PENNSYLVANIA BITUMINOUS MINE OFFICIALS STUDY GUIDE

TABLE OF CONTENTS

TABLE OF CONTENTS

MINING METHODS SECTION 1

Section 1. Relevant Underground Coal Mining Methods

A. INTRODUCTION

Coal mining, including underground coal mining, began in the U.S. in the early 1700s in Virginia (American Coal Foundation, 2005). Underground coal mining was done by pick and shovel at first, and then in the early 1800s by drilling and blasting (Bise, 2013, pp. 283-285), which is also called conventional mining. Haulage from the production areas was done by animals pulling loaded cars, and by the late 1800s, electric locomotives began to be used.

The laborious drilling, blasting and hand-loading technique continued until mechanization began to aid the mining process in the 1920s (cutting, drilling and loading machines) and 1930s (shuttle cars). By the late 1940s, continuous mining machines were developed and by the 1960s, entire mines were designed based on continuous mining.

Longwall mining was used in U.S. mines in the late 1800s, but it was not mechanized. In the 1960s, modern longwall mining began in the U.S., and its growth accelerated with the introduction of two-leg shields in the mid-1970s.

In 2013 less than 1% of the 1 billion tons of coal mined in the U.S. came from underground conventional mining. Thus only continuous mining and longwall mining are relevant today and will be covered in this section.

B. ROOM-AND-PILLAR MINING

After the portals (drift openings, shafts, and slopes) are constructed to reach a coal seam, room-and-pillar mining is used exclusively to develop the mine to the point where full production in mining "sections" is done. The full production sections are developed by continuous mining as well, but the primary production in a section may be done by either continuous mining or longwall mining, which is discussed later.

Many parameters are considered in planning and designing an underground coal mine, and these will be covered in Section 2. Two parameters are the direction or orientation of mining and the layout of the mining plan over the coal reserve to the boundaries.

The first mining from the portals is in a section called the "mains." After the mains is developed a planned distance, then "sub-mains" are developed in one or more directions, depending upon the planned size of the mine and the designed overall layout. Production sections, also called panels, are developed off sub-mains generally, but sometimes off mains as well to pick up small reserve areas that have economic value. Figure 1 shows the conception of the mains, sub-mains and sections of a coal mine.

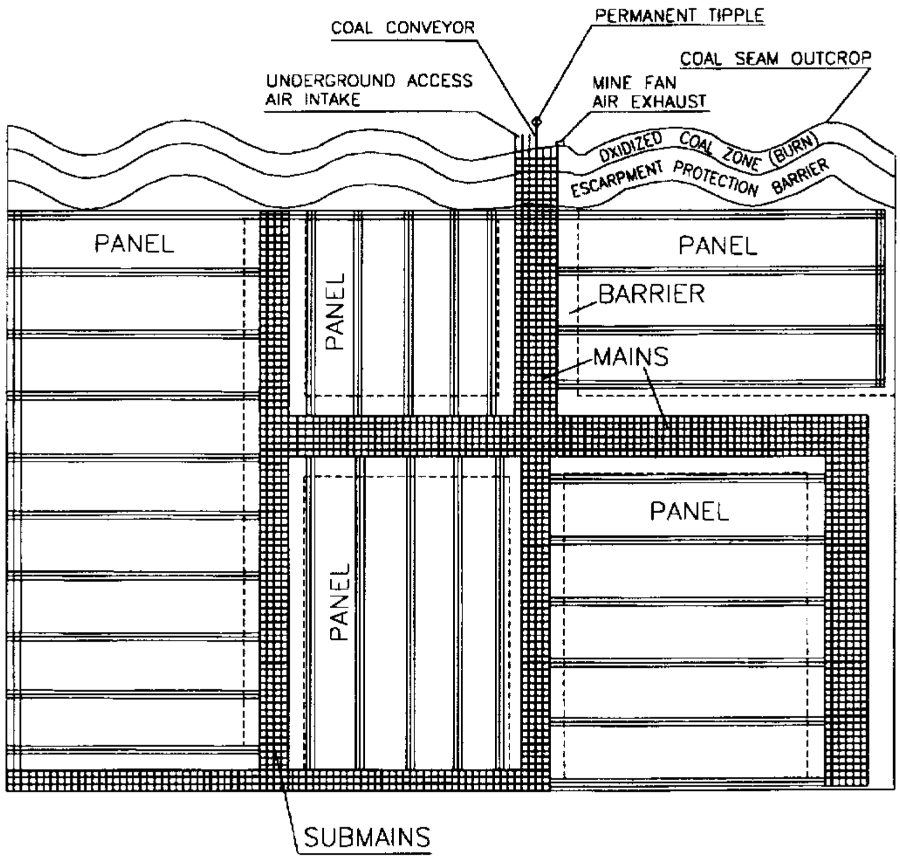


Figure 1. Conception of mains, sub-mains and panels of a coal mine (Peterson, 1998).

Additional Terminology

Development mining is accomplished by use of multiple "entries," also called "headings," in a section, and the entries are connected by "crosscuts" to enable efficient mining. In the process, "pillars" of coal are left behind for major ground support purposes. A generalized depiction of the terminology in a room-and-pillar production section is given in Figure 2. Other terminology, e.g., active workings, working place, and working section, can be found at the Mine Safety and Health Administration website at Title 30, Chapter I, Subchapter O, Part 75, Subpart A, Section 75.2, which gives definitions under mandatory safety standards for underground coal mines.

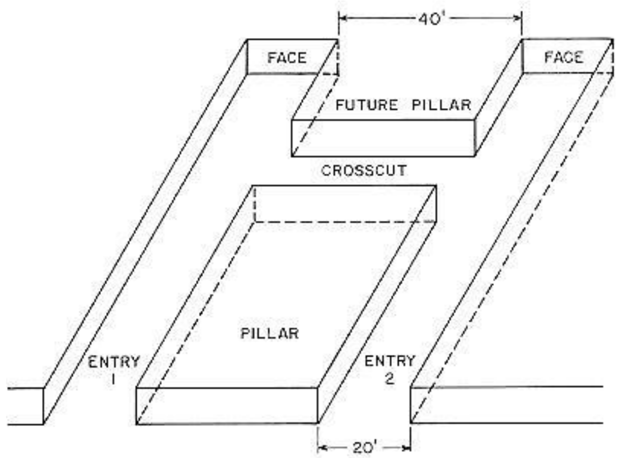


Figure 2. Terminology used in a room-and-pillar production section (Bise, 2013, p. 294).

Number of Entries

Generally, the number of entries in any section depends on the primary function of the section and the length of time over which that function is needed. More entries are needed for long-term compliance to meet legal requirements of mining laws and regulations, for preservation of designed functionality over time due to deterioration, and often for efficiency purposes. Thus entries in mains must last for the economic life of the mine, while entries in sub-mains must last for as long as the planned number of full production sections are being mined. In contrast, the number of entries in a production section off a sub-mains or the mains will last only until the section has reached its planned distance limit, possibly with some "retreat" mining of pillars on the way back out. One possible working section configuration is given in Figure 3 along with a cutting sequence using continuous mining.

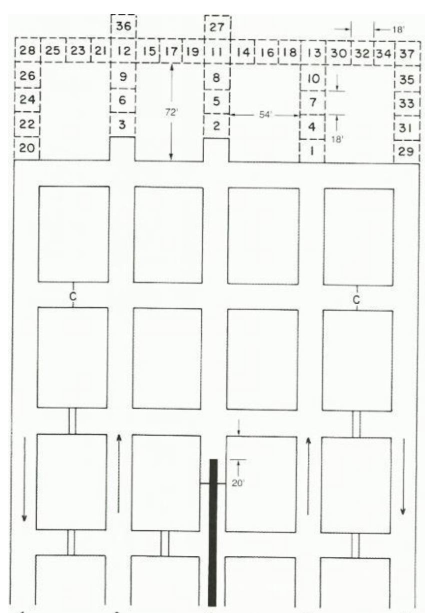


Figure 3. Continuous mining working section with cutting sequence (Bise, 2013, p. 298).

Approximately 49% of underground coal in the U.S. is mined by continuous mining. Figure 4 gives a picture of a continuous mining section in action. The operational parameters, crew size, equipment, and logistics of a continuous mining section will be described in Section 3.

Figure 4. Picture of a continuous mining section in action (Coal Leader, 2015).

C. LONGWALL MINING

Approximately 51% of underground coal in the U.S. is mined by longwall mining. In order to set up a longwall section, development must first occur using continuous mining. Most often in the U.S. three entries are used in developing the longwall "gate" entries, although sometimes either two-entry or four-entry longwall development sections are used under certain circumstances to be described later.

Generally multiple longwall sections are mined in sequence up a mains or sub-mains. For the first longwall section, both sets of "tailgate" and "headgate" entries must be developed. After the first longwall production section is fully developed, the next headgate section is developed, and when longwall mining of the first longwall production section is complete, its headgate entries become the tailgate entries for the second longwall production section. This sequence continues until all of the planned longwall production sections are completed in the mains or sub-mains area. Figure 5 shows a generalized layout of multiple longwall sections in a mains.

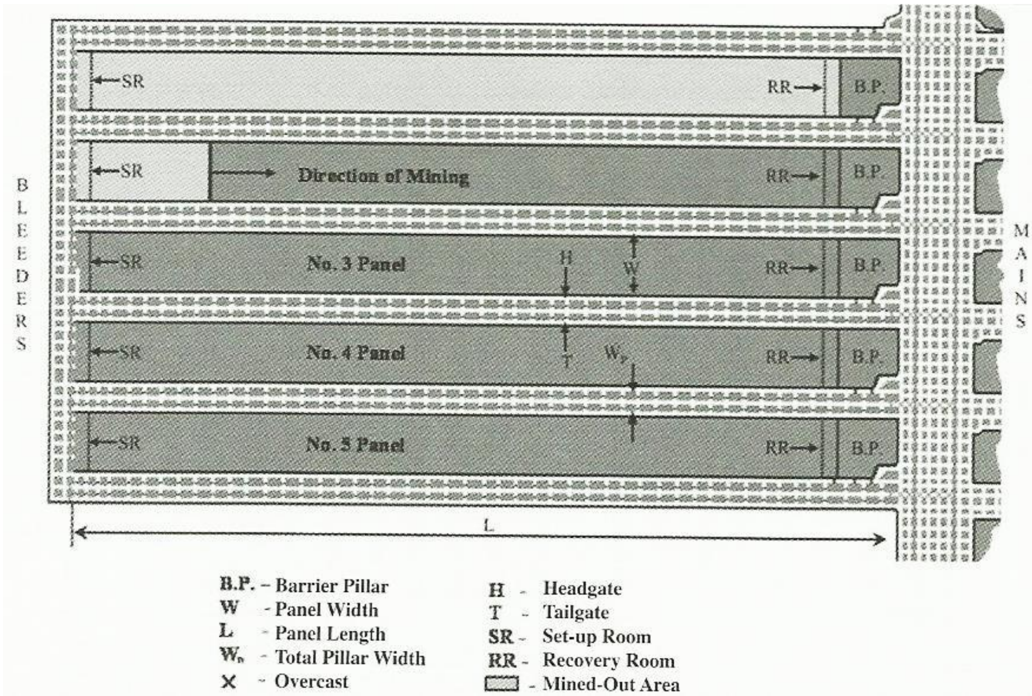


Figure 5. Generalized layout of multiple longwall sections in a mains (Base, 2013, p. 312)

In the 2012 longwall survey of 49 longwall set-ups in the U.S. (Coal Age, 2013), longwall coal section lengths (47 total), requiring continuous mining development, ranged from 2,400 ft. to 21,500 ft. The face width ranged from 650 ft. to 1,580 ft. Thus very large parts of a coal reserve are mined by a longwall section.

Figure 6 gives a picture of a longwall face being mined. The operational parameters, crew size, equipment and logistics of a longwall section will be described in Section 3.

