated separately. Then, area of an airflow was defined by the summation of its subareas.

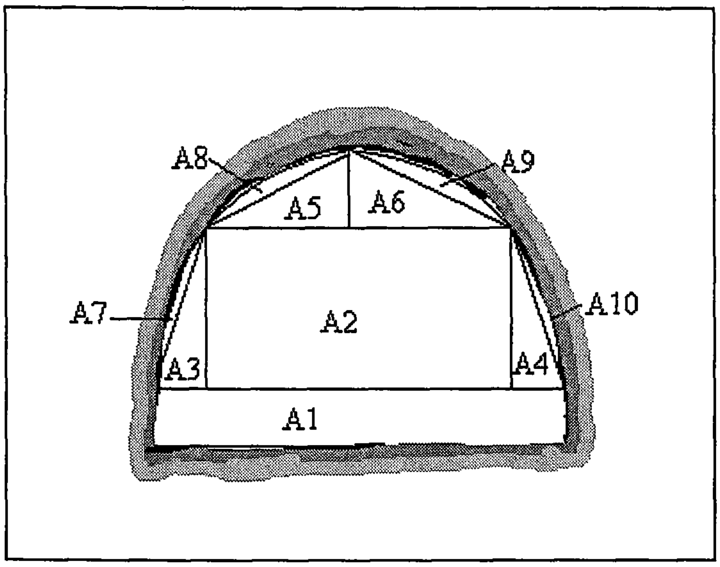


Figure 4.7 Cross sectional area measurement of the airways

Lengths of the airways in the Kef mine were also measured. The results of these area measurements of all airways in the mine are given in Table 4.5.

4.3. Pressure Drop Measurements

In mine ventilation, the pressure difference between the static pressures at two points (pressure loss) due to flowing air in the airway is very important. Pressure loss can be measured in two ways.

i. Measuring pressures at two points by an aneroid barometer and taking the difference.

ii. Using Manometer-Pitot Tube arrangements.

Table 4.5. Areas and lengths of the airways in the mine

Branch No.	From Junction	To Junction	Sectional Area (m^2)	Length (m)
1	1	2	9.22	692
2	2	3	7.54	36
3	2	12	6.47	67
4	3	4	7.54	43
5	3	13	6.47	68
6	4	5	7.54	124
7	4	18	6.18	119
8	5	44	5.96	188
9	6	7	7.52	918
10	7	19	6.15	38
11	7	8	7.52	38
12	8	14	6.14	19
13	8	9	7.52	97
14	9	16	4.87	43
15	9	10	7.52	12
16	10	16	4.96	43
17	10	11	6.85	331
18	11	32	5.16	122
19	11	12	6.85	194
20	12	13	6.85	63
21	13	17	6.34	63
22	17	18	6.68	57
23	18	32	5.94	52
24	14	15	7.36	20
25	15	22	5.82	17
26	16	24	5.76	73
27	19	20	7.46	7
28	20	26	5.74	73
29	20	21	7.46	19

Table 4.5. Areas and lengths of the airways in the mine (continued)

Branch No.	**From Junction**	**To Junction**	**Sectional Area (m^2)**	**Length (m)**
30	21	27	5.83	72
31	21	22	7.46	8
32	22	23	7.49	63
33	23	28	5.96	73
34	23	24	7.44	36
35	26	25	7.88	428
36	27	26	7.67	20
37	28	27	7.64	94
38	30	29	7.20	460
39	30	43	5.61	118
40	31	30	7.20	96
41	32	31	7.20	43
42	32	33	7.20	134
43	31	41	6.35	90
44	33	34	7.06	92
45	34	35	6.94	32
46	34	39	6.27	61
47	35	36	5.88	14
48	36	37	7.14	18
49	37	38	5.98	51
50	38	39	7.22	50
51	39	40	7.22	64
52	40	42	6.34	26
53	42	41	7.43	128
54	41	44	6.42	40
55	43	44	7.64	11
56	44	45	7.64	21
57	45	46	6.52	28
58	46	47	7.14	305

The second way is more efficient due to the fact that pressure drops in the airways are too small that, required precision can only be obtained by these techniques. In this study, pressure drop between two points was measured by a manometer-pitot tube arrangement. This method is shown in Figure 4.8.

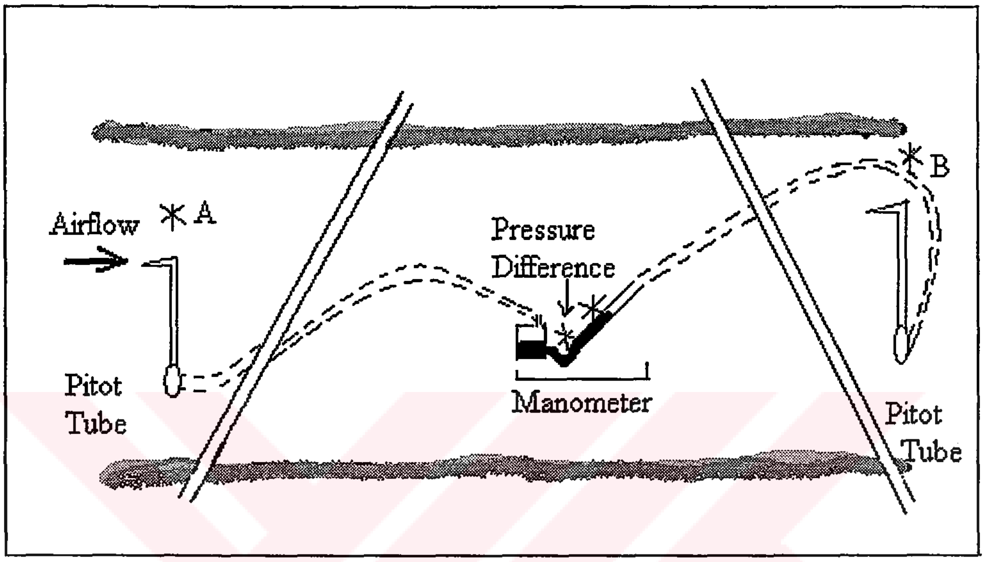


Figure 4.8 Pressure drop measurement by manometer-pitot tube

Pressure difference is measured in terms of head (mm. water gage). A gasoline with density of 0.825 g/cm^3 was used instead of water in the manometer. For recognizing the elevation at the manometer gage scale, the gasoline was colored. Pressure loss measurements were conducted with 100-m long plastic hose for precise measurements.

4.4. Air Temperature Measurements

Natural ventilation is the term used to describe airflow resulting from pressure difference caused by natural means. Natural force, which can create and maintain a substantial airflow, is thermal energy due to the temperature differences. Hence, air temperature measurements have to be done for the evaluation of the existing natural ventilation in the mine.

Dry and wet bulb temperatures of the mine air and the surface were measured throughout one year. The average dry and wet bulb temperatures at all junctions of the mine were determined monthly. The results are given in Table 4.6.

Table 4.6. Temperture Measurements

Month	**Mine Temperature (^{0}C)**		**Surface Temperature (^{0}C)**	
	Wet-bulb	**Dry-bulb**	**Wet-bulb**	**Dry-bulb**
January	11.50	13.50	5.50	6.50
February	11.75	13.50	7.50	8.75
March	13.00	14.75	10.50	12.25
April	13.50	15.00	16.00	18.25
May	13.75	15.25	18.00	20.75
June	14.00	15.50	22.50	26.25
July	14.50	16.00	26.50	30.25
August	14.75	16.50	29.50	33.50
September	14.25	16.00	22.00	25.75
October	13.50	15.00	17.00	20.25
November	10.75	13.50	11.00	14.25
December	11.00	13.25	6.50	9.50

In the mine air, dry and wet bulb temperature measurements, were done by psychrometers. In the study, three different types of psychrometers were used. The first type was whirling psychrometer. A portable small instrument which can measure the temperatures at the measuring stations. The second type being a stationary psychrometer, was hanged on the wall of the airway and the temperatures of mine air were measured continuously. The last and the most expensive type were specially designed fan fitted psychrometer. There was risk of dry bulb measuring thermometers to be wetted during measurements. But, in the last one, the thermometers were protected by a metal sheet. Airflow was blown by the fan to the thermometers. Therefore the first two were used at dry places and the last one was used at places where water income exists.

4.5. Dust Measurements

The presence of dust in mine air is caused by almost every mining operation. The main object of dust sampling is to obtain the amount of dust in the air in order to locate the sources of undesirably high dust concentrations, so that control or remedial measures can be introduced.

The measurements of dust concentrations were made at fixed positions assumed to be typical of all working places. Figure 4.9 shows sample locations of the dust measurements in the mine. Dust sampling were conducted at 18 locations in the mine.

In this study, two dust sampling equipments were used to determine dust concentrations. The first one was personal dust sampler, and second one was gravimetric dust sampler.

4.5.1. Personal Dust Measurements

The measuring instrument used was the Casella AFC 123 IS type gravimetric dust sampler. It measures the dust exposure of individual men in a particular operation or location. The sampled air is drawn through a membrane filter by a small electrically driven pump.