Figure 6. Picture of a longwall face being mined (Grayson, 2014**TABLE OF CONTENTS**

MINING METHODS SECTION 2

Section 2. Mine Planning and Design – Underground Coal

A. CONSIDERATIONS IN MINE PLANNING AND DESIGN

The primary objectives of coal mine planning and design are to achieve the following:

- Highest possible safety for personnel;
- Lowest cost per ton of product;
- Maximum productivity per employee shift;
- Most suitable quality of final product;
- Maximum recovery of reserve; and
- Maximum environmental consideration.

The complex planning and design process involves many inter-dependent variables and considerations. Mine planning and design is very site specific, too. As in the stages of exploration and economic evaluation, each stage of planning and design refines the accuracy of details of all aspects as various alternatives are evaluated.

Following completion of the decision to open a mine, after the final feasibility study, more specific plans and design of details must occur. The plans and designs must utilize myriad data and information gathered in the exploration and evaluation stages and brings the mine to "life." Details of all subsystems must now be designed so that they all work together in the overall mine complex in the safest, efficient, and cost-effective way.

For an underground coal mine, the subsystems for which details must be designed include the following:

- mine access and access methods (shaft, slope, drift, elevators, skip hoist, etc.),
- mine bottom facilities (shop, staging area, landings, lighting, etc.),
- the general mine layout (length, orientation, configuration),
- each production unit,
- the specific production equipment (extraction, roof bolting, section haulage, etc.),
- the ground control and roof support system (pillar sizes, entry dimensions, types of support, plan),
- the ventilation system (intakes, returns, escapeways, methane and dust control, auxiliary fans, face ventilation system, plan),
- the electrical power system (type, voltages, currents, equipment, cables, grounding),
- the drainage system (pumps, pipe sizes, routing, sumps, dams),
- the fresh water system (pumps, pipe sizes, routing),
- the haulage system (people, coal, supplies, equipment),
- work crews (sizes, responsibilities, skills),

- all health and safety support systems,
- all surface facilities (offices, shop, , and
- miscellaneous equipment and supplies.

Once the plans and designs are in place, there are many things which could potentially require changes and affect mine safety, mine cost, and productivity, such as:

- seam thickness,
- depth of cover,
- roof quality,
- floor quality,
- degree of methane liberation,
- hardness of the coal,
- quality of the coal,
- pollutants generated,
- water influx.
- workforce (human) aspects,
- new environmental regulations, and
- new safety regulations.

Short-, medium- and long-term mine plans, along with the associated economic and financial evaluations, are an ongoing process throughout mine development and the life of the mining operation. Many unanticipated changes will occur, and formal plans may change immediately or on a weekly, monthly, or quarterly basis. Each year a new annual plan is always made, and five-year, ten-year and/or life-of-mine plans may also be done.

B. DETAILED CONSIDERATION OF MAJOR DESIGN FACTORS

Once final-feasibility study exploration has been completed, information and data collected include physical and geotechnical information. This information is needed by the mine planning and design team for detailed design of mine structures and layout. For good understanding, major factors in design are described next in some detail.

1. GEOLOGY

General

Note: This part on geology was condensed and revised somewhat from the corresponding part from the previous version of the Mine Foreman's Study Guide (1992 version).

The study of the earth is called geology. This science attempts to explain the origin of the earth and its relation to the other heavenly bodies; it attempts to explain the reason for the atmosphere, the waters which flow on or cover parts of the earth, and the rock and minerals which form the land surfaces; and it attempts to reconstruct the history of life, going back beyond the age of humans and giving us the story of animals that first roamed the earth and telling us how they lived.

In discussing the rocks and minerals which form the outer crust of the earth's surface, several aspects must be considered. Moisture and certain gases, either separately or in combination, will cause decay of rocks; extreme changes of temperature aid in this decay as do acids carried in solution in ground water and plants and animals. These actions are partly mechanical and partly chemical, and taken altogether they result in the overall effect called weathering.

Solid is a term which misrepresents the true condition of rocks. The bedrock, which forms the outer shell of the earth, may appear to be solid but it is everywhere more or less shattered; that is, there are cracks and fissures, both large and small, running through it in all directions. It is in such cracks and fissures that the weathering action of the air takes place. Air enters the openings, carrying with it moisture and the various gases found in the atmosphere. Moisture itself is a good solvent (i.e., it will dissolve material), but when carbon dioxide or some of the other acid gases are present in the moisture, its solvent action is greatly increased. Oxygen dissolved in moisture changes the chemical composition (i.e., chemically it oxidizes) of certain materials that are insoluble (i.e., they will not dissolve); carbon dioxide in the moisture then converts these oxidized minerals to different compounds that are soluble in water, and also works on insoluble calcium compounds. The other acids carried in the moisture contribute to these changes, with the result that the rock is eaten away along the various cracks and is weakened. Constant repetition of this process results in the rock's breaking into several pieces and the decomposing action continues on each of these pieces.

Heat and cold as agents of weathering must be included with the work of the atmosphere. Rock masses which are subjected to extreme heat, as in some desert regions, will expand in the same manner as metals; when they cool, they contract to their former size. Constant repetition of this will, of course, result in the rock masses being broken into a number of small pieces.

Rocks, Disturbances and Mountain Formation

Rocks

In the weathering of rock, erosion of soil, and deposition of sediments, these sediments are compacted over geologic time and transformed into rocks called sedimentary rocks. If there were no crustal movements, eventually erosion would virtually cease and the earth would be essentially level. Since crustal movements do occur, and we have not yet discussed them and where the original rocks came from, these will now be discussed.

Igneous Rocks

The term igneous is applied to rocks that have solidified from molten material. All theories of earth formation agree that our planet was very hot at one time in its history, and an outer crust of hard igneous rock formed as it cooled. Undoubtedly, the cooling of the earth resulted in internal stress which would cause cracking and buckling of the outer crust. Under the force of such readjusting movements, the material inside the earth was forced to the surface as molten material. Thus, material held inside the earth in a molten state at enormous temperatures and pressures flowed to the earth’s surface to cool and form igneous rocks when cracks and fissures provided a passageway and the pressure on the material was reduced. We may assume, then, that the outer crust of the earth, which we call the surface, did at one time in its history consist entirely of igneous rocks.

Sedimentary Rocks

As the earth cooled, the moisture retained by the atmosphere was condensed and fell as rain to fill the depressions on the surface. With the appearance of this water, erosive forces came into being and the processes previously described began to tear down the land surfaces and carry this material to the oceans. Initially the only materials available for this erosion were the igneous rocks forming the elevated parts, so we find our first sediments were taken from igneous material.

The sediments thus deposited were probably laid down on areas under water near the continents. If subsidence took place in such areas of deposition, thick beds of sediments of various kinds were deposited. Later if there was an elevation (gradual uplift) of that same area, the sediments, now consolidated by pressure into rock beds, were exposed. The forces of erosion then proceeded to weather and carry away the sedimentary rocks as well as the igneous rocks, and combined the sediments from the two to form new types of sedimentary rocks. However, it must be noted that not all sedimentary rocks were deposited in this manner; some are the result of deposition o wind-borne material.

Metamorphic Rocks

While the aforementioned processes were occurring, the contraction of the earth continued and molten material (magma) from the inside was forced to the surface. This magma came in contact with both igneous and sedimentary rocks, and its great heat changed their characteristics. The term metamorphism means changed in form, so we apply the name metamorphic to this third type of rock. Not all metamorphic rocks have been produced this way; heat which accompanies pressure, as in the case of downward pressure due to great thickness of overburden and lateral pressure due to movements of the earth's crust, will also cause metamorphism of rocks. In fact, this type, called regional metamorphism, is responsible for most of the changes in rock and is thus the most important.

Present-Day Rocks

The processes producing the three types of rocks just described have been going on for millions of years. It is only natural to expect that all of the original igneous crust would have disappeared long ago; also, that sedimentary and metamorphic rocks would be found most abundantly, with igneous rocks interspersed throughout them because of molten intrusions of recent ages. Such is the case, regardless of what portion of the earth is inspected. Following is a tabulation of the various types of present-day rocks and the materials from which they were formed:

- Sediments: gravel, sand, silt and clay, lime deposits

- Compacted strata (rock): conglomerate, sandstone, shale, limestone

- Metamorphic rocks: gneiss and various schists, quartzite and various schists, slate and various schists, and marble and various schists, serpentines

- Igneous rocks: coarse-grained feldspars such as granite, fine-grained feldspars such as felsite and tuffs, ferromagnesian rock such as dolerite and basalt

The sedimentary rocks associated with coalbeds belong to one or more of the four classes of compacted strata given in the list. When certain sediments were deposited to form limestone or sandstone, we could hardly expect them to consist of pure lime deposits or pure sand; there would be some mixing of sediments. As a result we find limestone rocks containing small amounts of sand, and sandstone rocks containing small amounts of lime carbonate – rarely do we find a rock of a single element.

A rock belonging to any of the four main classes of compacted strata which, in addition, contains sand is classed as an arenaceous (sandy) stone; when small amounts of lime are present, the name calcareous (limey) is applied; and with small amounts of silt and clay, the name argillaceous (shaly) is applied. Thus we may have the following types of rocks associated with coalbeds:

- Conglomerates
- Pure limestone, argillaceous limestone, arenaceous limestone
- Pure shale, calcareous shale, arenaceous shale
- Pure sandstone, calcareous sandstone, argillaceous sandstone

Incidentally, overall on the earth's surface limestone rocks are estimated to be about 5% of all sedimentary rocks, sandstone rocks about 15% and shale rocks about 80%.

Fractures, Joints, Faults and Rifts

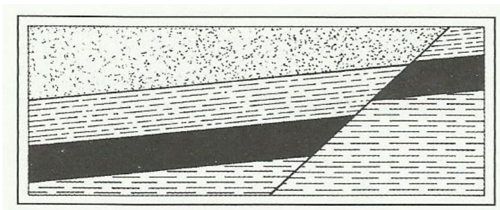
We have mentioned the many fractures that are found in rocks at the earth's surface and which play such a large part in the weathering processes. Some of these fractures are quite small while others are fairly large; the name joint has been applied to the small fractures regularly disseminated throughout rocks; the large more randomly oriented fractures have been termed rifts or fissures. Joints or fissures might have been open at one time, and generally are open at the surface, but below the surface they may be filled with mineral matter. Regardless of whether they are open or tight, they represent a break in strata and are fractures.

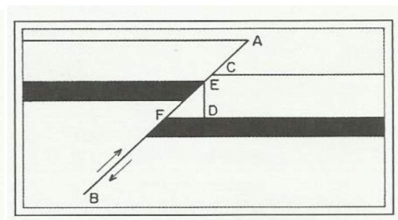
Stratified rocks, of which coal may be taken as an example, may be full of joints. Sometimes these will run in fairly definite directions and may intersect at right or other angles to form blocks of definite shapes. Such jointing may arise through contraction which the bed underwent when raised from under water and allowed to dry out. Or, again, twisting of the beds due to movements of the earth's crust may have produced the joints. Either way, the joints are extremely important in mining and quarrying because they make extraction easier and also affect roof and ground control.

Sometimes the rifts extending through the crust of the earth are the scene of movement.

The rock masses on one side of the rift move in some direction while the rock masses on the other side remain stationary. Such a displacement is called a fault.

Faulting is due either to compression or to stretching of the outer shell of the earth's surface. The stress produced exerts a pressure on the rock masses which finally give way at their weakest points, or along the fractures. Two types of faults are depicted in Figure 7 (normal and thrust fault). The readjustment of the rock masses forming the earth's crust is going on continuously; in some parts of the country, slight shocks due to faulting are being recorded every day. Sometimes, the stress accumulates until a tremendous movement, called an earthquake, takes place.





Normal (gravity) fault

Thrust fault

Figure 7. Illustration of a normal fault and a thrust fault. (Stefanko, 1983)

Mountain Formation

Mountains may be classified as having originated in one of three ways: by igneous agencies, by erosion, or by movements of the earth's crust. Igneous agencies represent the molten magmas which, under pressure, come to the earth's surface and either pile up material to form a mountain, as in volcanoes, or push on strata to aid in forming mountainous ridges. All three of these ways are depicted in Figure 8.

Formation of mountains by erosion occurs when an uplifted area of land is worn away The harder portions of the surface rocks resist the eroding action and become the mountains, standing out in contrast to the rest of the countryside. The Catskill Mountains of New York are of this type. Erosion of sedimentary rocks above and around igneous intrusions also produces mountains of this type.

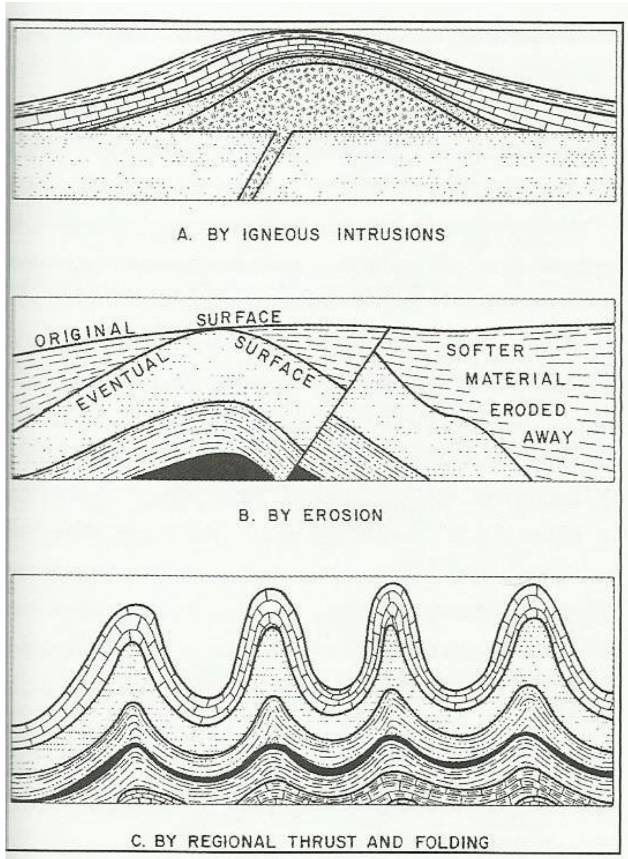


Figure 8. Examples of mountain formation. (Stefanko, 1983)

The earth's movements produce by far the greatest number of mountains. These movements may be divided into two types: faulting and folding, each of which has been responsible for the formation of great mountain ranges. In addition, faulting and folding combined have been responsible for the formation of some mountains, and igneous agencies have also assisted. Thus a grouping of forces has produced mountain ranges of great structural complexity.

The formation of folded mountain ranges required certain preliminary conditions. In the first place, most of the material forming the strata (the plural of stratum, or bed) came from other land surfaces. Since the total thickness of these strata is greater (over 25,000 ft. for the Appalachian Mountains), tremendous amounts of this material had to be transported and deposited. We can suppose, therefore, a large, trough-like area that was slowly sinking over a long period of time with a large and higher land mass close by from which the material could be taken. The sinking trough, sometimes several hundred miles across, is called a geosyncline.

The vertical movement of this trough area could not have been constant since the nature of the materials being deposited changed many times. It was deep enough at times to form a sea and permit the formation of thick beds of limestone from the remains of small sea animals. At other times, it rose to shed its water and become a swamp where coalbeds could form. The number of such changes in the nature of the material is an indication of the number of times such directional changes in the vertical movement took place.

The mountain building, or folding, process took place when these accumulated strata in the geosyncline were first raised and then subjected to enormous lateral-thrust pressures. The strata were folded or broken or both as the edges of the geosynclines were actually pushed toward each other. As soon as the folding caused this area to rise higher than the surrounding areas, erosion started to work on it and wear it down. In the case of the Appalachian Mountains, many of the coal seams were removed by such action long before humans appeared.

While most of the eastern United States was inundated during the coal-producing Pennsylvania epoch, the depth of the sea varied as well as the number of fluctuations between land and water. Because of the deeper inundation of the Interior Province over longer intervals of times during this period, fewer coal seams were formed there compared to the Eastern Province of Appalachia where approximately 100 minable seams have been encountered over the whole range of time during this period. Post-depositional changes eroded away the coalbeds formed in what is now western Ohio and eastern Indiana.

Origin of Coal

It is now generally accepted that coal is of vegetal origin, that the geologic processes which in past ages produced the great beds of coal we mine today are still operating to form deposits which are the basis of coal, and that the several kinds of coal now mined are the result of different degrees of alteration of the original material. The formation of coal represents the final result of the accumulated efforts of organisms, of erosion, of deposition of sediments, and of movements of the earth’s crust. An illustration of the coal-forming areas in the U.S. during the Pennsylvanian geologic epoch is given in Figure 9.

Vegetal material (derived from vegetation) is largely composed of carbon, hydrogen, and oxygen. It also contains mineral substances and in some cases a small amount of nitrogen. The chief substance composing the framework is cellulose. For the purpose of explaining the formation of peat from which coal is derived, we will consider cellulose as forming the organic matter of plant life.

When dried organic plant matter, or cellulose, is burned in the presence of normal air, it is oxidized completely with the formation of carbon dioxide and water. If the burning takes place where the supply of oxygen is limited, as when wood is burned in a kiln or with some earth thrown over it, the oxidation is incomplete. In this case part of the carbon and all of the hydrogen and oxygen are removed from the cellulose, leaving pure carbon in the form of charcoal. The decay or rotting of vegetal matter is a process of oxidation caused by the growth and action of bacteria and other extremely small organisms. If this decay takes place in normal air, as when leaves, small plants, or trees fall to the ground, the oxidation may proceed to completion.

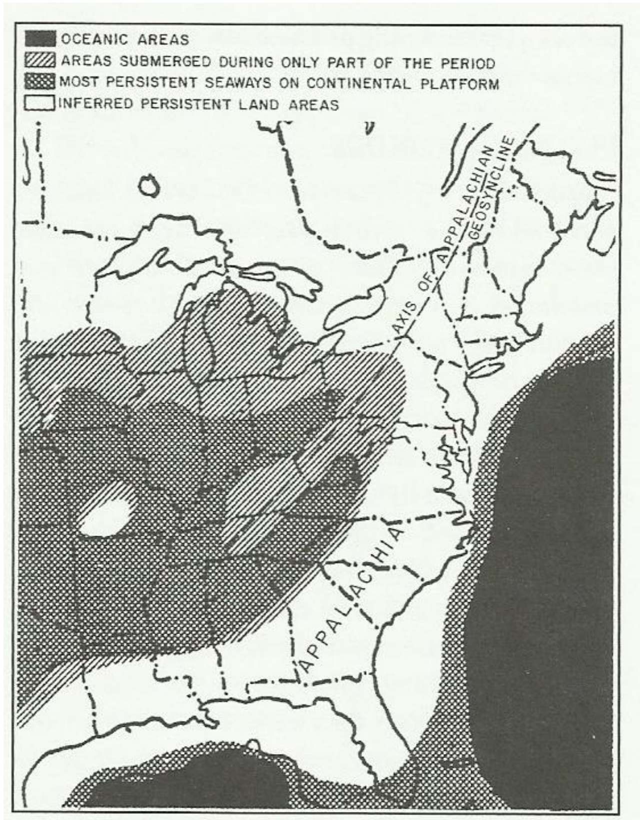


Figure 9. Pennsylvania epoch of coal formation in the U.S. (Stefanko, 1983).

The decay of organic matter in the absence of oxygen, as when it takes place under water, parallels the forming of charcoal. The absence of oxygen results in hydrogen being removed in the form of water (H_2O) with some of the carbon removed as carbon dioxide (CO_2) or carbon monoxide (CO), and some of both removed as methane (CH_4).

The product formed by the initial decomposition of vegetal matter is known as peat. During the formation of peat, the woody material becomes greatly altered, both chemically and physically. The extent to which this biochemical action proceeds affects the structure of the coal that will be formed from the peat.

Peat can be formed in bogs, marshes, and freshwater swamps. Peat formation begins in these shallow watery depressions. Plants begin to grow around the fringe of the lake, then decay and fall into the water, forming a marshy edge. The first plants to develop in the lake are the pond weeds and water lilies. Farther out will be found the bulrushes, and beyond them will be the floating plants that do not reach to the bottom. As this vegetation grows and dies, the remains become water-logged and sink to the bottom where the slow disintegration previously mentioned transforms the woody material into peat. A deposit of peat thus forms along the shore and, as it approaches the level of the water, the various plants gradually shift their positions toward the center of the lake. Later, a larger grayish-green or whitish moss, called sphagnum or peat moss, appears on the exposed peat surface. This is followed still later by small trees of the coniferous species. As the peat becomes still firmer, larger trees of the deciduous species, such as white birch, may replace the conifers, and eventually the lake will be filled and the area overgrown with trees. The tree remains become part of the peat bog as they wither through natural death or are blown down during storms.

Neither peat bogs nor marshes are considered as important as freshwater swamps for the accumulation of the extensive, thick peat deposits of the past that produced the coals mined today. The Dismal Swamp of Virginia and North Carolina is probably the best present-day illustration of the conditions that favored peat deposition. This swamp adjoins the Atlantic coast line and has an elevation that varies from about 5 to 25 ft. above sea level. At the present time, it is slowly sinking, thus providing favorable conditions for the deposition of peat and sediments. The formation of peat here follows the description given previously for peat bogs except that in the Dismal Swamp there are frequent inundations of the area as the water level rises.

The Dismal Swamp peat is formed by the falling trees, seed spores, leaves, and other debris varying from 1 to 20 ft. in thickness, and it has been estimated that 1500 square miles of the swamp is covered with peat to an average depth of 7 ft. The characteristics of this peat deposit indicate that, if turned into coal, it would compare quite favorably with present-day coals in quality and ash content. However, it would provide a bed with a maximum thickness of only 20 inches (estimated) and would not compare in size with the more important commercial coalbeds now being mined.

The rate at which peat accumulates depends on the rate of vegetation growth and its subsequent rate of decay. The conditions under which the vast deposits of peat of past geologic ages were formed did not differ greatly from present-day conditions. It is probable that the climate was relatively warm, that there was a more abundant and more regular rainfall than at present, and that the vegetation that was suitable and properly located for peat deposition was more plentiful than now. As a result of these conditions, the rate of peat deposition was possibly twice as great during the coal-forming geologic periods as it is today. Considering all factors, it has been estimated that the time required for the deposition of peat sufficient to provide 1 ft. of thickness of the various ranks of coal was: lignite, 160 years; bituminous, coal, 260 years; anthracite, 490 years. These values are given for comparison only, for it should be recognized that the basis on which the calculations were made was an estimated period of time. They will serve, however, to indicate the difference in the time periods necessary for the formation of beds of different ranks of coal. For example, an 8 ft. bed of Pittsburgh (bituminous) coal required about 2100 years for the deposition of the necessary peat, while an anthracite bed with a thickness of 30 ft. (measured where the bed was level) required about 15,000 years. We now recognize that the thickness of the coalbed can give a false impression of the thickness of the original peat bed where the strata has been subjected to faulting and folding, and some of the thick anthracite beds may be the result of horizontal compression of thinner peat beds.

Topographic Conditions Favorable to Coal Formation

Examination of the great coal deposits of the world indicate they were formed in areas with the following characteristics:

- at or near sea level on which abundant vegetation could grow,
- poor drainage so that a swampy condition prevailed,
- slow subsidence so that thick peat deposits could accumulate, and
- near higher areas that provided sufficient drainage to maintain a freshwater condition.

The coal-forming periods followed earlier periods during which shallow, continental seas covered wide areas of the various continents. Withdrawal of these seas left large expanses of low-lying land poorly drained and excellent for the development of vast swamps.

During those periods when conditions were favorable for the deposition of peat, thick deposits were formed in all of the submerged areas. In the Appalachian trough, these deposits were formed in the many depressions that had resulted from the buckling and warping of the geosyncline strata (see Figure 10 for an illustration). It is probable that general subsidence of this geosyncline was taking place all of this time, with small areas within the geosyncline subsiding while adjacent areas were being raised by crustal movements. This would explain, in part, the greater thicknesses of peat deposits in certain local areas of this geosyncline. The deposits in the areas covered by the interior seas west of the Cincinnati arch were more uniform in extent and thickness. Thus the vast coalbeds in the Midwest and the Appalachian region were formed at the same time.