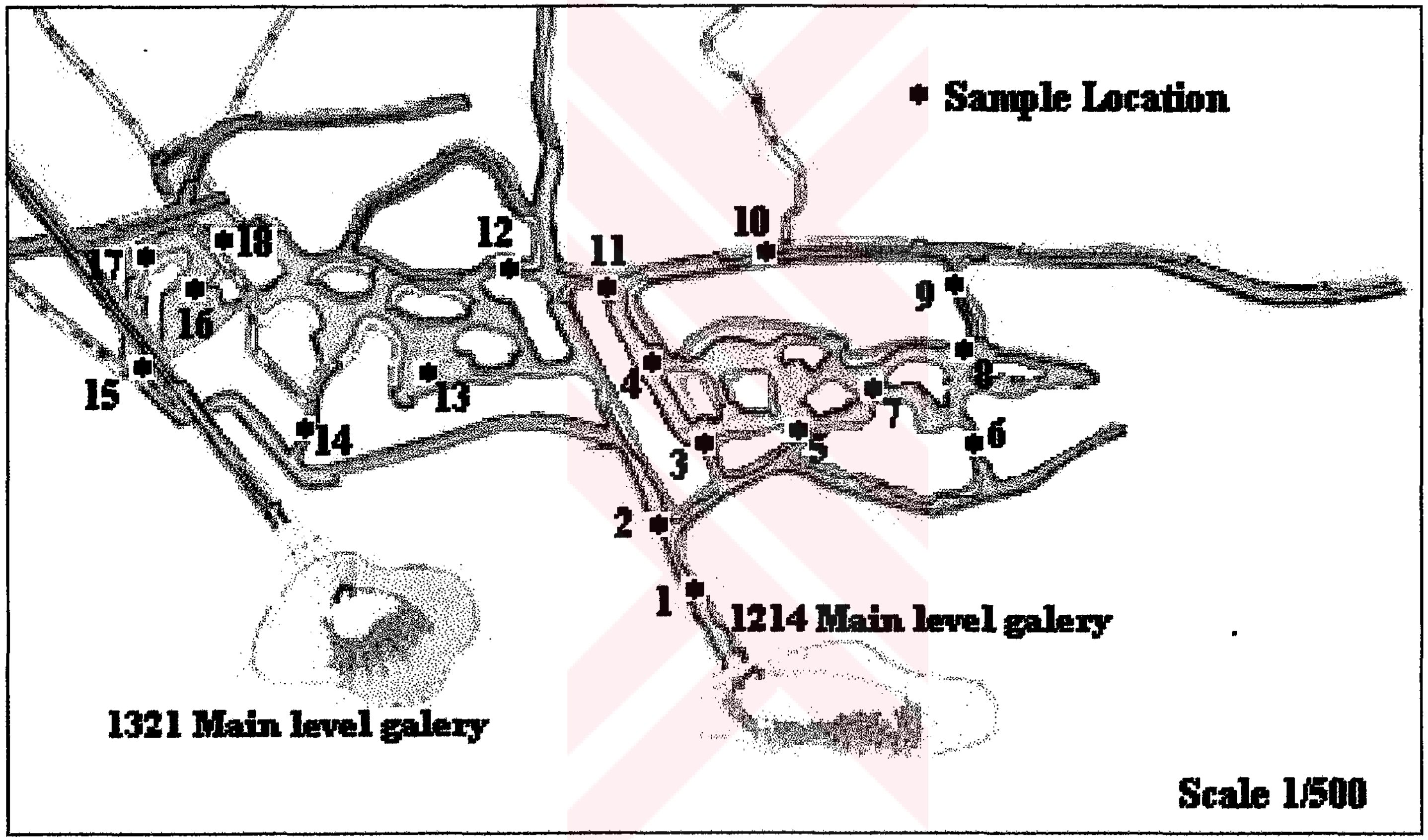


Figure 4.9 Dust sampling Locations in the Kef mine

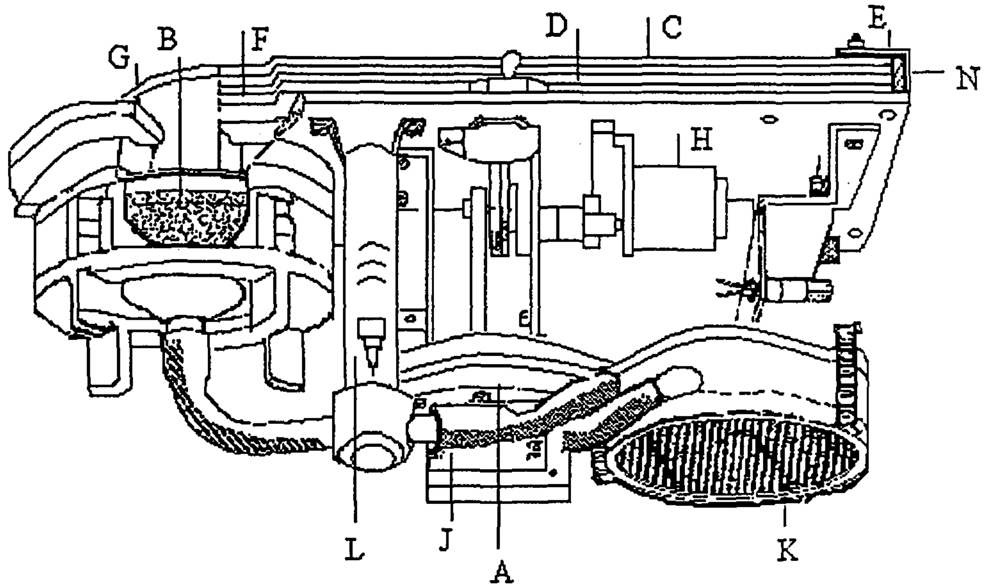
In the sampling study Whatman's GF/A glass microfibre filters with 2.5 cm in diameter were used. The filters were weighted in the laboratory with a Sortorious 0.1-mg sense type balance before and after sampling. Therefore the difference between weights was the dust quantity in mg. For each sampling procedure the filters remained 45 minutes in a 105 ^{0}C heated furnace before and after sampling.

Flow rate of the personal dust sampler was already adjusted according to individual men operation. The rate was adjusted as 2.2 l/min for drill holes equipment user, 2.1 l/min. for workers and mine machine users and 1.9 l/min for technicians and engineers.

4.5.2. Gravimetric Dust Measurements

Gravimetric dust sampler Casella type 113A instrument was used for dust measurements for determining dust concentration in the mine environment. Figure 4.10 shows The gravimetric dust sampler. The gravimetric sampler was placed at a height of 1.5 m. above the floor about 5 m. inby. The instrument sampled dust laden air at a flow rate of 2.5 l/min. and continuously for up to 4 hours.

The filters used were Whatman's GF/A glass microfibre filters with 5.5 cm in diameter. The dust sampling process was the same as in personal dust sampling procedures. Dust sampling surveys were carried out during operation and after blasting. Usually after blasting, the mining operation was stopped to clear the dust generated.



A: Pump
B: Filter
C: Size selector
D: Elutriator plate
E: Intake Plate
F: Rear plate
G: Transfer hood
H: Motor
J: Leaf valves
K: Pulsation damper
L: Flowmeter
N: Substantional steel plate

Figure 4.10 Gravimetric dust sampler

CHAPTER V

COMPUTER APPLICATIONS IN THE MINE VENTILATION AND ENVIRONMENT

5.1. Introduction

Over the past forty years, the role of the computers has expanded in an unprecedented manner. Present day utilization of computers in the mining industry varies across a wide spectrum of activities: from operations research techniques to mine management through payroll processing to mine design, monitoring and control (Bandopadhyay, 1992). The use of computers to solve mine ventilation networks began at Pennsylvania State University by Hartman and Trafton (1963).

After successive expansion of program capabilities by the Trafton and Hartman (1964), Wang and Hartman (1967), Wang and Saperstein (1970), a widely used ventilation simulator was presented by Luxbacher, Ramani, and Stefanko (1977). DMVENT Fortran coded ventilation simulation software was developed by Hardcastle (1980). ENVIRON 2.5 was developed by the Council for Scientific and Industrial Research in South Africa has been a useful mine ventilation simulation tool (Grayson, 1982). The Subway Environment Simulation (SES) (1982) and Computational Fluid Dynamics (CFD) which is a numerical technique for modeling complex fluid flow (1985) were developed by Canada Center for Mineral and Energy Technology. They have gained increased acceptance as a tool for analyzing ventilation problems. 3dRAD and MINE developed by Canadian Institute for Radion Safety, allow professionals and mine ventilation engineers to quickly simulate mine air concentrations in complex mine networks (Ray and Zigh, 1999). Then, numerous ventilation simulation programs were developed. The important ones are; SENES by Queen's University (1992), VENT-1 to VENT-4 by Dimitrov and Stefanov (1992-1996), 3D-CANVENT by Hardcastle (1995), VENT by University of Nottingham (1996), ASD Master by Electric Power Research Institute of U.S. (1996), VENTRAD by CANMET (1998) and Vnet PC 2000 by Mine Ventilation Service Inc. in US (2000).

VnetPC 2000 is the latest upgrade of the popular ventilation simulation program, combining the power of full 32-bit screen graphics. The applicability of it to subsurface ventilation system design ranges from the initial concept through to the system operations phase of a project. Given information that describes the geometry of a ventilation network, airway resistance or dimension, and the locations and characteristic curves of fans, it produces listings and visual graphics of many parameters. The output includes predicted airflow, frictional pressure drops, air power losses in airways, contaminant flows and concentrations and fan operating points.

5.2. Overview of VnetPC2000

The VnetPC program is designed to assist the mine ventilation practitioner in the planning and monitoring of underground ventilation layouts. It can simulate existing ventilation networks in such a way that the values of fan operating points, airflow quantities and frictional pressure drops are approximately the same as those of actual system. This is accomplished using data from ventilation surveys together with information determined from known airway dimensions and characteristics.

The program has been developed with assumption of incompressible flow and is based on Kirchhoff's Laws. The code utilizes an accelerated form of the Hardy Cross-iterative technique to converge to a solution. Main program features are:

i. Full-color, interactive 3D network schematic

ii. Enhanced, expandable coordinate system

iii. Data input and output via the schematic or tabular views

iv. Color coding of branches for airway type

v. Import DXF files from CAD and mine planning programs

vi. Ability to enter series and parallel arrangements for fans

vii. Imperial and SI units with full data conversion

viii. Parallel airway tool for rapid adjustment of branch resistance

ix. Automatic allocation of surface branches to close meshes around surface nodes

x. Notepad to enter detailed description of simulation

xi. Full annotation capabilities in all views

xii. Automatic calculation of branch length from coordinate values

xiii. Regulator orifice sizing tool

xiv. Pure 32-bit application with rapid execution times

xv. 5,000 branches of default network size limit with 600 fans

xvi. Steady state contaminant distribution analyses

xvii. Direct graphic printing and multi-colored plotting

xviii. Export DXF files to CAD and mine planning programs-multiple layer

xix. Computer running Windows 95, 98 or NT

5.3. Data Preparation and Input

The program is structured such that the user moves between views or windows, where input and output data are located. A single file is used for the network input, schematic coordinates and contaminant data. A separate archive file is used to store multiple fan curves. The program comprises 9 screens for the input and display of program data. These are; model information, branch input, fixed quantity, branch results, fan input and fan results, schematic, junction data and contaminant data.

This section details the content and form of the input data required for the program. The data requirements are presented in six categories, such as ventilation network, descriptive data, branch data, fan data, fixed quantity and contaminant distribution analysis data.

5.3.1. Ventilation Network

Ventilation network represents the ventilation line diagram and consists of a set of junctions and interconnecting lines, which denote airflow routes. The network form a closed circuit of interconnected branches, but those branches connected to surface were specified as being either surface intake or surface exhaust branches.

The ventilation network of the Kef district is shown in Fig. 5.1. Junction numbers were assigned to each junction and surface connection from 1 to 47. Branch numbers were also assigned to each branch from 1 to 58.

5.3.2. Descriptive Data

Descriptive data consists of both required and optional information for documentation and program initiation. The descriptive information is modified in the model information view. It allows data to directly enter into cells. A file name the Kef mine was assigned for saving data.

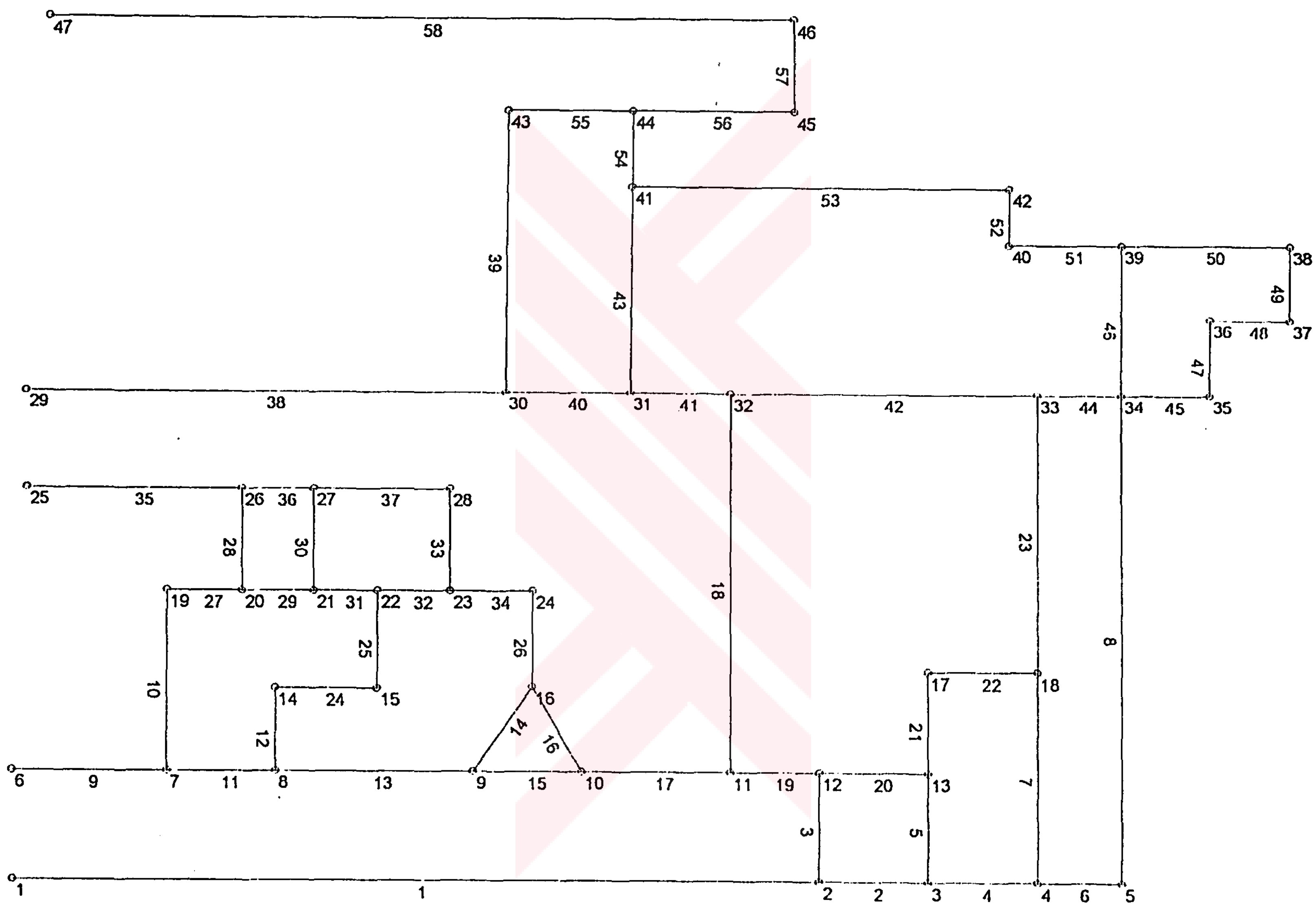


Figure 5.1 Ventilation line diagram of the mine

The program supports both Imperial and SI units. SI units were chosen throughout this study. Electrical power cost, 0.4 $/kWh, was entered to determine the operating cost for the system fans. Average underground air densities were entered for each pressure differentiations.