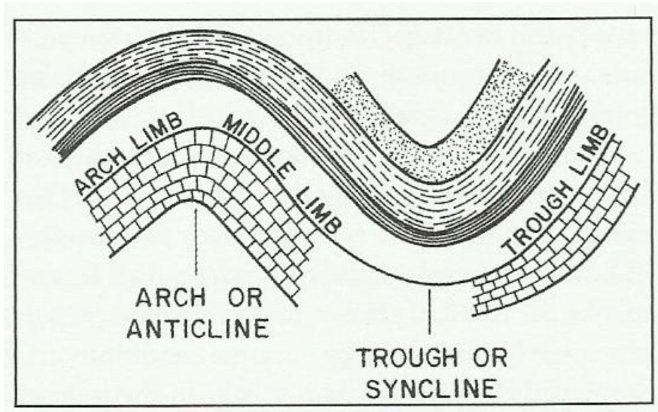


Figure 10. Illustration of geologic anticline and syncline from folding. (Stefanko, 1983)

When the subsidence of the coal-forming areas permitted encroachment of the seas, marine sediments would be deposited, thus forming the limestone beds that are frequently associated with coalbeds. When there was slight elevation to or above the level of the sea, sediments from the nearby elevated lands would be deposited on the peat, and these sediments formed the sandstones and shales that are also associated with coalbeds. Elevation above sea level was probably never extensive or the coal deposits would have been eroded and wiped out. Local elevation undoubtedly did allow some erosion of the coal forming material, and later deposition of sediments provided the wants, cut-outs, or rock faults that are now encountered in many coal mines. Seas began to withdraw from the North American continent, a drier climate prevailed, and the deposition of coal-forming material was interrupted. The eastern part of the continent was affected by the Appalachian revolution which elevated the land and permitted erosion of sediments and portions of the coal deposits. Twisting and contorting of the strata had much to do with the type of coal formed from deposits in certain areas.

The transformation of peat into lignite (brown coal), the lowest rank of coals, is now considered to be due entirely to pressure from the overburden. Peat swamps which subside sufficiently to permit deposition of sediments from nearby raised land may have an overburden of sediments from a few feet or several thousands of feet in thickness.

The great pressures required for the transformation of peat into bituminous and higher ranks of coal have been the result of horizontal thrusts that accompanied crustal movements after the peat beds had been deposited and covered with a great thickness of sediments. In a few isolated instances, igneous intrusions in nearby sedimentary beds have caused the formation of anthracite coal from a lower rank of coal, but the great mass of higher rank coals has undoubtedly been the result of thrust pressures that accompanied mountain-making movements. All of the world’s anthracite and other high rank coals are found in areas that have suffered crustal movements at some time. The anthracite region of Pennsylvania is a good example of the twisting and faulting of the coalbeds and adjacent strata by crustal movement, in this case, during the formation of the Appalachian Mountains. As we proceed westward from the area of greatest disturbance, we find that the violence of the crustal distortion lessened with distance, the thrust pressures decreased, and the coals became progressively lower in rank. Coals that are of lignite quality in North Dakota became sub-bituminous farther west, then bituminous, and finally anthracite in the mountainous regions of Montana and Utah. The rank of these coals increased as the beds in which they occurred and which are of the same geologic age were subjected to successively greater thrust pressures to the west.

The change from peat to coal is accompanied by a compositional decrease in moisture, oxygen, and volatile matter (carbon dioxide, carbon monoxide, and other gases), and an increase in the percentage of fixed carbon, sulfur, and in many cases ash content. The devolatilization of the coal, indicated by the increase in fixed carbon content, has a bearing on its rank. For example, in the Appalachian region, the fixed carbon content varies from 55 to 60% in Ohio coals and from 60 to 65% in western Pennsylvania coals, while central Pennsylvania coals may contain as much as 83% fixed carbon and anthracites located where the crustal movements were most pronounced will show as high as 97% fixed carbon.

Coalbeds are always associated with heavy, strong beds of sandstone, conglomerate, and limestone, and with weak beds of clay and shale. When thrust pressure is applied to these beds, the weaker ones yield while the stronger beds resist deformation. Pressures are applied horizontally, vertically, and at all angles to both horizontal and vertical, to the beds and the associated sedimentary strata are frequently distorted and shortened from their original horizontal lengths. Coal behaves as a plastic mass, especially during the early stages of its transformation, so it is capable of accommodating itself to any space that the distortion of the stronger beds provides. Thus, coalbeds that are normally a few feet thick on the flanks of a syncline or anticline will be many times thicker at the bottom of the syncline or the crest of the anticline, and there will be no other trace in the coalbed of this movement.

The geologic age of a coalbed has no direct bearing on its rank. If all other factors were equal, the older beds would be of higher rank because the transforming processes would have had a longer period of time for action. However, there are marked differences in the ranks of coals of the same geologic age, sometimes in the same bed, and these differences are directly traceable to devolatilization of the coal through the effect of thrust pressures resulting from crustal movements.

In summary, the progressive devolatilization, loss of moisture, and consequent increase in the rank of coal are produced by several geologic factors rated in approximate order of decreasing importance as follows, with the first three factors by far most dominant:

- pressure and heat associated with depth of burial,
- time,
- structural deformation,
- heat of nearby intrusive igneous rocks, and
- plant composition and environment of coal accumulation.

2. GROUND CONTROL PRINCIPLES

Note: This part on ground control principles was condensed and revised from the corresponding part from the previous version of the Mine Foreman's Study Guide (1992 version).

Introduction

Not only is ground control as important as ventilation to safety and productivity, they are also strongly interrelated. They must be considered together with design compromises usually having to be made between them.

A large area for airways is desirable in ventilation and this is most easily achieved by widening the entries. However, with increased width, destructive stresses resulting in failure may be produced in the mine roof, floor, and pillars causing serious limitations in practical width. Thus increased airway area is usually achieved by driving multiple parallel entries with intervening pillars, but there is a limit, too, to the span that can be supported from one solid pillar rib to the other. Taking into consideration the properties of the mine structure (roof and floor rock and the coal seam), the proper sizes of openings and pillars must be established as well as the maximum number of parallel openings that will remain stable. While the principles of rock mechanics can be helpful, this is an inexact science dealing with variable materials, and the situation is rarely encountered, at least in coal mines, in which ground stability can be achieved merely with proper sizing of pillars and openings. Usually, artificial supports must be installed to satisfy regulatory requirements and also to achieve stability. The nature of supports will vary according to whether continuous mining or longwall mining is being done.

Note that the term ground control is used here rather than the more common roof control. While control of roof usually is of the greatest importance because roof falls produce the greatest number of fatalities historically, especially within 25 ft. of the face (the danger area), the reactions of the roof, floor, and ribs to ground stresses are interrelated, and so the term ground control is preferred. It becomes apparent then that ground control is a three-part process:

- proper sizing of openings,
- proper sizing of pillars, and
- selection of proper artificial supports.

The ultimate objective is to maintain mine structural stresses below the strength values of the structural elements (roof, floor, pillars) to avoid failures. In this section, the principles governing the sizing of openings and pillars will be covered as well as the principles for selecting artificial supports. Specific types of artificial supports will be covered in sections on continuous mining and longwall mining separately, because there are significant differences.

Imposed Stresses and Stress Concentrations

Figure 11 illustrates a vertical profile of a horizontal coal seam lying at some depth below the surface. Since the overburden loads the coal seam, by using the average specific gravity of sedimentary rock (2.6 sp. gr.), a vertical stress of 1.1 pounds per square inch (psi) per foot of overburden depth will be the pressure on the coal, represented by the following formula:

$$\sigma_v = 1.1 \text{ D}$$

where σ_v is the original (virgin) vertical stress in psi and D is the depth of the overburden in feet.

As an example, if a coal seam lies at a depth of 1000 ft., then the vertical stress would be 1.1 X 1000, or 1100 psi.

Since σ_v is the maximum compressive (crushing) stress, it is designated a principal stress, and the vertical broken lines indicating directions of the principal stresses are referred to as principal stress trajectories.

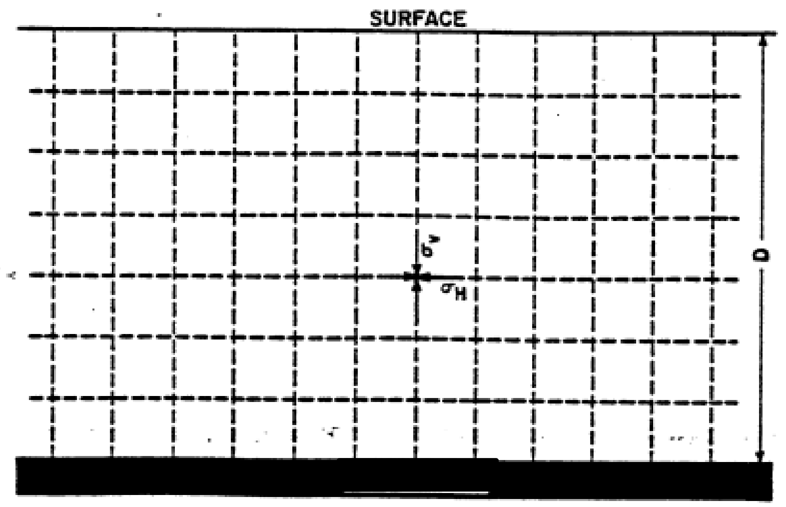


Figure 11. Original stress distribution of a virgin coal seam.

When the overburden rock and coal seam are loaded by these vertical forces, they try to expand laterally to provide stress relief, but, as they cannot do this because of their confinement, a lateral horizontal stress (σ_h) is produced which represents the least compressive stress, and it, too, is termed a principal stress. Thus the complete stress trajectories, shown by the vertical and horizontal broken lines in Figure 12, include both the major and minor principal stress trajectories, and are mutually perpendicular.

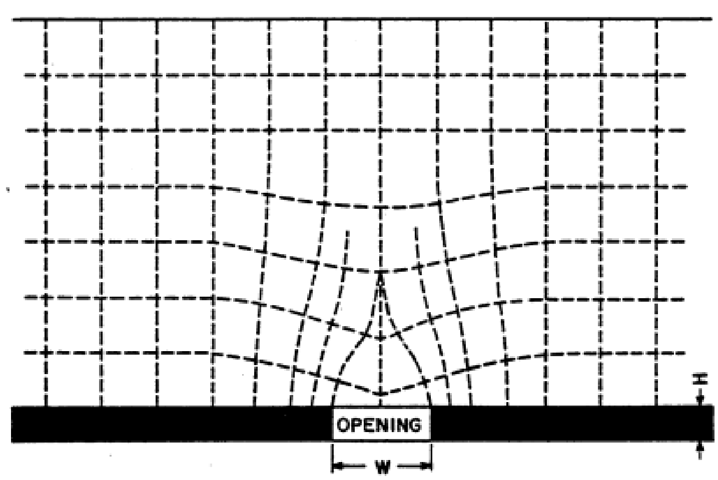


Figure 12. Stress trajectories around a single opening (vertical and horizontal).

It is important to note that these principal stresses reflect only the superincumbent overburden load. There are many regions in the U.S. where there are effects of either past or present tectonic geologic activities that result in horizontal stresses which may be in excess of the vertical stresses. There is significant variation in the magnitude of the horizontal stress across these regions, and considering these stresses in designing mine openings and pillar sizes requires special considerations in design. Because of the difficulty in determining horizontal stress, according to Heasly (Base, 2013, p. 51), "using the dead weight of the overburden is quite often the best method to calculate the in situ vertical stress." Thus, once the horizontal stress field is determined through existing knowledge or field research, then the calculation of opening and pillar sizes must be recalculated using more sophisticated techniques.