5.3.3. Branch Data

Branch data consists of descriptive information for documentation and initiation of the program. The branch data requirements are presented in three categories, such as branch data formats; branch input view and schematic view.

5.3.3.1. Branch Data Formats

The program recognizes four branch data formats. Branch data types are given in Table 5.1.

Table.5.1. Branch Data Types

Data Type	**Entry form**	**Comments**
1	R (airway resistance)	Fixed resistance
2	P, Q (frictional pressure drop and quantity	Pressure drop, volume data
3	K, L, L_{eq}, A, C (friction factor, length, equivalent length, area and perimeter)	Required input for Atkinson's Equation
4	R/Length, L, L_{eq} (resistance per unit length, airway length and equivalent length)	Allows direct calculation of resistance from previously measured resistance

The available branch types were available in the branch-input view from a drop down list under appropriate column.Data type 1 requires that a resistance value to be used as input for the branch. Data type 2 requires pressure and quantity as input; these values were obtained from pressure-quantity survey. Data type 3 requires the physical characteristics of the airway as input. This data type computes branch resistance empirically. Data type 4 requires resistance per unit length, branch length and equivalent length. This allows direct calculation of resistance from previously measured resistance (Duckworth et al., 1995).

5.3.3.2. Branch Input View

The branch characteristic data were entered and modified in a manner typical of Windows-based spreadsheets. If a new model is being developed or branches are being added, it can be easily selected from Add Branch command under the Edit Menu. Branch Input View of the Kef District is given in Table 5.2. This section allows entering a description for each branch, allocating symbols to the branch, entering a branch code and specifies whether branch is connected to the surface.

5.3.3.3. Schematic View

The ventilation network of the Kef district was developed entirely within the schematic view. The networks were established using functions obtained from the Tools Menu or Tools Palette. The schematic views allow developing perspectives using a level scheme. A level in the program was defined as a group of junctions falling within a range of z coordinates. It incorporates five choices in viewing perspective for the network. These are; plan view, cross section view, long section view, 3d view and single level view.

Table 5.2 Branch input view

Branch No.	From	To	Surf. State	Type	Branch Resistance (Ns²/m^8)	F Q i	Length (m)	Airflow (m³/s)	Friction Factor (kg/m³)	Resistance per Length (R/1000m)
1	1	2	Surf. Intake	R	1.84643					
2	2	3	Neither	R	0.28993					
3	2	12	Neither	R	0.28954					
4	3	4	Neither	R	0.14835					
5	3	13	Neither	R	0.17854					
6	4	5	Neither	R	0.01588					
7	4	18	Neither	R	0.06860					
8	5	34	Neither	R	0.12563					
9	6	7	Surf. Intake	R	1.96412					
10	7	19	Neither	R	3.97213					
11	7	8	Neither	R	0.81265					
12	8	14	Neither	R	1.19872					
13	8	9	Neither	R	0.05422					
14	16	9	Neither	R	0.09624					
15	9	10	Neither	R	0.06232					
16	16	10	Neither	R	1.16535					
17	10	11	Neither	R	0.19548					
18	11	32	Neither	R	0.01254					
19	12	11	Neither	R	0.28524					
20	12	13	Neither	R	0.02599					
21	13	17	Neither	R	0.07459					
22	17	18	Neither	R	0.08655					
23	18	33	Neither	R	0.23065					
24	14	15	Neither	R	0.12645					
25	15	22	Neither	R	0.13256					
26	24	16	Neither	R	0.17856					
27	19	20	Neither	R	1.93659					
28	20	26	Neither	R	0.36986					
29	20	21	Neither	R	1.81972					
30	21	27	Neither	R	0.15965					
31	21	22	Neither	R	0.16399					
32	22	23	Neither	R	0.09653					
33	23	28	Neither	R	0.16985					
34	23	24	Neither	R	0.06533					
35	26	25	Surf. Exhau	R	0.02548					
36	27	26	Neither	R	1.05979					
37	28	27	Neither	R	1.04739					
38	30	29	Surf. Exhau	R	5.93750					
39	30	43	Neither	R	0.93096					
40	31	30	Neither	R	0.82971					
41	32	31	Neither	R	0.06562					
42	33	32	Neither	R	0.03789					
43	31	41	Neither	R	0.98658					
44	33	34	Neither	R	0.06986					
45	34	35	Neither	R	0.06325					
46	34	39	Neither	R	0.82651					
47	35	36	Neither	R	0.10325					
48	36	37	Neither	R	0.07125					
49	37	38	Neither	R	0.12036					
50	38	39	Neither	R	0.05653					
51	39	40	Neither	R	0.06258					
52	40	42	Neither	R	0.01256					
53	42	41	Neither	R	0.12458					
54	41	44	Neither	R	0.83699					
55	43	44	Neither	R	0.74258					
56	44	45	Neither	R	0.24156					
57	45	46	Neither	R	0.53246					
58	46	47	Surf. Exhau	R	0.11265					

5.3.4. Fan Data

The program allows adding fans in the branch input, fan input or schematic views. In the branch input view a fan is added by referencing the edit menu. When a fan is added, an F which stands for fan, appears in the F/Q/I column. It is important that the junction numbers should be entered in the expected direction of airflow for fan branches. In the schematic view a fan is added using fan tool command, and dropping the fan on the required branch.

The branch junction numbers dictate the fan location. The order in which the junction numbers are entered defines the direction of the fan. A fan can be entered with either a fixed pressure or with a characteristic curve. In this study initially two fans were entered with fixed pressures in order to model the natural ventilation pressures of the mine. Fan curves may enter by selecting edit curve from the fan data sheet or edit fan curve under the edit menu in the fan input view.

5.3.5. Fixed Quantity

A fixed quantity can be added to a branch in the branch input, fixed quantity or schematic views. This is used to simulate control of airflow and determine the resistance, pressure drop and orifice area for regulator or the operating pressure of a booster fan.

The resistance of a branch is usually the natural resistance without a regular or booster fan. Upon execution, the program calculates the regulator resistance and orifice area or the pressure of the booster fan required to achieve the specified airflow. Within the fixed quantity dialog box there is the option of specifying the branch as an Inject or Reject branch. This is used to add or remove air from selected junctions to account for compressibility effects, ducts, compressed air lines or areas of the facility not otherwise represented in the network.

5.3.6. Contaminant Distribution Analysis Data

The contaminant distribution utility incorporated in the program utilizes results of the network exercises to evaluate contaminant concentration and gas flow distribution. During the experimental studies, dust and gas measurements were conducted. The location and magnitude of contaminant sources were specified. The program uses airflows from the last execution of the ventilation simulation to calculate contaminant flow assuming steady-state conditions.

Contaminant data can be four decimal places and are entered in the schematic view using the contaminant tool and pointing to the branch where the contaminant should be added. The contaminant distribution analysis routine requires that branches representing intakes carrying fresh air directly from the surface be identified.

5.4. Operating the Program

This section details the procedure of operating the program. The procedure requirements are presented in three categories, such as management of network files, execution of ventilation simulation and viewing the results of the simulation.

5.4.1. Management Network Files

The program utilizes conventional Windows protocol for managing files. The files are searched for under the designated VNW file extension, and fan files under the FAN extension. Files may be accessed from the host computer or via a network system. It allows the import of files from the previous version of VnetPC for windows. It is important that the file has coordinates specified for all the junctions in the network. If the coordinates had not been specified for all the nodes, then errors would appear when the file was opened.

5.4.2. Execution of Ventilation Simulation

Executing the program was accomplished by selecting the tool menu on the menu bar and then choosing execute simulation. This was done when the branch, fan and descriptive data for the network had been fully entered. When the program had finished execution, each view and any previously accessed windows were been updated with current information. The relative pressure analyses were conducted during every execution of the code. Outputs of the relative pressure and air quantity distribution can be viewed on the schematic views.

5.4.3. Viewing the Results of the Simulation

Once the program had been executed, the result of the simulation using the branch results was shown in Table 5.3. This table includes the branch number, junction number, airway total resistance, airflow, pressure drop, air power, branch description and a symbol indicating whether the branch contains a fan, regulator or booster fan (FRB). The output data are fan results, fixed quantity, schematic and contaminant data. Any computational errors are automatically listed in the error dialog box immediately after execution (VnetPC, 2000).

Table 5.3 Branch result data

Branch No.	From	To	F R B	Total Resistance (Ns²/m^8)	Airflow (m^3/s)	Pressure Drop (Pa)	Air Power Loss	Operating Cost ($/yr)
1	1	2		1.84643	22.99	976.1	22.44	12097
2	2	3		0.28993	13.02	49.2	0.64	345
3	2	12		0.28954	9.97	28.8	0.29	155
4	3	4		0.14835	19.10	54.1	1.03	557
5	3	13		0.17854	-6.08	-6.6	0.04	22
6	4	5		0.01588	33.85	18.2	0.62	332
7	4	18		0.06860	-14.76	-14.9	0.22	119
8	5	34		0.12563	33.85	144.0	4.87	2628
9	6	7		1.96412	13.10	336.9	4.41	2379
10	7	19		3.97213	3.45	47.4	0.16	88
11	7	8		0.81265	9.64	75.6	0.73	393
12	8	14		1.19872	0.43	0.2	0.00	0
13	8	9		0.05422	9.21	4.6	0.04	23
14	16	9		0.09624	1.32	0.2	0.00	0
15	9	10		0.06232	10.53	6.9	0.07	39
16	16	10		1.16535	2.46	7.1	0.02	9
17	10	11		0.19548	12.99	33.0	0.43	231
18	11	32		0.01254	0.04	124.4	0.00	3
19	12	11		0.28524	-12.96	-47.9	0.62	335
20	12	13		0.02599	22.93	13.7	0.31	169
21	13	17		0.07459	16.85	21.2	0.36	193
22	17	18		0.08655	16.85	24.6	0.41	223
23	18	33		0.23065	2.10	1.0	0.00	1
24	14	15		0.12645	0.43	0.0	0.00	0
25	15	22		0.13256	0.43	0.0	0.00	0
26	24	16		0.17856	3.78	2.6	0.01	5
27	19	20		1.93659	3.45	23.1	0.08	43
28	20	26		0.36986	1.86	1.3	0.00	1
29	20	21		1.81972	1.59	4.6	0.01	4
30	21	27		0.15965	-0.63	-0.1	0.00	0
31	21	22		0.16399	2.22	0.8	0.00	1
32	22	23		0.09653	2.65	0.7	0.00	1
33	23	28		0.16985	-1.13	-0.2	0.00	0
34	23	24		0.06533	3.78	0.9	0.00	2
35	26	25		0.02548	0.10	1091.4	0.11	59
36	27	26		1.05979	-1.76	-3.3	0.01	3
37	28	27		1.04739	-1.13	-1.3	0.00	1
38	30	29		5.93750	10.46	649.9	6.80	3665
39	30	43		0.93096	6.58	40.2	0.26	143
40	31	30		0.82971	17.04	240.8	4.10	2212
41	32	31		0.06562	20.63	27.9	0.58	310
42	33	32		0.03789	20.59	16.1	0.33	179
43	31	41		0.98658	3.59	12.7	0.05	25
44	33	34		0.06986	-18.50	-23.9	0.44	238
45	34	35		0.06325	8.99	5.1	0.05	25
46	34	39		0.82651	6.37	33.5	0.21	115
47	35	36		0.10325	8.99	8.3	0.07	40
48	36	37		0.07125	8.99	5.8	0.05	28
49	37	38		0.12036	8.99	9.7	0.09	47
50	38	39		0.05653	8.99	4.6	0.04	22
51	39	40		0.06258	15.36	14.8	0.23	123
52	40	42		0.01256	15.36	3.0	0.05	25
53	42	41		0.12458	15.36	29.4	0.45	243
54	41	44		0.83699	18.95	300.5	5.69	3070
55	43	44		0.74258	6.58	32.1	0.21	114
56	44	45		0.24156	25.52	157.4	4.02	2165
57	45	46		0.53246	25.52	346.8	8.85	4771
58	46	47		0.11265	25.52	73.4	1.87	1010

CHAPTER VI

EVALUATION OF RESULTS AND FAN DESIGN

6.1. Introduction

Mine ventilation involves maintaining desired air quantities, advising on the fans and airways necessary to ventilate new areas. The object of air quantity and pressure surveys are essential to determining the amount of air quantity passing through the mine and pressure loss over a known length of airway.