When a mine opening is created in a coal seam, the coal removed is no longer available to support the overburden directly above it. If the roof rock is competent, however, it will act as a bridging beam and the overburden load existing directly above the opening will be transferred to the sides or ribs of the opening and onto the pillars (see Figure 12). The squeezing together of the stress trajectories in the ribs indicates concentrations of stress or high stress gradients. While the stress trajectories still remain perpendicular, this distortion of the rectangular elements representing the stress trajectories is obvious and is an indication of the additional presence of shear stresses. Actually, throughout any loaded geologic structure as shown in Figures 11 and 12, an infinite number of stress trajectories exist because a force per unit area (stress) exists at every point; only a few are shown to make the drawing simpler and visualization possible.

From another perspective, Figure 13 shows the distribution of the critical stresses around a single opening in massive overburden rock comprised of different strata. Analytical, laboratory, and field studies have been conducted to determine the distribution and magnitude of these stresses around openings and how they might be minimized. We now know that the geometric shape of the opening and its width-to-height ratio (W/H) significantly affect the stress concentrations around it.

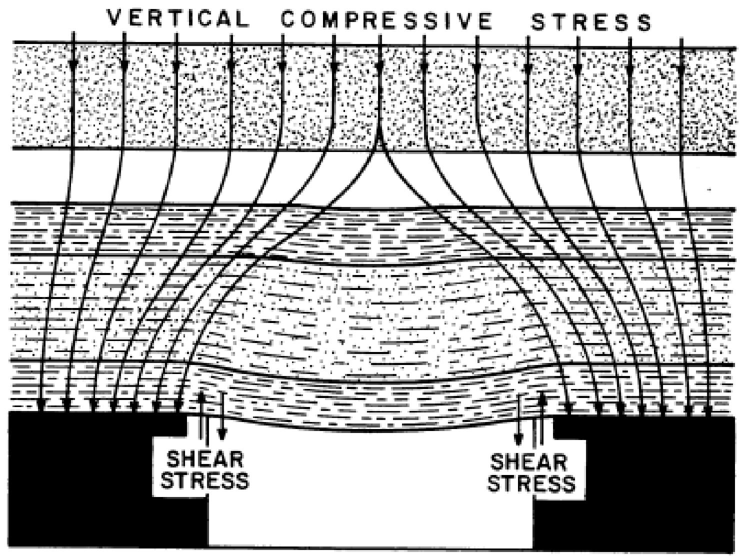


Figure 13. Critical stresses around a single opening in a massive overburden.

The maximum stress concentration occurs on the periphery (skin) of the opening. However, it is highly localized and rapidly dissipates until in the center of a pillar that is three times the width of the opening, the original stresses remain. Since the stresses are concentrated within a zone 1.5 times the width of the opening, providing a pillar at least three times the width of the opening ensures no superpositioning of stresses from one opening on another; the stresses around the various openings then act independently.

Stresses resulting from two independent sources may accumulate or be superposed at a third point. This is the law of superpositioning of stresses, and care must be taken to avoid such a situation.

It should be apparent that, for elastic materials, a very high stress concentration can exist on the ribs of an opening as a result of adverse geometry and unfavorable W/H ratios. If this stress exceeds the compressive strength of the pillar, the pillar will rupture and the tremendous amount of elastic strain energy stored during loading will be released, causing violent upheaval of the fragmented materials. This phenomenon, termed a rock burst or a coal bump, will occur in coal mines under heavy cover of strong and massive overburden strata in an elastic seam that has a high compressive strength. Such bump conditions prevail in only a few coal seams in the U.S., primarily in southern West Virginia and Utah, and they can be very dangerous and require careful mine design.

Since coal is sedimentary in origin and bedded sedimentary rocks overlie coal seams, the beds vary greatly in their individual competency and thickness. As long as there is no separation along bedding planes, the stress distributions for massive rock already explained will prevail. However, because the bonding between bedding planes is usually weak, as the overburden weight comes down on the roof rock over an opening, the rock bends downward into the opening and, if allowed to deform sufficiently, will ultimately separate along the bedding planes, creating a very high tensile stress at the top of the separated beam directly over the rib that can initiate failure (see Figure 14). As the rock fracture propagates downward, the beam is detached and falls into the opening, and the stress condition repeats itself in the overlying layer with fracture continuing upward layer by layer until stability is finally reached. The progressively decreasing beam span ultimately results in reduction of the maximum tensile stress below the rock tensile strength because the tensile stress is proportional to W^2 and inversely proportional to the beam thickness (t), thus:

$$\sigma_t = W^2/t$$

It can then be stated that there is a maximum width of opening that can be successfully bridged and still provide support for the overlying rock; the caved or otherwise affected zone over an excavation extends upward a distance twice the width of excavation to form an ellipse with its major axis vertical, as shown in Figure 14. Actually, this height of the ellipse is not constant, but also varies with other rock properties. This so-called maximum-pressure-arch concept is not limited to a single opening, but also holds true for a set of parallel entries with intervening pillars as long as the pillars yield and transfer the stresses to abutments over the solid ribs.

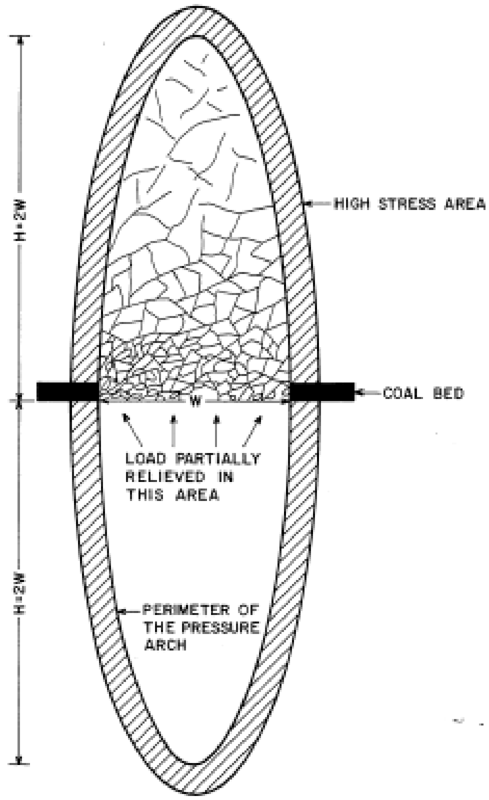


Figure 14. Arching effect around a mine opening.

Mine Openings

Because the rapid build-up of destructive stresses with increasing spans is well recognized, mine openings are limited by law to widths of up to 20 ft. when a single method of artificial support is used. Although the narrower opening may be more stable, there is a practical lower limit (about 12 ft.) below which highly mechanized mines with bulky machines cannot operate effectively. Therefore, most mine openings will vary in width from 12 to 20 ft., with the shallower and more competent rock permitting the larger span. The length of time the opening must remain stable also influences its safe width. Rock and coal are subject to deterioration over time.

At one time, the width of opening was a more critical factor affecting production than it is today. In conventional mining, with the large number of places and pieces of equipment required for cycling, wide places were preferred because it required virtually the same time to prepare a narrow place as a wide one, and the wide one provided much more coal and greatly reduced nonproductive tramming time. However, because the continuous mining machines now almost exclusively used, except on longwall faces, are all narrow-ranging machines, production is no longer greatly affected by narrow work, especially since the depth of cut today is usually much greater (19 to 35 ft.) than it was in conventional mining.

In summary, for the greatest stability, the width of openings should be limited, the width-to-height ratio (W/H) should be as low as possible, and the preferred shapes of openings oval, rectangular, and elliptical, in that order.

Pillar Size

The dimensions of pillars are largely determined by their strength properties, what is expected of them, and the period of time over which they must remain stable. We have already seen that shallower, short-lived pillars are small, whereas the deeper, more permanent pillars can be very large.

3. LAYOUT OF MAINS, SUB-MAINS, AND PRODUCTION UNITS

For each minable area within a mine property and based on core hole data near it, a suitable and practical design of the size of mine openings and pillar sizes for the mains, sub-mains and production units (sections) within the area, over the appropriate timeframe, must be done. These determinations will then allow the planning of the life-of-mine layout over the entire economically feasible areas of the mine. The time period over which the integrity of mine openings must be preserved will be considered as the configurations are determined as well as the level of artificial supports required to maintain stability.

The functionality of various entries will also be determined to accommodate infrastructure needed, i.e., conveyor belts; ventilation; escapeways; intakes; returns; haulage ways for supplies, manpower and equipment; etc. In some cases, special excavations will be needed as for bunkers, shops, motor barns, dams, overcasts, personnel landings, etc.

Once all of the aforementioned considerations and calculations are complete, the number of entries for the mains, sub-mains, and various sections will be determined. Then starting at the first portals, the layout of the mains, then the sub-mains, and finally all sections will be determined.

Lastly the entire series of layouts will be made and connected in the mine map using a chosen software, which will accommodate the timing of development throughout the mine all the way to the property limits or economically determined stopping points away from the property line.

Finally, the various operational support systems that are required for the underground mining operation must be designed. These will be discussed in detail in the following Section 3.

C. OTHER MINE SYSTEMS AND CONSIDERATIONS

The support systems for the underground operation, some of which were given previously in the list of mining complex systems, include the following:

- mine bottom facilities (shop, staging area, landings, lighting, etc.),
- the ground control and roof support system (types of support, plan),
- the ventilation system (intakes, returns, escapeways, methane and dust control, auxiliary fans, face ventilation system, plan),
- the electrical power system (type, voltages, currents, equipment, cables, grounding),
- the drainage system (pumps, pipe sizes, routing, sumps, dams),
- the fresh water system (pumps, pipe sizes, routing),
- the haulage system (people, coal, supplies, equipment), and
- work crews (sizes, responsibilities, skills) for each system.

Significant considerations for the design of each will be discussed next.

Mine Bottom

The facilities at the mine bottom are numerous. The elevator (or cage) landing is where management, supervisors, miners, inspectors, and other visitors enter and leave the mine. It must be well lighted and, as with all mine-bottom facilities, well protected to ensure it functional and safe for the life of the mine, generally speaking (unless other portals are established in the future, and new mine-bottom facilities established).

Near the elevator, a staging area and landing for loading personnel into mobile vehicles (mantrips, jeeps, etc.) is needed and must be well lighted with electrical cables guarded to prevent contact of people with them.

A maintenance and supply shop is needed, and possibly a separate ‘motor barn’ where mobile transportation and haulage equipment will be repaired, as needed. Some storage capacity will also be needed for parts and supplies to sustain the maintenance function. Some space will be occupied by specialized equipment (welder, grinder, compressor, etc.). The shop and/or motor barn will have to be ventilated with a separate split of air directed to a return entry, as predicated by both federal and state law.

A landing and storage areas will be needed at the bottom of a slope, as supplies and equipment are lowered into the mine and used parts and trash are taken out of the mine. The bottom transfer area requires adequate lighting as well.

If diesel equipment will be used in the mine, primarily for various haulage purposes, then a diesel supply and distribution system will be needed. It must comply with federal and state laws in its construction and use.

Ground Support and Roof Control System

The ground support and roof control system embraces all of the variations required as development and full production progresses throughout the mine. Thus the support system on the mine bottom, constructed to preserve stability over the life of the mine, is very different from the support system of the mains, sub-mains, and sections. Haulage and travel ways in the mains and sub-mains are generally more substantial than for sections; mains must maintain stability longer than sub-mains, and the same is generally true for sub-mains vs. sections. For example, continuous mining developed sections often must remain stable for one to two years while sub-mains will require five to ten years or more.

The entries and crosscuts most used at the mine bottom will generally be supported by roof bolts on first development, later coated with a strong gunite (a mix of cement, sand, and water), and additional supports (steel beams, etc.) would be added if necessary to maintain stability of weaker rock formations.

Mains entries critical for primary ventilation, escape from a mine emergency (two required by law), travel into and out of the mine, and for conveyor belts to haul coal must be supported to maintain stability for a very long time (10 to 30 years for relatively large and very large mines). After initial roof bolting on development, additional support such as steel beams and/or cribs will be installed. A depiction of one such method of additional support along a track haulage way is shown in Figure 15. This type of support may be needed in certain troublesome areas of a conveyor haulage way as well.