The main object of dust sampling is to obtain the amount of dust in the mine air in order to locate the sources of undesirably high dust concentrations, so that control or remedial measures can be introduced.

By conducting ventilation and dust surveys throughout a mine, it is possible to determine existing conditions and to obtain information which may form the bases for future planning. These surveys comprised the measurement of pressure loss, airflow quantity, airway dimensions, air density, dust concentration during operation and dust concentration after blasting (Eston et al., 1993).

In the field study, all of the above surveys were conducted for the Kef mine throughout one year. The survey results are evaluated in detail in this chapter.

6.2. Evaluation of Results

Evaluation of results mainly consists of pressure survey, quantity survey and dust concentration measurements. Natural ventilation pressure and quantity surveys were evaluated in such a way that the mine characteristic of the Kef mine was determined. This was followed by the required quantity of air in for the mine and fan selection procedures.

6.2.1. Analysis of Airflow Quantity

The unit of air quantity most commonly used in the mining industry is volume in cubic meters per second. This unit is the product of the average airflow velocity and cross sectional area of the airway at the point of measurement. This can be expressed by the following equation;

$$Q=V\times A \tag{6.1}$$

Where;

Q = Quantity of airflow in m^3/s,

V = Velocity of airflow in m/s,

A = Cross sectional area of airway in m^2.

In the field study, average velocity of airflow for all main galleries were defined throughout the year. The result is given in the Table 4.4. Cross sectional areas of the airways in the mine were also calculated and the results were given in the Table 4.5. Therefore, the annual airflow quantities of the mine for all main galleries can be calculated by the above equation. Airflow quantity for main galleries in the mine is given in Table 6.1.

Table 6.1. Airflow quantity in the main galleries

Months	Average Airflow Quantity (m^3/s) (+: in, -: out)				
	1214 Gallery	**1235 Gallery**	**1304 Gallery**	**1321 Gallery**	**1416 Gallery**
January	+8.16	+6.52	-2.38	-3.24	-9.06
February	+5.54	+4.16	-1.62	-2.00	-6.08
March	+1.46	+1.16	-0.16	-0.34	-2.12
April	-2.08	-0.34	0.00	+0.52	+1.90
May	-4.40	-0.99	+0.36	+1.18	+3.85
June	-8.92	-3.16	+1.86	+2.96	+7.26
July	-10.52	-5.04	+2.86	+4.26	+8.44
August	-11.62	-6.62	+3.82	+5.38	+9.04
September	-8.21	-2.70	+1.58	+2.58	+6.75
October	-3.02	-0.71	+0.09	+0.91	+2.73
November	+0.74	+0.54	-0.04	-0.16	-1.08
December	+6.94	+5.32	-1.96	-2.71	-7.59

Main galleries are represented by numbers that indicate the elevation of the galleries. The bottom gallery has an elevation of 1214-m. The top gallery has an elevation of 1416-m. Elevation difference between top and bottom galleries is more than two hundred meters. This difference leads to natural ventilation due to unequal density in the mine air and surface air. Consequently, in summer and winter seasons, the air quantity values are high but in spring and fall seasons, the values are relatively low.

The positive values in the Table 6.1 show right direction ventilation and the negative values show reverse direction of ventilation. The airflow quantity in the main galleries is graphically shown in Figure 6.1. The horizontal and vertical axes denote months and air quantity, respectively.

The curves in the Figure 6.1 represent main gallery airflow quantity and direction. Positive values of the vertical coordinate show that, airflow enters to the main gallery and negative values show that, airflow return from the main gallery. If a curve intersects the horizontal axis, it means that airflow direction of that gallery change in this month. Airflow quantity of a gallery decreases if its curve approaches to the horizontal axes. In case of approaching all curves, the total quantity of the mine decreases.

The total air quantity of the mine decreases in spring and fall seasons. Therefore, mining activity is interrupted for long time in these seasons due to insufficient ventilation. During this study, sometimes the interruption was observed to take four hours. In winter and summer seasons. the ventilation quantity are better but inadequate for continuous mining operation. The curves in the Figure 6.1 are departing from the horizontal axis in winter and summer seasons. The interruption of the mining activity in these seasons is a little bit shorter but it still takes two to three hours between two successive operation shifts.

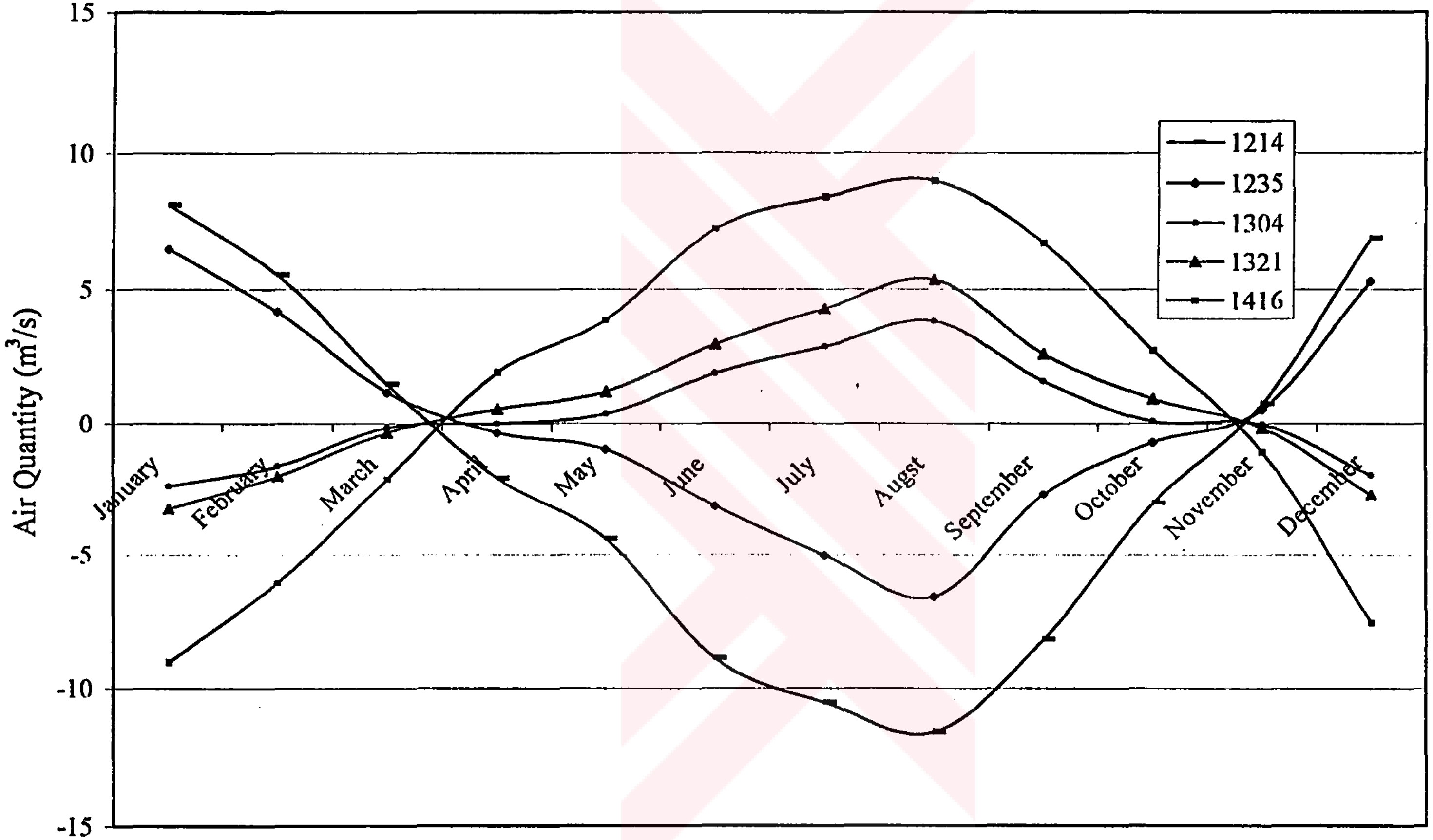


Figure 6.1 Air quantity distribution in the main galeries of the Kef mine

Airflow distribution in all branches of the mine was determined by using the VnetPC2000 ventilation software program. The air quantity distribution measured experimentally is very close to the air quantity distribution obtained from the software. This difference was in a range of 0.1-2.5%.

Air quantity distribution in all airways in the mine can be computed if only natural ventilation pressure is known. The air distribution for January is given in the Table 6.2. Branch numbers 1 and 9 are for 1214 and 1235 main galleries, respectively. The airflow quantity in the computer output given in Table 6.2 for these galleries are 7.98 m^3/s and 3.14 m^3/s, respectively which are very close to experimentally measured 8.16 m^3/s and 6.52 m^3/s quantities given in the Table 6.1. Such difference could be tolerable for further developments in the mine ventilation system.

6.2.2. Analysis of Pressure Surveys

Analysis of the existing mine ventilation system, including the evaluation of modifications to the system, requires accurate input data that can be developed only by a detailed pressure quantity survey in the mine. The purpose of an accurate underground pressure survey is to obtain a pressure gradient along the circuit or circuits under investigation and determines the amount of natural ventilation pressure.

By conducting pressure surveys throughout the mine it is possible to determine the existing conditions and to obtain information required for the future ventilation design. The results can be used to identify airways in which the pressure drop is abnormally high, and to determine resistance figures of all branches in the mine ventilation network.

Table 6.2. Airflow distribution in the mine (January)

Branch No.	Airflow Quantity (m^3/s)	Branch No.	Airflow Quantity (m^3/s)	Branch No.	Airflow Quantity (m^3/s)
1	7,98	21	3,70	41	7,11
2	4,06	22	3,70	42	3,24
3	3,93	23	4,40	43	1,65
4	3,84	24	0,74	44	1,16
5	0,22	25	0,74	45	2,52
6	3,14	26	0,24	46	1,78
7	0,70	27	1,68	47	2,52
8	3,14	28	1,69	48	2,52
9	6,09	29	0,01	49	2,52
10	1,68	30	0,64	50	2,52
11	4,41	31	0,65	51	4,30
12	0,74	32	0,09	52	4,30
13	3,67	33	0,33	53	4,30
14	0,88	34	0,24	54	5,95
15	2,80	35	2,66	55	2,14
16	0,63	36	0,97	56	8,09
17	3,43	37	0,33	57	8,09
18	3,87	38	3,32	58	8,09
19	0,44	39	2,14		
20	3,48	40	5,46		

The most important parameters, which are measured in pressure survey, are pressure drop measurements and dry-wet bulb temperatures of the airflow in the mine. Dry-wet bulb temperature measurements were carried out throughout the year. These measurements were conducted at all junctions in the mine. As an example, dry-wet temperature measurements for January are given in Table 6.3.

Pressure difference (pressure loss) between two points in an airway is determined by height of the column in the manometer. This height is also called as pressure head and usually it is converted to pressure unit Pascal (Pa) in calculations.

The relationship between the pressure unit and head can be expressed by the following equation:

$$P = Hgw \tag{6.2}$$

Where;

P = Pressure in Pa,

g = Gravitational acceleration in m/s^2,

w = Density of manometer liquid in kg/m^3,

H = Head in mm.

Table 6.3. Temperature measurements at junctions in the mine (January)

Junct. No.	Dry-Bulb ^0C	Wet-Bulb ^0C	Air Density kg/m^3	Junc. No.	Dry-Bulb ^0C	Wet-Bulb ^0C	Air Density kg/m^3
1	8,40	5,50	1,035	25	8,40	5,50	1,022
2	13,50	11,10	1,034	26	13,60	11,90	1,021
3	13,60	11,10	1,034	27	13,60	11,90	1,021
4	13,70	11,20	1,033	28	13,70	12,00	1,020
5	13,70	11,20	1,033	29	8,40	5,50	1,021
6	8,40	5,50	1,033	30	14,10	13,00	1,016
7	13,40	10,40	1,031	31	14,10	13,00	1,016
8	13,50	10,50	1,030	32	14,20	13,10	1,016
9	13,50	10,50	1,030	33	14,20	13,10	1,015
10	13,50	10,50	1,030	34	14,20	13,10	1,015
11	13,60	10,50	1,029	35	14,30	13,20	1,015
12	13,70	10,50	1,028	36	14,30	13,40	1,014
13	13,70	10,50	1,027	37	14,30	13,40	1,014
14	13,70	10,60	1,022	38	14,40	13,70	1,010
15	13,70	10,60	1,022	39	14,40	13,70	1,010
16	13,60	10,50	1,028	40	14,40	13,70	1,010
17	13,60	10,50	1,028	41	14,40	13,70	1,007
18	13,70	10,60	1,026	42	14,40	13,80	1,007
19	13,80	10,60	1,026	43	14,50	14,00	1,004
20	13,80	10,70	1,026	44	14,60	14,00	1,003
21	13,80	10,70	1,026	45	14,70	14,10	1,002
22	13,80	10,70	1,026	46	14,70	14,10	1,000
23	13,80	10,80	1,026	47	8,40	5,50	1,000
24	13,90	10,80	1,025				

Pressure loss measurements were carried out at the all branches of the mine throughout the year. The results were computed monthly. As an example, pressure drop in the branches for January is illustrated in Table 6.4.

Table 6.4. Pressure loss distribution in the mine (January)

Branch No.	**Pressure (Pa)**	**Branch No.**	**Pressure (Pa)**	**Branch No.**	**Pressure (Pa)**
1	117.6	21	1.0	41	3.3
2	4,8	22	1,2	42	0,4
3	4,5	23	1,3	43	2,7
4	2,2	24	0,1	44	0,1
5	0,0	25	0,1	45	0,4
6	0,2	26	0,0	46	2,6
7	0,0	27	5,5	47	0,7
8	1,2	28	1,1	48	0,5
9	72,8	29	0,0	49	0,8
10	11,2	30	0,1	50	0,4
11	15,8	31	0,1	51	1,2
12	0,6	32	0,0	52	0,2
13	0,7	33	0,0	53	2,3
14	0,1	34	0,0	54	29,7
15	0,5	35	99,4	55	3,4
16	0,5	36	1,0	56	15,8
17	2,3	37	0,1	57	34,8
18	4,1	38	65,6	58	7,4
19	0,1	39	4,3		
20	0,3	40	24,7		

6.2.3. Natural Ventilation Pressure of the Kef Mine

There are many methods to calculate natural ventilation pressure. These are the calculation of natural ventilation pressure from densities, pressure quantity, pressure across a stopping in fan drift and fan running at two different speeds. Among them in a naturally ventilated mine, the pressure can be calculated only from densities.

The amount of the natural ventilation pressure is defined from density differences of mine air column and surface air column. The following equation is used to calculate natural ventilation pressure:

$$NVP = Hg(w_d - w_u)$$

$$NVP = \frac{HgB}{287{,}1}\left(\frac{1}{T_u} - \frac{1}{T_d}\right) \tag{6.3}$$

Where;

NVP = Natural ventilation Pressure in Pa,

H = Height of the column in m,

B = Barometer reading in kPa,

T_u = Average temperatures in the upcast shaft.

T_d = Average temperatures in the downcast shaft.

In this study natural ventilation pressure of the mine was calculated throughout one year. The results were reprocessed for obtaining monthly natural ventilation pressures. Table 6.5 illustrates the natural ventilation pressure of the mine.

Mine air density is greatly affected by temperature, hence it is very dynamic. Temperature in the mine does not vary too much when compared with outside temperature. Temperature difference between inside and outside of mine causes change in air density.

Table 6.5. Natural ventilation pressure of the mine.

Months	**NVP (Pa)**	**Air Quantity (m^3/s)**
January	208.39	14.68
February	90,99	9,70
March	6,64	2,62
April	5,66	2,42
May	28,09	5,39
June	141,11	12,08
July	234,12	15,56
August	321,72	18,24
September	115,11	10,91
October	13,45	3,73
November	1,58	1,28
December	145,35	12,26

The amount of this difference determines the natural ventilation pressure quantity. Natural ventilation pressure of the mine is graphically shown in Figure 6.2. NVP increase in winter and summer seasons but it decreases in spring and fall seasons. This situation can be observed from the Figure 6.2.

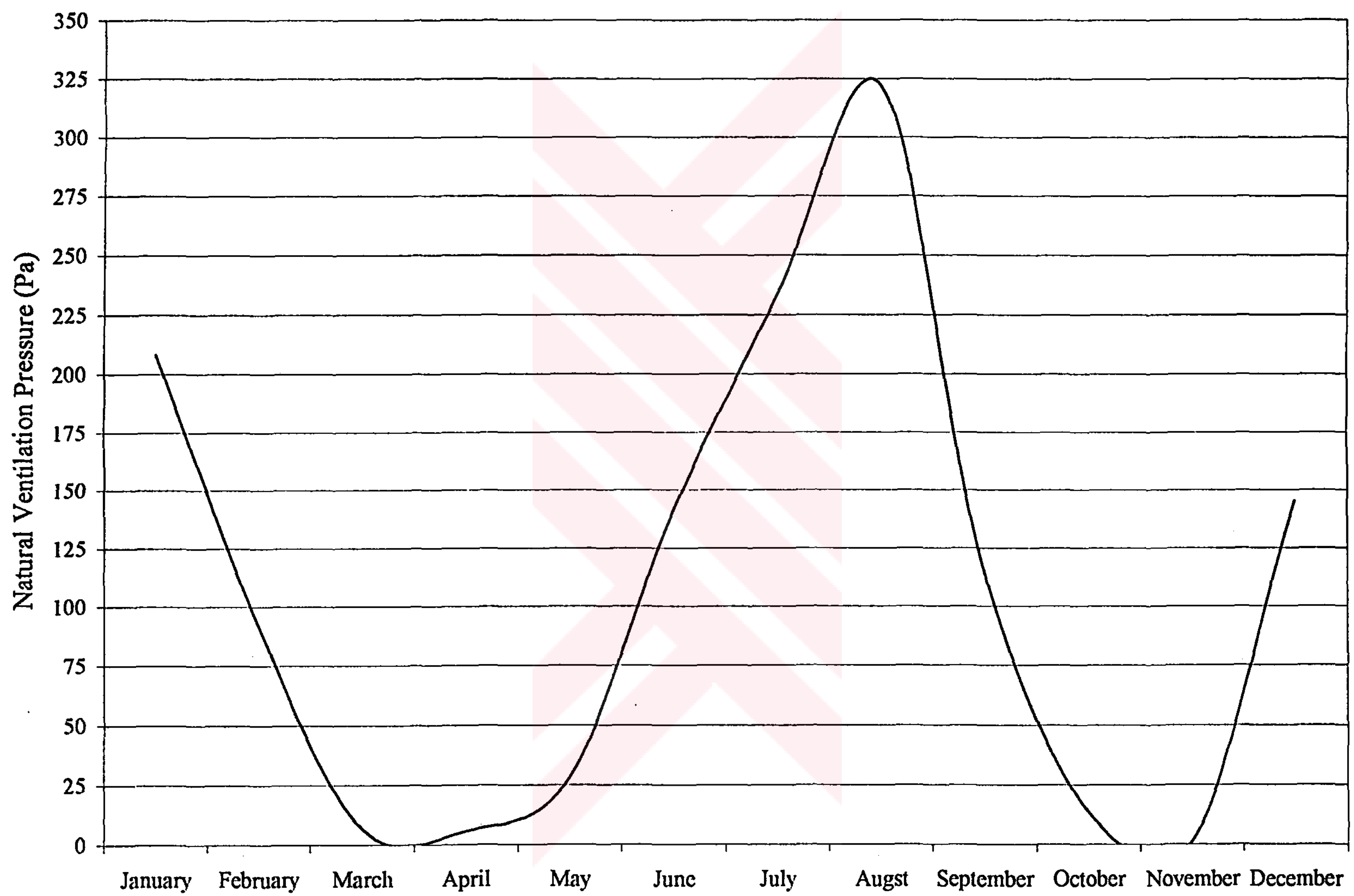


Figure 6.2 Natural ventilation pressure throghcut one year in the Kef mine

6.2.4. The Characteristic Curve of the Natural Ventilation

The characteristic of the natural ventilation is a curve that relates natural ventilation pressure to the airflow passing through the mine. The natural ventilation pressure and airflow quantity of the mine was determined in the experimental study. These quantities were illustrated in the Table 6.5.

The characteristic curve of the natural ventilation is also called mine characteristic curve. Mine characteristic curve is very important for the mine ventilation design and fan selection procedure. Mine Characteristic of the Kef mine is shown in Figure 6.3.

6.3. Analysis of Dust Concentration

In this study, dust concentration measurements were carried out for two cases. These cases were dust concentration during mining operation and dust concentration after blasting. The dust samples collected from the mine were weighed and procedure for determining the dust concentration was followed. The dust concentration of the mine was determined by the following equation in unit of mg/m^3 during production and after blasting.

$$\text{Dust Concentration} = \frac{C_f - C_i}{F \times T} \tag{6.4}$$

Where;

C_i = Corrected initial filter mass in mg,

C_f = Corrected final filter mass in mg,

F = Airflow rate in m^3/min,

T = Sampling time in min.

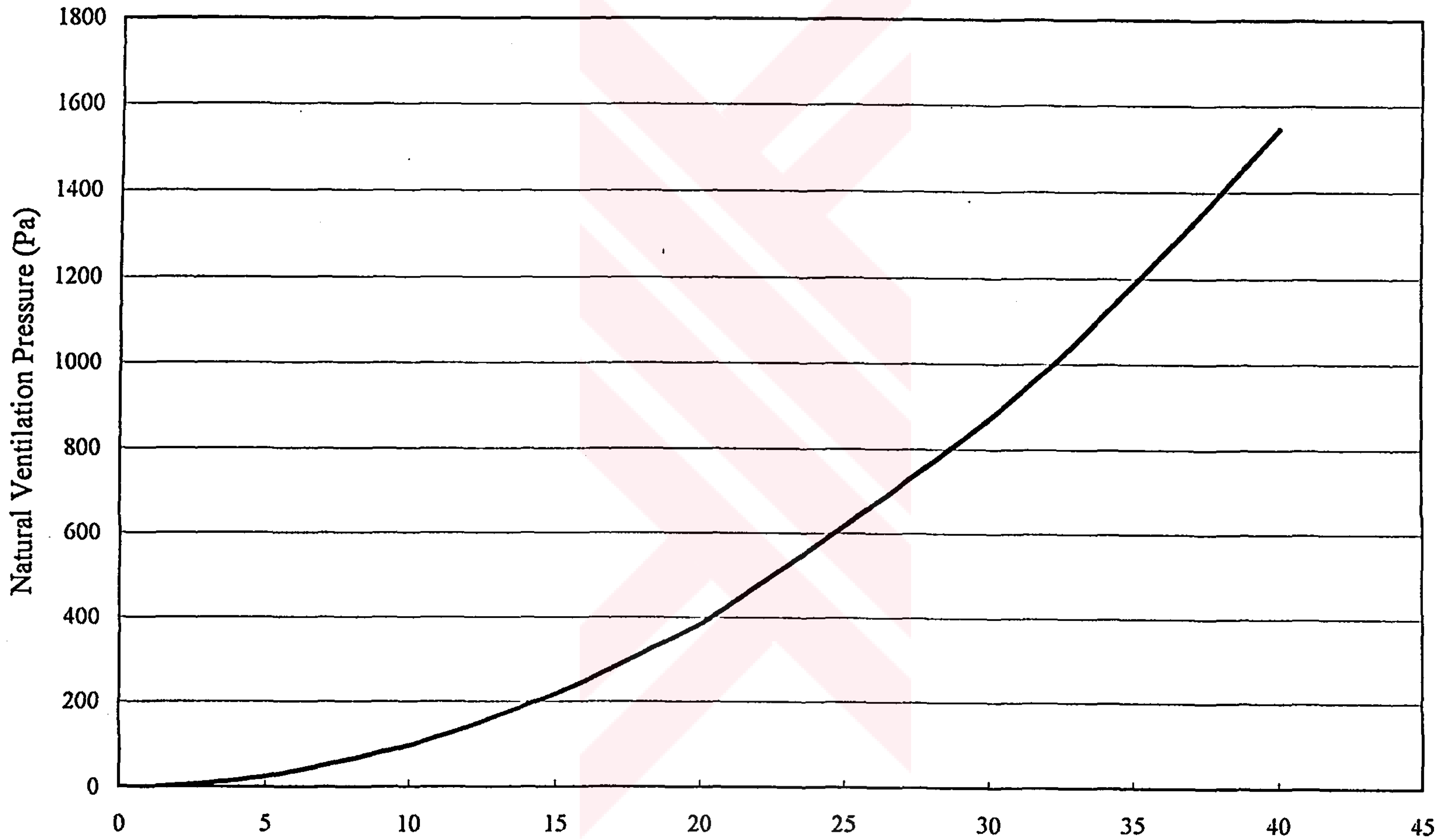


Figure 6.3 The characteristic curve of the Kef chromium mine

The dust measurements during operation in the mine were carried out for one year in the study. The dust concentration were determined monthly at working places of the 1214 level. The results have been illustrated in Table 6.6.