Often the use of cribs in crosscuts along a haulage way or conveyor belt will be enough to stabilize roof conditions. An example of a crib built in a crosscut along a main haulage way is shown in Figure 16. Various types of cribs may be constructed; the one shown is a wooden crib.

Additional support in sub-mains may resemble the support used in mains, depending on the severity of the ground control problem, but more often wooden posts may be constructed in crosscuts and a wire mesh is installed along the haulage entry with occasional steel beams installed where needed.

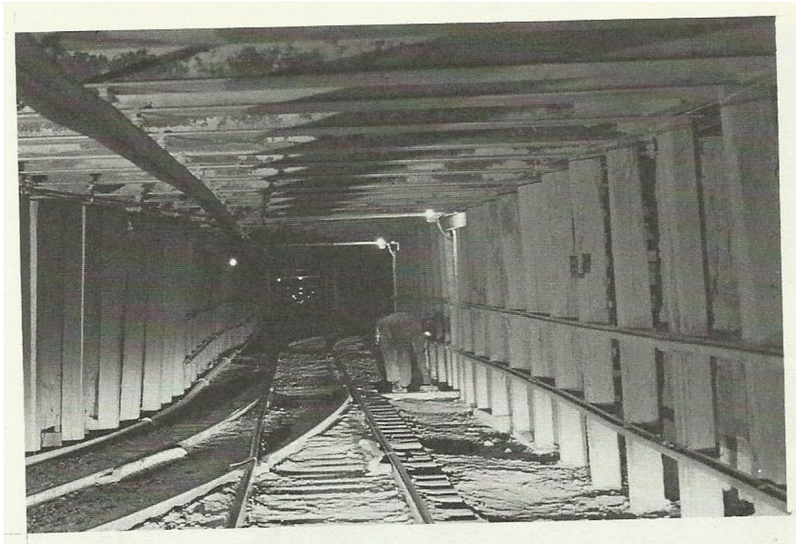


Figure 15. Illustration of additional support in a main haulage way. (Crickmer and Zegeer, 1981, p. 113).

Figure 16. Use of a crib in a crosscut along a haulage way. (Klemetti, 2013)

Less heavy support is generally needed in sections, except in longwall gate entries, unless potentially compromising conditions are encountered, in which case heavy support will be added. Often posts may be used in crosscuts, along ribs, or along the conveyor belt entry and/or haulage way for assurance of stability.

Ground control for production sections will be covered separately for continuous mining and longwall in Section 3. It is important to note that a roof and rib control plan is required by federal and state regulations, and it contains all methods of support that is or is planned to be used in the mine. The plan will be reviewed every 6 months for possible changes. Any desired changes to the plan must be approved in advance of use. Violations on not following the approved roof and rib control plan are one of the most frequent types of violations cited by federal inspectors, and fines can be very costly depending of the gravity and severity of the situation and condition of the violation.

Ventilation System

As with the ground control system, the mine ventilation system is critical for the mine, particularly in sustaining life and removing contaminants, especially potentially harmful gases and dusts. The design of the ventilation system hinges on determining a sufficient number of entries, both intake and return, for the mains, sub-mains, and sections such that the mine total head and total quantity of airflow provided are optimized. The details on the design of the mine ventilation system are covered in another volume, except for the ventilation of working faces, which will be covered under continuous mining and longwall mining separately in Section 3.

Electrical Power System

In order to operate all electrical equipment and devices, the electrical power system much be designed based on the overall requirements for voltages and currents needed. Protective devices must also be included in the design. The design of the electrical power system will be discussed in another volume of the study guide.

Drainage System

The design of the mine drainage system, comprised of sumps, pumps, pipelines, and possibly dams, along will control devices, are covered in Section 5 of this volume.

Fresh Water System

The design of the mine fresh water system is similar to the design of the mine drainage system, except that clean water (not potable water) is provided for use by equipment. Normally the water is taken from a source on the surface (river, stream, municipal supply), sent down a borehole with gravity assist, and distributed throughout the mine where needed, and particularly to the production sections. If the quality of the water in a dam is sufficient (low solids and acid content), then dam water may be used.

Haulage System

The haulage systems for continuous mining and longwall mining sections will be covered in Section 3. The types of equipment options will also be covered. The outby haulage systems (coal, supplies and materials, personnel, and equipment) will be covered here, but not the detailed design of them.

Ingress of personnel, supplies and materials, and often equipment will be down an elevator or down a slope, unless drifts exist for mine entry. In a mine without drifts, egress of personnel is also by elevator, unless there is an emergency, in which case egress may be up a slope or by an emergency escape hoist at a mine shaft. Sometimes equipment may be dropped down a shaft nearer to the location where it is needed, but the major components of the equipment will normally have to be lowered separately, and the equipment reassembled.

Coal haulage today is almost always by conveyor belts, but the size and speed of the conveyors differ according to where they are located in the mine and how much coal must be handled. The conveyor belt on a continuous mining section may be 24” or 36” in width, while one on a longwall section may be anywhere from 36” to 54”, depending on the amount of coal produced per unit time.

Most medium-size and large mines have multiple production sections. There more sections there are, the wider the conveyor belts on the mains and sub-mains must be. Of course in design, the size of a belt is coordinated with the speed of the belt to achieve the desired combined flow.

System of Work Crews

Crews of miners are formed for various work functions. Crew size varies by function, e.g., production section, maintenance, construction for ventilation, drainage and fresh water systems, electrical systems, etc. A good way to determine the size of the entire workforce is to start the design at the production section level and then design the various types of support crews for the outby underground mine, and lastly design the personnel/crews needed outside the mine. All crew sizes and all other personnel, hourly and supervisory, as well as management personnel will not be covered here, but the personnel requirements for continuous mining and longwall sections will be presented in Section 3.

TABLE OF CONTENTS

MINING METHODS SECTION 3

Section 3. Section Operations and Equipment

In this section the unit operations, ground control methods, ventilation schemes, equipment, haulage methods, and responsibilities and duties of the assistant mine foreman (section boss) will be described in detail separately for continuous mining and longwall mining.

A. CONTINUOUS MINING

A. UNIT OPERATIONS

When beginning section development operations, a layout of entries and crosscuts, their dimensions, and the dimensions of pillars have already been planned and now production will begin. An optimal cutting sequence will have also been planned by management and included in the approved ventilation plan. The following unit operations will be executed:

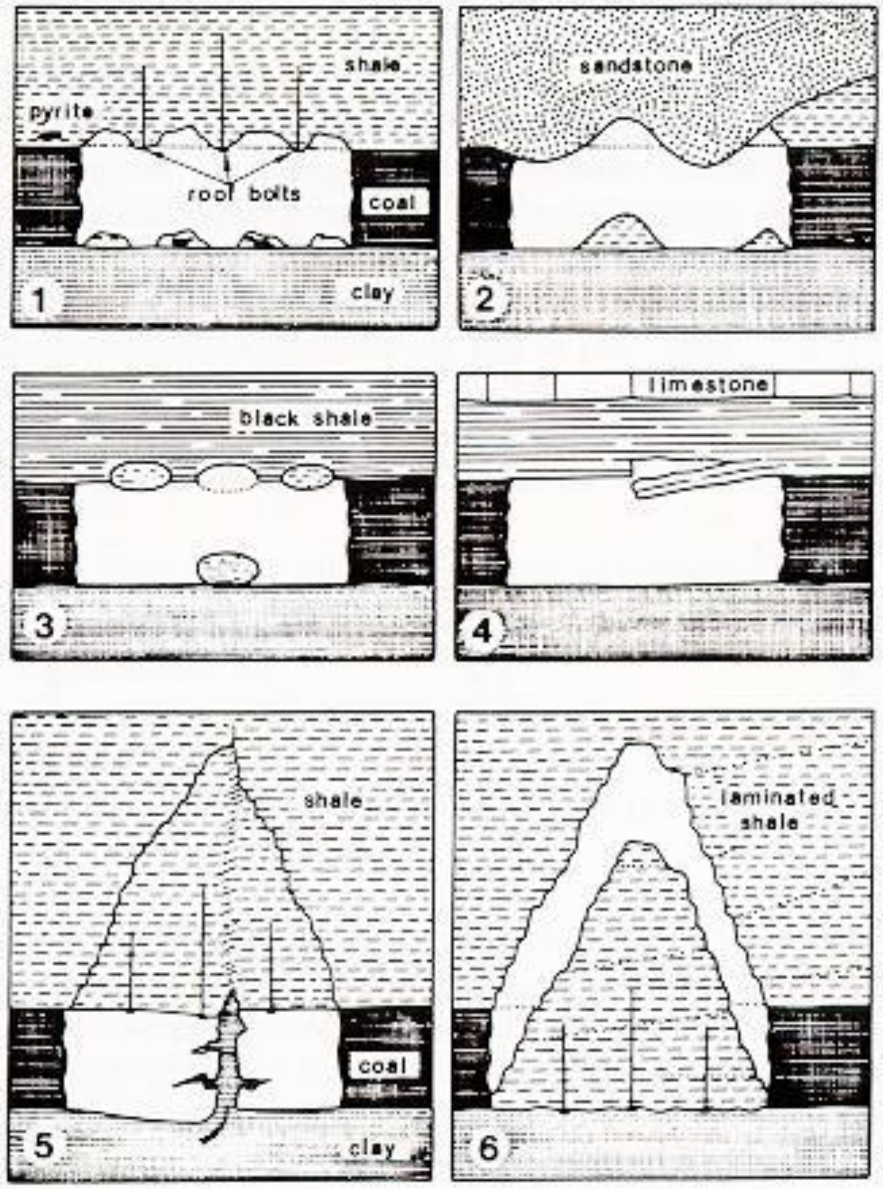
- Excavation of the coal and loading it into intermediate haulage equipment,
- Transportation of coal to the feeder-breaker for transport outby, and
- Installation of roof supports, and rib supports, if necessary.

Once an entire cutting sequence has been completed, generally the section power center and high-voltage cable, the parts car, the tailpiece of the belt conveyor, the supplies in the haulage entry, and other miscellaneous items must be moved forward, after which another full cutting sequence will be pursued.

B. GROUND CONTROL

Once mining begins, the types of roof failures that could occur in coal mines include those shown in Figure 17. There are other types, e.g., cutter roof, kettle bottoms, clay seams, faults, etc.

Mechanisms for roof support include active supports (e.g., tension roof bolts, tensioned cable bolts) and passive supports (e.g., steel arches, wood posts, cribs, and untensioned bolts). Support mechanisms for roof bolts are 1) suspension of a weaker roof by bolting into a more competent roof above, 2) building various strata of the immediate roof to form a stronger beam for support, and 3) keying different strata together across fractures in them. These three mechanisms are depicted in Figures 18, 19 and 20 (Grayson, 2014).



1. dust roof fall
2. lenticular roof fall
3. concretion roof fall
4. slate roof fall
5. clay squeeze and fall
6. massive roof fall

Figure 17. Types of roof failure in coal mines.

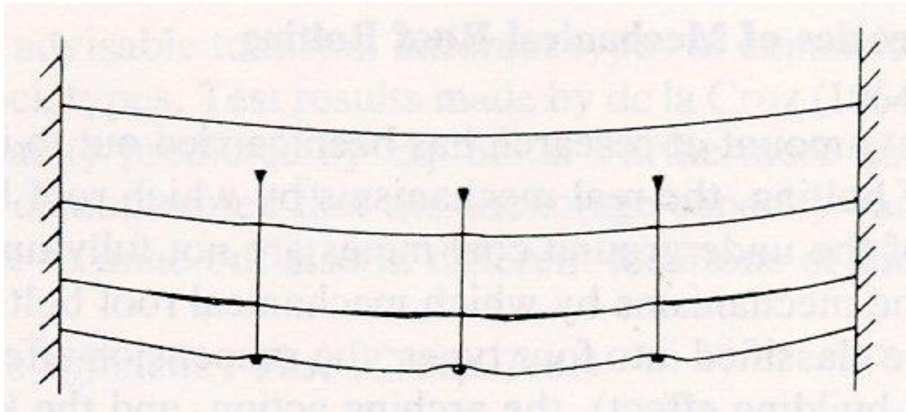


Figure 18. Suspension mechanism of roof bolting.