Table 6.6. Dust concentration during the mining operation.

Months	**Dust Concentration (mg/m^3)**
January	38.27
February	46.42
March	48.85
April	49.91
May	45.28
June	38.34
July	35.93
August	32.82
September	39.34
October	50.51
November	50.66
December	43.61

The dust concentration is not constant and changes during in the year. It increases in the spring and fall seasons, but decreases in the winter and summer seasons as shown in Figure 6.4. Maximum allowable dust concentration in such mines is 5 mg/m^3 according to Turkish official law of health and safety. The dust concentration exceeds this limit in every month.

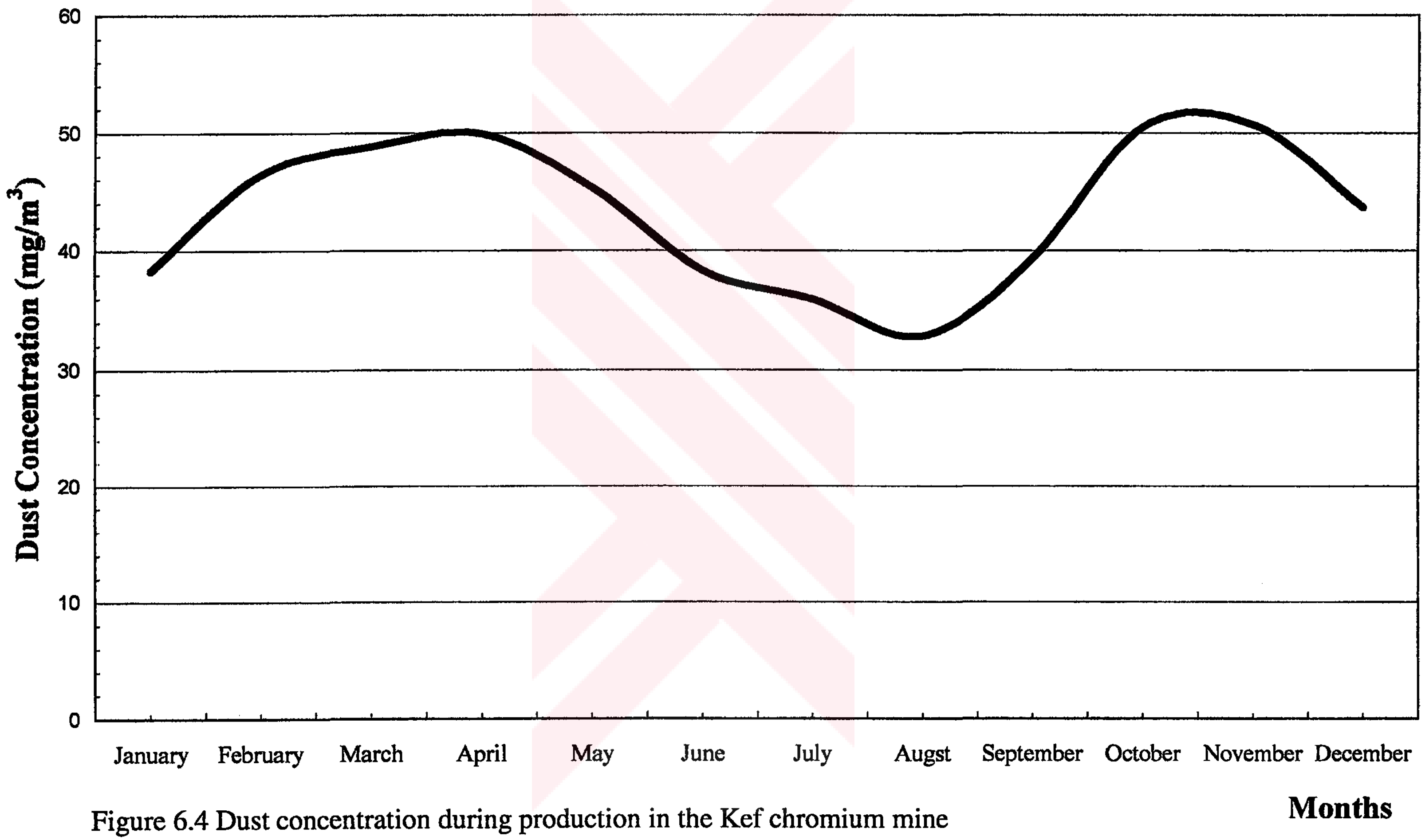


Figure 6.4 Dust concentration during production in the Kef chromium mine

The dust concentration measurements after blasting were also carried out during one year in the study. These measurements were for the determination of the dust concentration after blasting and observing the behavior of the dust concentration based on elapsed time. The results are illustrated in Table 6.7.

Table 6.7 Dust concentration after blasting.

Months	Dust Concentration (mg/m^3)							
	30 (min)	60 (min)	90 (min)	120 (min)	150 (min)	180 (min)	210 (min)	240 (min)
January	25.54	19.09	14.27	9.77	6.12	3.42	1.64	0.67
February	27.57	21.45	16.23	11.10	6.95	3.93	1.89	0.78
March	28.90	22.74	17.00	11.57	7.36	4.13	2.02	0.85
April	28.97	22.87	17.53	12.07	7.68	4.30	2.19	0.96
May	26.78	20.83	15.56	10.66	6.67	3.73	1.78	0.73
June	21.76	16.90	12.46	8.41	5.34	3.01	1.53	0.57
July	19.22	14.95	11.32	7.35	4.54	2.76	1.41	0.54
August	17.95	13.78	10.02	6.27	3.82	2.14	1.09	0.41
September	24.78	19.50	14.76	10.18	6.14	3.35	1.58	0.69
October	29.47	22.93	16.90	11.41	7.13	4.13	1.98	0.81
November	29.61	23.03	16.93	11.00	7.00	4.06	1.94	0.78
December	26.02	20.25	14.92	10.07	6.14	3.32	1.55	0.65

The dust concentration level is very high after blasting. The mining activity must be interrupted for 150 minutes to 190 minutes for providing safe dust conditions. These results graphically represented in Figure 6.5.

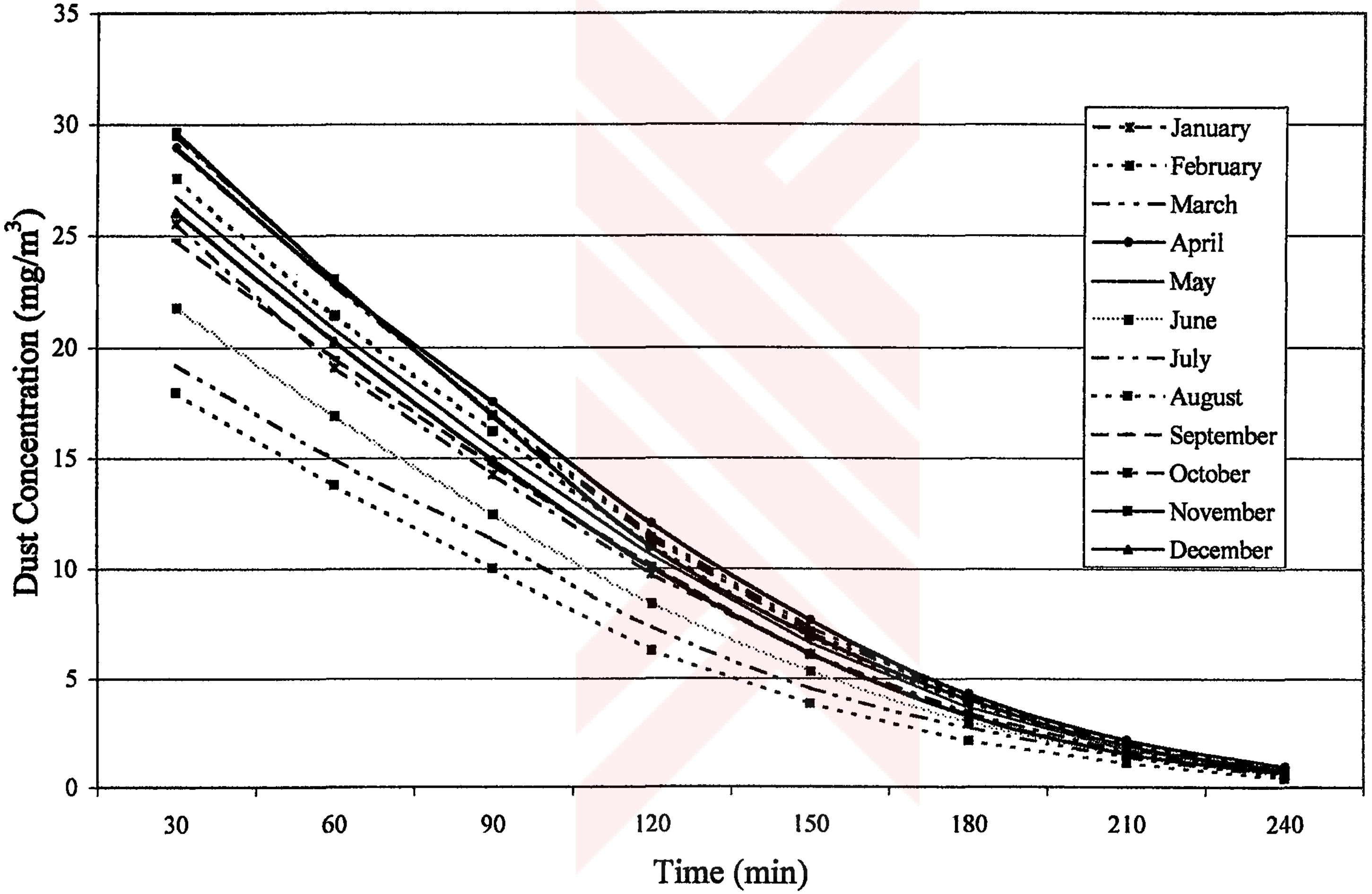


Figure 6.5 Dust concentration after blasting in the Kef chromium mine

6.3.Fan

A fan is an appliance, which produces airflow. There are two main types of fans used in the mining industry namely centrifugal and axial flow. In centrifugal fans, the air enters axially parallel with the impeller drive shaft and then moves radially and finally tangentially through the impeller casing. As the blades rotate, air enters from one side (single inlet) or from both sides (double inlet) because of the vacuum created by the rotating impeller and exists from delivery side. Axial flow fan consists of a shaft with a hub or boss to which is attached a number of blades. Blades can either be fixed or have adjustable pitch. Axial flow fan produces a flow of parallel to the shaft or axis of the fan.

Fan characteristic curves indicate how much air a fan can deliver at any particular pressure and how much power is required to drive the fan and the efficiency of the fan for each case. The curves are applicable to a particular fan for specified fan speed, impeller diameter and air density. As one of these changes, fan characteristics curve are also subject to change. The new characteristics under the new values of density, speed and diameter can be found by applying the fan laws.

In this study, mine characteristic of the Kef mine was determined. In order to select a proper and efficient fan, required quantity of air should be defined.

6.3.1. Required Air Quantity

Amount of air needed for the mine should satisfy different requirements such as breathing, dilution of harmful gases and dust produced in the mine. This amount of air is necessary for the ventilation design and further development in mine planning.

6.3.1.1. The Amount of Air Needed for a Human Being

The amount of air needed for a human being can be calculated theoretically by considering lower acceptable oxygen content in air and maximum allowable concentration of carbondioxide of the mine. Oxygen content should not be less than the recommended lower limit level. The amount of air needed is found as follows;

$$\begin{pmatrix}\text{Amount of } O_2 \\ \text{in intake air}\end{pmatrix} - \begin{pmatrix}\text{Amount of } O_2 \\ \text{used for breathing}\end{pmatrix} = \begin{pmatrix}\text{Amount of } O_2 \\ \text{in return air}\end{pmatrix} \qquad (6.5)$$

Amount of O_2 used for breathing depends on the type of activity of a person. At rest, moderate working and vigorous cases 0.28, 1.96 and 2.80 lt/min. of O_2 is needed respectively.

In breathing operation, all or some of oxygen is converted to CO_2 depending on the ratio of CO_2 expelled to O_2 consumed. This ratio is known as respiratory quotient and illustrated in Table 6.8.

Table 6.8. Inhalation rates in breathing

Activity	**Respiratory Rate (min)**	**Air Inhaled/ Respiratory (lt)**	**Air Inhaled (lt/min)**	**Oxygen Consumed (lt/min)**	**Respiratory Quotient**
At rest	15	0,55	8,7	0,28	0,75
Moderate	30	1,75	52,5	1,96	0,90
Very active	40	2,50	100,0	2,80	1,00

The respiratory quotient is a volume ratio. A person working moderately would consume 1.96 lt/min. of oxygen but liberate only $1.96 \times 0.9 = 1.764$ lt/min of CO_2. The lowest acceptable O_2 concentration is 20% and maximum allowable concentration for CO_2 is 0.5%.

In order to find the amount of air required for a very active person equation 4.5 is used. O_2 concentration in intake air is 20.5% and consumed O_2 in lt/min/man is 2.8 so;

$$(0.205)\ Q - 2.8 = (0.2)\ Q$$

$$Q = 0.0093\ m^3/s/man$$

Another consideration is the amount CO_2 produced during breathing. The amount of required air is defined from the following equation.

$$\begin{pmatrix}\text{Amount of } CO_2 \\ \text{in intake air}\end{pmatrix} + \begin{pmatrix}\text{Amount of } CO_2 \\ \text{produced from breathing}\end{pmatrix} = \begin{pmatrix}\text{Amount of } CO_2 \\ \text{in return air}\end{pmatrix} \qquad (6.6)$$

The amount of CO_2 in intake air is 0.3%. The air required for a very active person can be determined as follows;

$$(0.0003)\ Q + 2.80 * 1.00 = 0.005\ Q$$

$$Q = 0.0099\ m^3/s/man$$

The bigger quantity is chosen for determining required air quantity. This value is multiplied by the number of person in the mine. Approximately 50 people work in the mine. Therefore the required air quantity is 0.495 m^3/s. If a safety factor 2 is chosen for the mine, the final required air quantity is 0.990 m^3/s.

6.3.1.2. The Amount of Air Needed to Dilute the Harmful Gases

In the study three major gases were examined. These were methane (CH_4), carbonmonoxide (CO) and hydrogenesulfur (H_2S). The gas measurements were carried out during operation and after blasting.

Methane was measured by a portable instrument equipped with an indicator. The instrument measured methane as percent. The maximum methane concentration was found 0.25% in few measurements. The maximum allowable methane concentration is 2% according to safety rules.

A digital instrument is used for measurement of carbonmonoxide. The instrument measured carbonmonoxide in ppm. The gas did not exceed 8 ppm throughout the study. The maximum allowable carbonmonoxide concentration is 50 ppm.

Hydrogenesulfur was measured by a digital instrument. The instrument measured hydrogenesulphur in ppm. The gas did not exceed 4 ppm throughout the study. The maximum allowable hydrogenesulphur concentration is 50 ppm.

The concentrations of all gases were defined low. There is no need for taking any precaution against these gases. Therefore, any calculation to determine air quantity considering gases is unnecessary.

6.3.1.3. The Amount of Air Needed after Blasting

This is important especially in headings. The ventilation direction of the mine changes the amount of required air. In the forcing case of ventilation the following equation can be used to determine the amount of air;

$$Q = \frac{0.14}{t}\sqrt{FV^2} \tag{6.7}$$

Where;

Q = Amount of fresh air in m^3/s,

t = Ventilating time in s,

F = Amount of explosive used in kg,

V = Volume that should be ventilated in m^3.

In the exhaust type of ventilation the following empirical equation can be used.

$$Q = \frac{0.1}{t}\sqrt{FA(75 + A)} \tag{6.8}$$

Where;

Q = Amount of fresh air in m^3/s,

t = Ventilating time in s,

F = Amount of explosive used in kg,

A = cross sectional area in m^2.

Both type of the ventilation take place in the mine therefore, these two equations must be used in the calculations. Amount of explosive was approximately 150 kg per shift, and the volume to be ventilated was 2200 m^3. Here it is assumed that 600 seconds are enough for air to become clean.

By using these values in the equations 4.6 and 4.7, the required air quantity were founded 6.287 m^3/s and 1.616 m^3/s for forcing type and exhaust type respectively.

6.3.1.4. The amount of Air Needed for Mine Dust

The calculation of required air quantity for dust is important in poor ventilated production places and in headings as well. In the study the main problem was excessive dust generation during production and after blasting.

In order to determine the required amount of air, dust generation rate must be determined. The following equation is used to determine required air quantity;

$$Q = \frac{G}{TLV - B} \tag{6.9}$$

Where;

G = Dust production in g/min,

TLV = Threshold limit value of dust concentration in mg/m^3,

B = Dust Concentration with intake air in mg/m^3.

Dust production rate is 5.79 g/min for recovery of 180 tons/shift ore by drilling and blasting operation (Sengupta, 1992). In this study, the dust production rate was determined as 6.72 g/min. The ore production amount in the Kef mine is 600 ton/day. The threshold limit value for such mines in Turkey is 5 mg/m^3. Dust concentration within intake air was about 0.1 and 0.4 mg/m^3.

The required air quantity was found to be 17.71 m^3/s. If a safety factor of 2 is chosen the final required air quantity is 35.42 m^3/s.

6.3.2. Fan Selection

Fan should be selected in such a way that it operates effectively and efficiently. Fan must deliver the required quantity of air at least possible cost. The cost of the power needed to drive the fan is usually greater than the actual cost of the fan itself. Required air quantity of Kef mine is 17.71 m^3/s. Since characteristic curve of the mine is known, the selection of suitable fan can be done. If fan characteristic curve and mine are drawn on the same graph, the intersection point will indicate the operating point that will give pressure and quantity values that will provide while fan is operating (Güyagüler and Güngör, 1999).

In this study many fan characteristics study were investigated. The selected fan characteristic is shown in Figure 6.6. It has high capacity according to required air quantity of the mine. The primary reason of choosing this fan is the possibility of supplying the required quantity of air to the mine. Another consideration is further development of the mine towards the east side and any attempts for producing ore from lower levels of the bottom gallery.

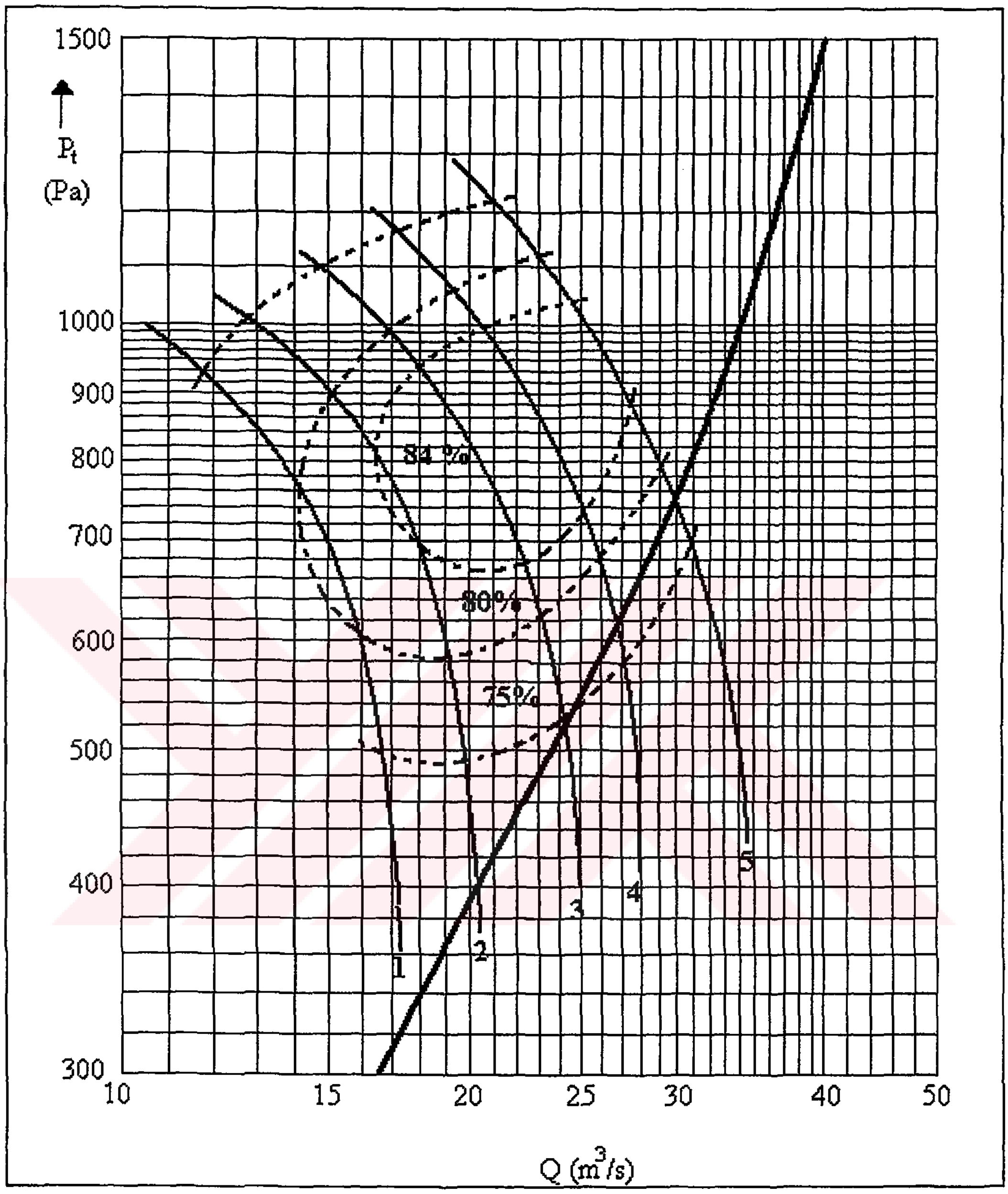


Figure 6.6 Fan and the mine characteristic

The practicing of installing fans underground tends to favor axial flow fans over centrifugal. The main reason for installing fans underground, rather than on surface was because of the inconvenience that would be created by airlock necessary in those operating shafts where the fans were installed. Another critical consideration is that underground installation of main fans in metal mines is allowed.

Axial flow fan is selected for the Kef mine. Variable blade pitch angle can be used on all levels of the mine. The majority of ventilation main fans are installed on the production level at a distance of approximately 90 to 450 meters from the fresh air intake (Mutama and Pelletier, 1999)

6.3.3. Design Procedure

Air quantities are usually determined by practical consideration and experience. Air velocities even in main airways should not exceed 8 m/s, both for comfort and to reduce head losses due to friction. Air velocities at a minimum should be kept above 0.25 m/s. as this is about lowest velocity at which air movement can be sensed. In working places, air velocities should be 1.0 to 2.0 m/s. Intake air normally is where people enter. Metal mines usually use booster fans underground to divert airflow or to give additional air moving power where needed. Auxiliary fans are relatively small fans used in directing air to individual faces or small areas.

Reuse of air is allowable in metal mine situations, especially where the used air is mixed with fresh air, and this is normally preferred for economic reasons to using air only once. Coursing air is to direct the air to a stopping raise and upcast it to the level above. This is named as ascentional ventilation and used for two reasons; mining method in which ore extraction is commenced at above and proceeds upward with successive cuts, and the second reason is chimney effect which leads to air rise because of rock heats (O'Neil and Johnson, 1982).

Generally, the design procedure for a metal mine is as follows:

i. Main ventilation inlets and outlets are selected.

ii. Airflow requirements and tentative ventilation schematic design are determined. This must be done for various stages in the life of the mine.

iii. Main and booster fan locations are tentatively specified.

iv. Airflow resistance for all branches is determined, and a computer model of the mine ventilation system is built.

v. Head loss calculations are simplified to initially determine fan characteristics.

vi. A computer simulation of ventilation system is run.

vii. Adjustments of branch resistances, fans, etc., are done where needed, and the program is rerun until computer output shows the model to be a viable.

viii. Alternative setups are considered.

ix. The planned setup is reviewed and is looked over the plan from standpoint of safety.

6.3.4. Ventilation Layout for the Kef Mine

The mine was designed as taking into account ventilation, development ventilation, haulage, equipment maneuverability, material transportation and recovery. There are five main airways connecting to surface in the mine. Two main airways are located at west side, with an elevation of 1235-m and 1304-m, respectively. The other three remain at east side and have an elevation of 1214-m, 1321-m and 1416-m, respectively.

The selected fan initially operates at low efficiency. In order to increase the fan efficiency mine resistance is increased to 2.149 gaul or 0.82 m^2 equivalent orifice should be supplied to the return airway. Figure 6.7 shows the new fan and mine characteristic curve.