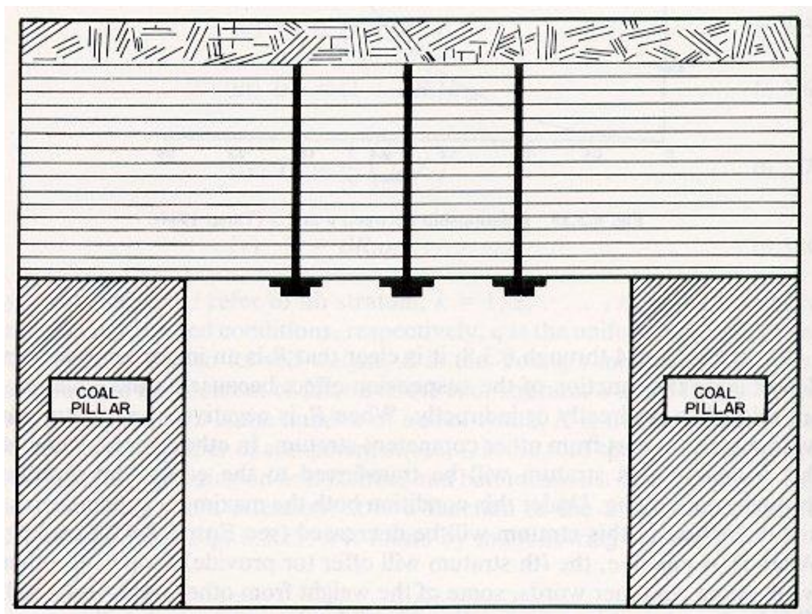


Figure 19. Beam building mechanism of roof bolting.

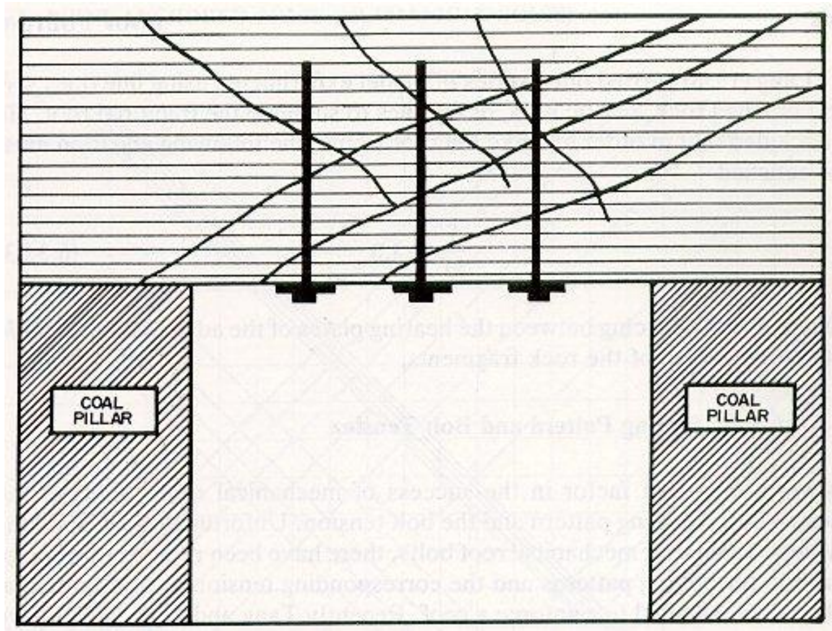


Figure 20. Keying mechanism of roof bolting.

There are also roof bolts available that use two of these mechanisms in one application, e.g., a combination bolt. Available types of roof bolts include the following (Dolinar and Bhatt, 2000):

- Mechanical anchor bolts,
- Fully-grouted bolts,
- Resin-assisted mechanical anchor (or point-anchor) bolts,
- Tensioned-rebar (or torque-tension) bolts, and
- Combination bolts.

They are illustrated in Figures 21, 22, 23, and 24. The mechanical anchor bolt was the first type to be used. It is inserted into a drilled hole and the bolt and the rock are placed under tension by expanding the shell, which is torqued to an approved torque range (given in the roof control plan). The shell grips a competent stratum and suspends the strata below from it. It is the most inexpensive type of bolt and is fast and simple to install; however, the contact area of the shell and rock is small. The roof rock needs to be reasonably competent for the bolt type to be effective over time; a weak shale rock, for example, is difficult to maintain stable. Also the hole is open to air (and moisture in it) and the bolt-rock interface can deteriorate from weathering.

It is universally recognized that roof bolts must be installed immediately upon roof exposure before bed separation occurs; later installation is less effective. Also, when mechanical roof bolts are used, proper hole diameter is essential so that the bolt can expand and grip the walls of the hole effectively. While a raveled or spiraled hole is unsuitable for mechanical anchorage, a rough, non-uniform hole may allow for great contact and friction between the resin-grouted bolt and the hole wall, although this is not universally accepted.

The more common and effective way of achieving distributed anchorage is by the use of a resin-grouted roof bolt, the grouting being either full-hole or partial. The most-used fully-grouted bolt helps prevent weathering of the bolt-rock interface by encapsulating the roof bolt. The bolt is tightly bonded to the rock around it, which is a stiff reinforcement that resists both axial and lateral loads (Dolinar and Bhatt, 2000). There is no tension initially placed on the bolt and strata, however, but some tension is created when roof rock moves. With proper installation, and especially mixing of the resin, the mechanism can be very effective. The beam that is created must resist any potential sag, which can be a problem in certain conditions.

The resin-assisted mechanical anchor bolt combines the mechanisms of tension and envelopment of the bolt over a two to three foot length into a single installation. Some weathering can occur in the parts of the bolt that are no enveloped by resin.

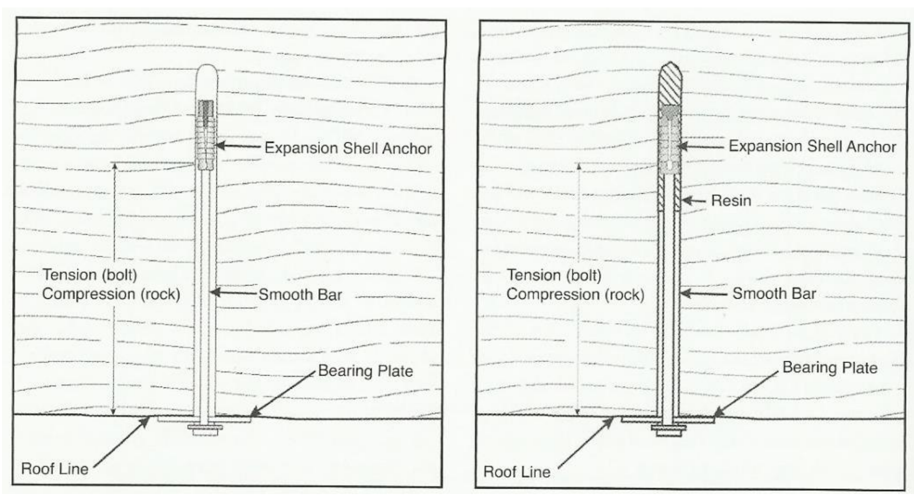


Figure 21. Illustration of mechanical anchor bolt and resin-assisted anchor bolt. (Dolinar and Bhatt, 2000)

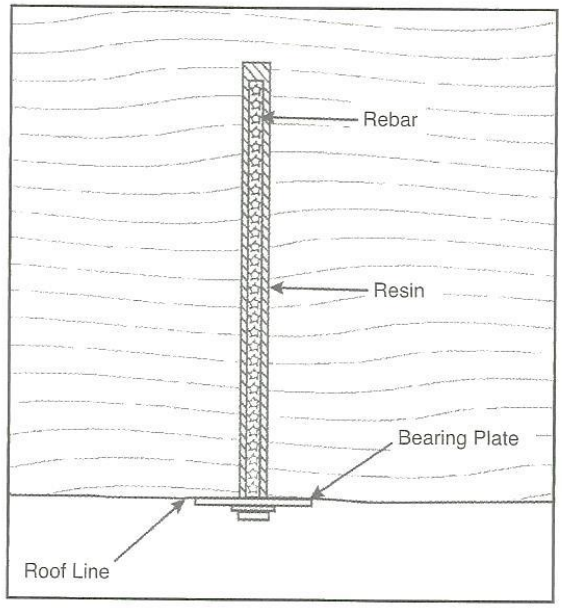


Figure 22. Illustration of fully-grouted bolt. (Dolinar and Bhatt, 2000)

The tensioned rebar bolt was designed to improve the high anchorage and stiffness capacity of the fully-grouted bolt by adding initial tension to clamp the strata together as well (Dolinar and Bhatt, 2000). This is accomplished by using two resin mixes, one of which is a slower curing one at the bottom of the hole that allows successful tensioning of the bolt. Thus the installation process is more complex and the bolt is more expensive, but the added stiffness is a benefit in some conditions.

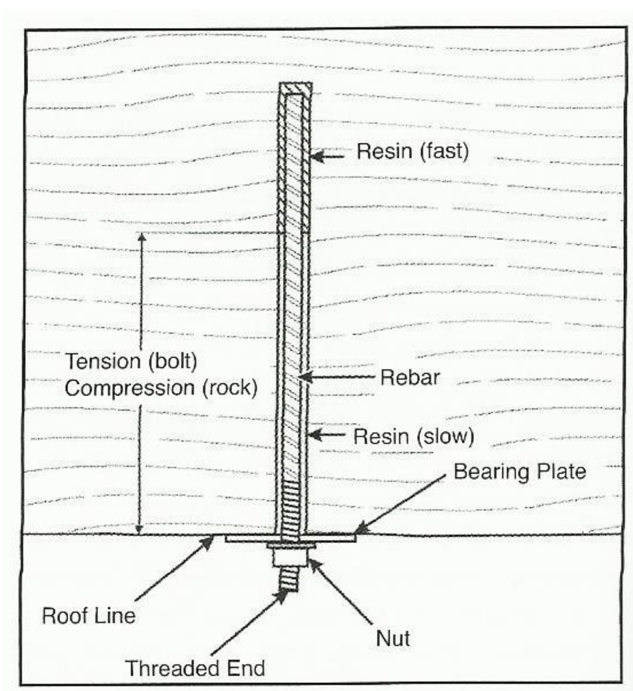


Figure 23. Illustration of tensioned rebar bolt. (Dolinar and Bhatt, 2000)

The combination bolt has a resin-grouted rebar in the upper part and a standard type bolt in the lower part which is tensioned. The two parts are coupled together.

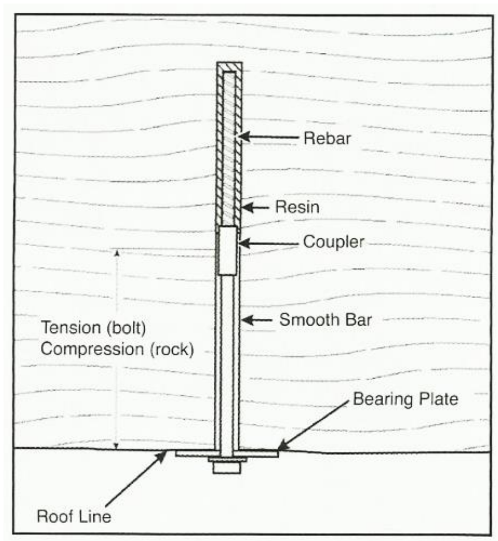


Figure 24. Illustration of a combination bolt. (Dolinar and Bhatt, 2000)

A different type of bolt, which has proven to add stability in certain adverse heavy-loading conditions, is the cable bolt, which is largely used as supplemental support. It is illustrated in Figure 25. It has seven strands with one being a "king" wire in the center. The steel comprising the strands is very strong. The length of the cable bolt can be from 8 to 18 ft., and it must be inserted manually, which can be difficult going through the resin.