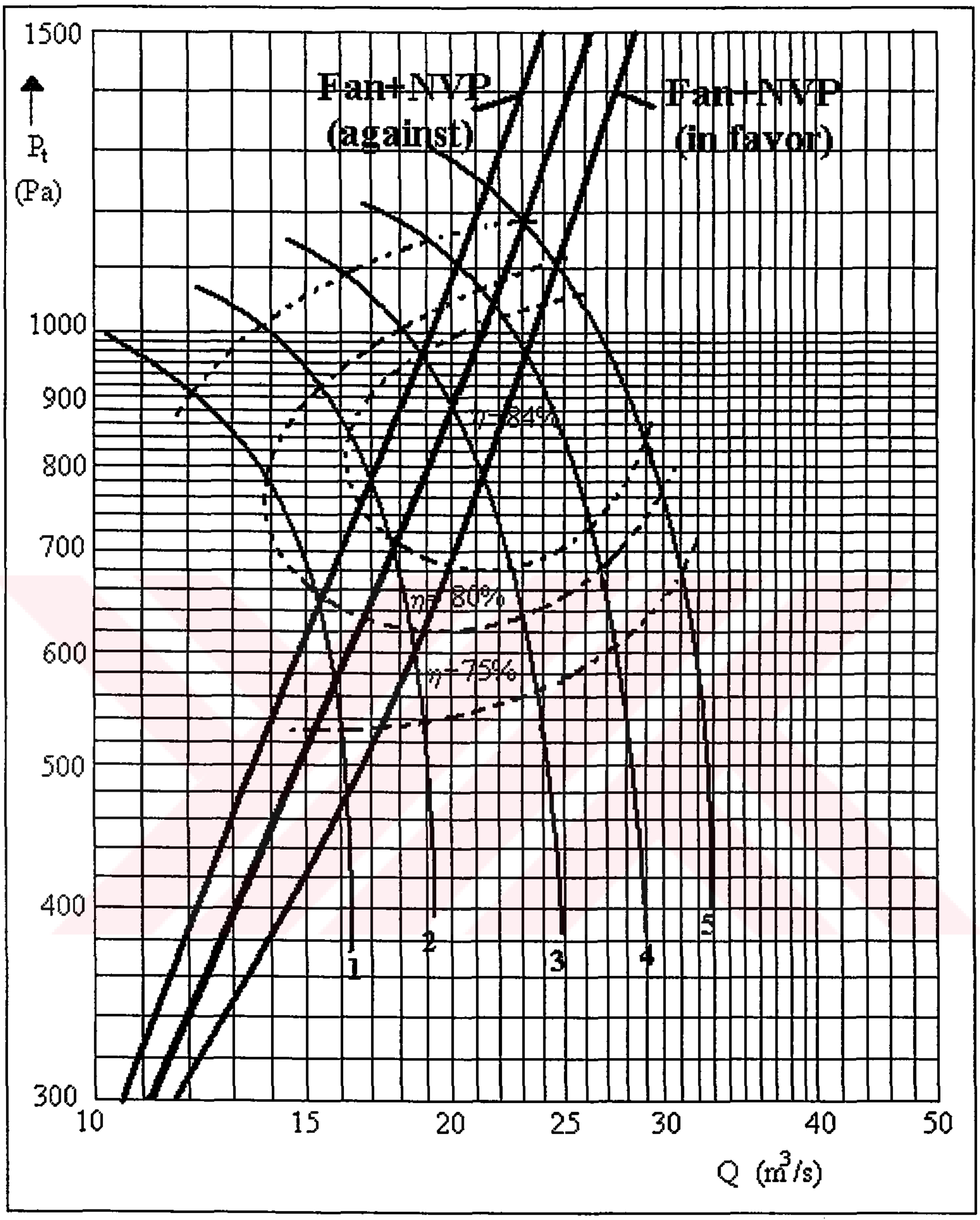


Figure 6.7 Rearrangement of the mine characteristic and fan

The ore deposit of 1304 main gallery at west side will be produced by open pit mining method. The 1235 gallery has very good and straight geometry. It has been connected to east part by a drift with smooth inclination and intersects a raise from the 1214 to the1321 gallery at an elevation of 1265-m. so it is considered as main intake airway for ventilation design consideration.

The east side galleries were important due to high ore production and the reserve amount they have. The production started from very high elevation, being about 1657 m, by open pit mine at this side. Then underground mining started. Although there are old working places on top of the 1416 gallery, there is not any connection to surface because all galleries shut downed by open pit mine overburden. This gallery which is still open was considered as main return airway in order to increase the natural ventilation pressure quantity. The 1321 is under operation and it was considered as main return airway. Dimensions of the orebody are not determined exactly; the miners only follow ore and surrounding rock contaction. Therefore ventilation system should be considered for further developments. The last main airway is the 1214 gallery. It is the most important main airway since the biggest part of ore deposits exists there. It is also haulage way and workers enter to the mine from here. The orebody has an average thickness of 40 m. The width of the orebody at this level lies up to 180 m. All surface facilities has been established around the entrance of 1214 main gallery. It was considered as main intake airway for the ventilation design purpose.

Main fan was initially installed to the system and the computer program was simulated. This fan is located at the main intake airways in the 1214 main airway at east side of the mine. The following outputs can be specified:

i. Airflow distributions,

ii. Pressure losses,

iii. Relative pressures,

iv. Operating cost,

v. Air power.

The results of these parameters can be illustrated on the schematics of the mine. Fan with various pressure quantities entered to the ventilation system. Fans, totally providing 20 m^3/s air quantity is sufficient for such this mine. Therefore, regarding natural pressure amount, a fan with 0.86 kPa was installed at the1214 main intake airway.

The 1304 main return airway was closed after installation of the fans. There was a big amount of airway leakage from this return airway without reaching to working places. The program was rerun to specify the new situation of the ventilation system.

Outputs have shown a better ventilation condition than the original case. But, it must be considered for the future developments of the working panels and in case for deeper investigations or productions. A booster fan would be suitable in the 1214 level to provide higher amount of air quantity. A booster fan with 0.2 kPa pressure was installed in the ventilation system.

The booster fan could provide very good ventilation conditions. But there is very big amount of air leakage from the branch number 18. For a better design of ventilation system, this point must be taken into account, and it was decided to close this airway like the 1304 main return airway.

Finally, by completing these changes in the mine, with small arrangements of branch resistance, the ventilation system can provide better comfort in the mine environment. For instance branch number 23 may be redesigned by regulators. If the resistance of this airway increases the amount of ventilating air to new working place will increase.

CHAPTER VII

CONCLUSIONS AND RECOMMENDATIONS

This study indicated how computerized network analysis and information from pressure and quantity surveys enabled evaluation of alternatives to improve ventilation system of the existing mine. Analysis of the experimental data and results of computer simulations show that air quantity supply by natural means and dust concentration are at critical levels in the mine throughout the year.

The computer application for calculating air distribution and distribution control in ventilation networks provides processing of a number of alternatives and realization of most suitable solutions over very short time periods.

The major conclusions based on the field study and computer applications are summarized in the following:

i. Airflow velocity in the main galleries changes between a range of 0.0 to 1.27 m/s. in April and August, respectively. It is very low for the most of mining activities such as drilling operation.

ii. The difference in dry-bulb temperature of the mine air was only 3.25 ^{0}C (in August 16.50 ^{0}C and in December 13.25 ^{0}C) throughout the year. On the other hand the difference between atmospheric temperature was 27 ^{0}C (in August 33.50 ^{0}C and in January 6.50 ^{0}C). These differences indicate low degree of humidity in the surface air.

iii. The difference in wet-bulb temperature of the mine air was 4.0 ^0C (in August 14.75 ^0C and in December 10.75 ^0C) throughout the year. This quantity is slightly higher than that of dry-bulb measurement and shows the presence of water in the mine. On the other hand the difference between atmospheric temperature was 24 ^0C (in August 29.50 ^0C and in January 5.50 ^0C). This also indicates the low degree of humidity in the atmospheric air.

iv. Airflow direction in the mine changes with respect to unequal air density of inside mine air and atmospheric air as a result of natural ventilation pressure. Airflow direction was upward in November, December, January, February and March, but airflow direction was downward in the remaining months of the year. The change in ventilation direction effects mining operation layouts, such as the settling of intake and return airways.

v. Maximum and minimum airflow quantities by natural means were obtained as 18.24 m^3/s in August and 1.28 m^3/s. in December. The air quantity required for the mine is higher than that supplied by natural means. Therefore, mechanical ventilation is essential for this mine.

vi. Required air quantity of the mine is approximately 18 m^3/s for production rate of 600 tons ore per day. This air quantity was not achieved by natural ventilation but a few days of August and January. In order to supply the desired air quantity to the mine a fan by Korfmann Company with 1000-mm blade diameter was selected which can supply up to 38 m^3/s with approximately 900-pascal pressure. Mechanical ventilation should be applied to the mine as soon as possible.

vii. Blade number 3 is the most suitable for the selected fan since, this blade provides the required air quantity of the mine with maximum efficiency.

viii. Total resistance of the mine is 0.967-gaul (Ns^2/m^8) and at this resistance the fan efficiency drops by approximately 50%. At the initial stage of mechanical ventilation, this resistance is increased to 2.149-gaul and accordingly fan operates at 860 Pa to supply 20 m^3/s air quantity. The resistance is achieved by putting a regulator with an equivalent orifice of 0.82 m^2. In the further development of the mine or in the case of higher production levels, increasing the equivalent orifice can easily decrease this resistance and the new required air quantity is satisfied.

ix. Optimum operating point of the fan is 0.86 kPa for supplying 20 m^3/s. air to the mine.

x. There was not any air circulation in the mine due to equal density of the mine air and atmospheric air in sometime in March, April, December and November. Mining activity was completely interrupted as a result of insufficient ventilation at these days. This situation causes some economical losses for the mining company. Therefore, by mechanical ventilation this economical loss will be overcome.

xi. Natural ventilation pressure quantity of the mine has a range of between 1.58 to 321.78 Pa. in November and in August, respectively. These quantities are high enough to take into consideration for ventilation planning of a metal mine.

xii. Natural ventilation pressure was zero in the last week of March, first few days of April, last few days of October and first ten days of November. Ventilation of the mine should be achieved fully by mechanical ventilation at these days for continuous mining operation.

xiii. Dust concentration level of the mine during production changes between 32.82 and 50.66 mg/m^3 in August and in November, respectively. These high dust concentration rates exceed maximum allowable dust concentration ratios throughout the year. Dust suppression techniques should be applied to the mine for decreasing dust concentration to a tolerable range. At present, the mining activity is interrupted due to excessive dust concentration for a long time, up to four hours, between two successive operation shifts.

xiv. Dust concentration level after blasting changes between 17.95 and 29.61 mg/m^3 in August and in November, respectively. These high dust concentrations cause interruption of the mining activity too. The maximum and minimum time for decreasing the dust concentration by natural ventilation to a tolerable level of 5 mg/ m^3 after blasting is140 and 190 minutes in August and in November, respectively. Therefore, mechanical ventilation is here also essential to the mine in order to achieve better mining operation.

xv. Three types of mine gases; namely methane, hydrogenesulfur and carbonmonoxide were measured during the field study. Methane concentration was generally low and maximum occurrence was observed as 0.05% during blasting operation. Hydrogenesulfur concentration was measured in a range of 1 to 4 ppm. Carbonmonoxide concentration was measured between 1 and 8 ppm. The maximum allowable concentrations for methane, hydrogenesulfur and carbonmoxide are 2%, 50ppm and 50 ppm, respectively. Therefore, it is not necessary to take any precautions to dilute these gases.

xvi. Mechanical ventilation is a vital importance if the mine taken into economical consideration as well as health and safety.

Based on this study the following recommendations can be made:

i. Ore reserve under the 1214 main gallery is not explored in detail. For further ventilation planning; the definite reserve and boundary conditions should be investigated.

ii. The mining method applied to ore deposit is true but pillars in ore are left for supporting purpose. These pillars contain 15% of total ore reserve. Necessary studies must be conducted on filling material strength and filling facilities

iii. Larger natural ventilation pressure may be obtained if the 1416 gallery connected to 1627 old work places. Therefore, this situation needs some investigation and experimental work.

iv. Studies on dustability of chromite and surrounding rock may be helpful for better understanding of the dust generation and suppression subject.

v. If blasting time is selected at the time of day that maximum natural ventilation pressure exists, the interruption time of mining activity can be decreased.

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