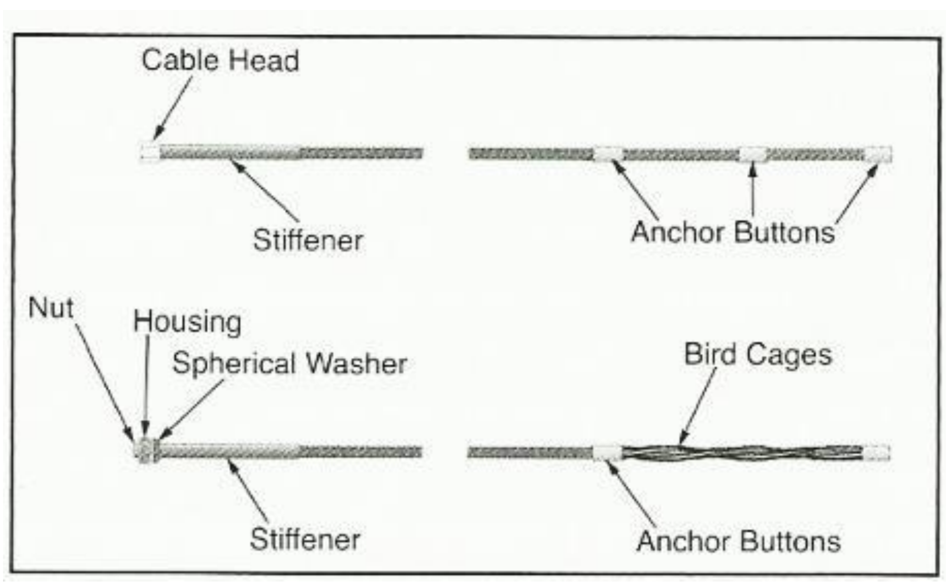


Figure 25. Illustration of cable bolts. (Dolinar and Bhatt, 2000)

One other method of support, although very expensive, is the roof truss assembly, which can be used in very adverse conditions and in intersections, particularly when stability has not been achieved with other types of support. The truss assembly and it support mechanism through induced forces is shown in Figure 26. The abutment stresses, produced over the pillar ribs by the over-bridging of the roof, create effective clamping of the expansion shell anchors in the hole. Tensioning the assembly places the rock in compression, thereby eliminating adverse tensile stresses.

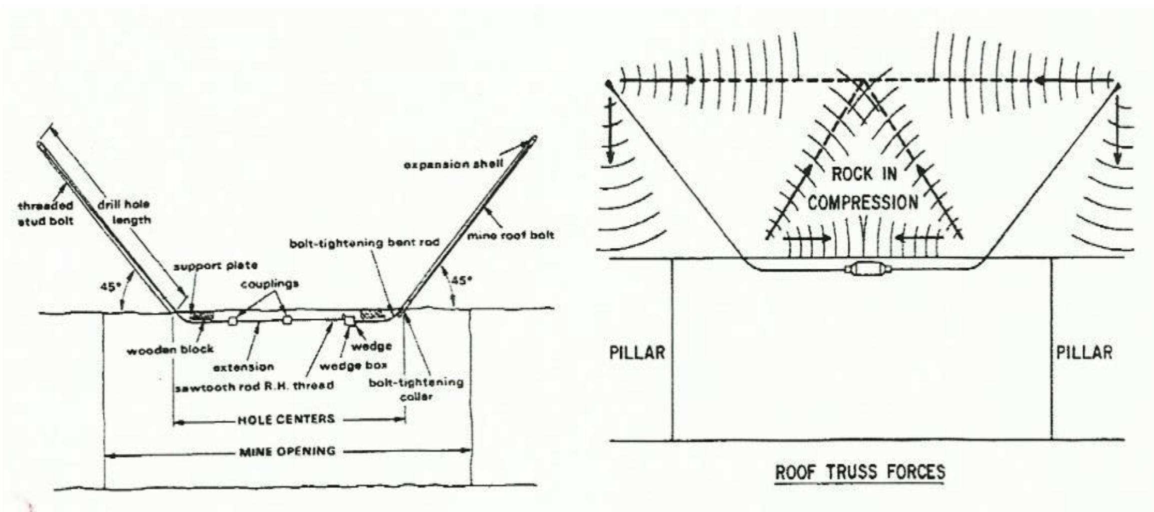


Figure 26. Roof truss assembly with illustrated of forces generated.

Other Supplemental Supports

Some types of supplemental supports, available primarily for more surface coverage and protection of the miners, are the following:

- Straps of various types,
- Large pan-type roof-bolt plates,
- Mesh (wire of synthetic), and
- Rib bolts with or without spraggs.

The proper selection of a support requires the consideration of many factors and satisfactory answers to a number of pertinent questions, which follow:

1) The magnitude and type of forces or stresses expected on the support after installation,
2) The amount of roof and bottom convergence or closure anticipated at the site,
3) How permanent the opening and thus the support must be,

4) Environmental conditions to which the support will be subjected, and
5) The economic factors consistent with safety.

Considering all of these factors, the following features should be considered in a support:

1) Strength,
2) Rigidity,
3) Mobility,
4) Permanence, and
5) Economy.

However, almost every selection procedure requires some compromise since the ideal support material for a given situation rarely, if ever, exists.

The total cost of a support must always be kept in mind. A common failing is to consider only the initial expense of materials and labor. A cheaper support is very uneconomical if it must be replaced several times. This is not only because labor for replacement is very expensive, but also because it involves additional manpower exposure to hazards and production losses resulting from support failures. Neither is it good economy to use a long-life, expensive support as a temporary support when a cheaper one will be adequate.

In any event, it is not possible to design a foolproof natural mining system. Artificial supports will be required to support excavations, if for no other reason than to provide a safety factor. Roof conditions can change so rapidly that obeying the legal requirement that no one be allowed under unsupported roof is a prerequisite of good safety practices. Experience has revealed that most serious accidents occur under what is considered good top; poor top is always carefully tested and supported, good top frequently is not. It is absolutely essential to test the roof periodically while miners are working, always using temporary support for protection until the permanent supports are installed.

Upon commencement of sectional operations, a roof and rib control plan will also have been previously approved. Optional support methods are incorporated in the plan, just in case conditions change as mining progresses. Regardless of what support method is used, the sequence of installing supports, specific types of roof bolts, and all items using in supporting the roof and ribs must be included in the plan. The approved primary sequence of installing roof bolts must be followed, or an approved optional sequence.

C. VENTILATION

Only section ventilation will be covered in this section; overall mine ventilation will be covered in another volume of the study guide. In the design of the mine ventilation system, a sufficient quantity of air is planned to be provided to each working section to meet federal and state ventilation regulations. Regulators are set in all but one split of one section to regulate the amount of air required in each section. The section foreman may not change regulator settings, which will be controlled by the mine foreman.

The section foreman, on the other hand, is responsible to maintain the quantity of air reaching the last open crosscut(s) and the working faces such that the minimum quantity of air required is always met. Thus somewhere more air is usually provided at each location to ensure that fluctuation of the airflow in the mine would cause noncompliance with a regulation. The quantity of air provided at the working faces must be sufficient to dilute, render harmless, and carrying away pollutants (dust and methane primarily) so they do not pose a hazard to miners working in the area. Thus the quantity of air required for proper dilution could fluctuate as influxes of pollutants occur. The on-shift and pre-shift examinations are done to ensure the level of pollutant influx is detected and addressed. A typical layout of a section ventilation method is shown in Figure 28.

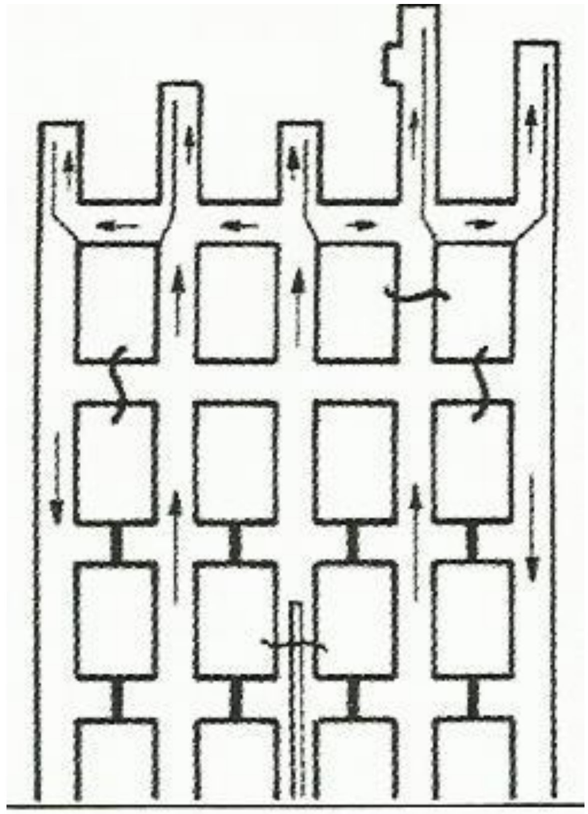


Figure 28. Typical two-split section ventilation scheme. (Bise, 2013, p. 135)

Nearly all continuous mining machines purchased today have scrubbers with fans. They are primarily used to dilute mine dust generated during cutting. The amount of ventilation induced by the fan will meet and usually exceed the quantity of air necessary to comply with federal and state regulations, and hence also control methane emissions. Different schemes/configurations of using ventilation devices for ventilating the face are shown in Figure 29. Panel (a) of Figure 29 shows the use of a diffuser fan and exhaust tubing, panel (b) shows the use of a scrubber with an exhaust line brattice, panel (c) shows the use of spray fans and exhaust line brattice, and panel (d) shows the use of a push-pull tubing system.

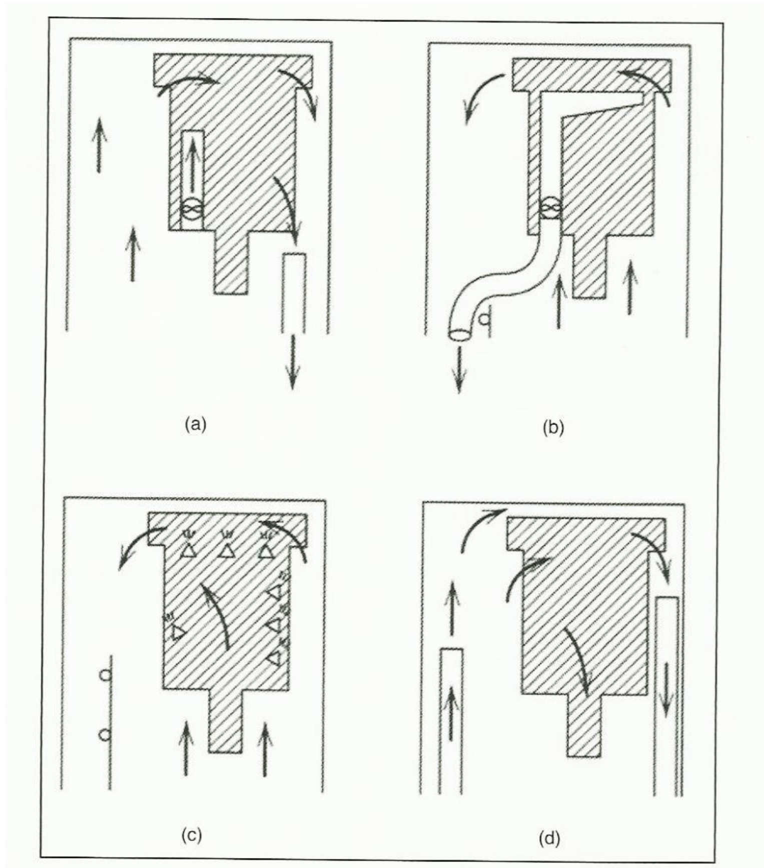


Figure 29. Different schemes/configurations of using ventilation devices for ventilating the face. (Bise, 2013, p. 138)

The most popular continuous miner-based scrubber-fan setup is shown in the schematic in Figure 30. It shows the layout of the scrubber on the continuous mining machine. The exhaust from the scrubber fan goes out the back end of the unit and is directed to an exhaust line brattice. This particular configuration has a dual inlet for better collection of the respirable dust. The individual components of the scrubber are shown in Figure 31.

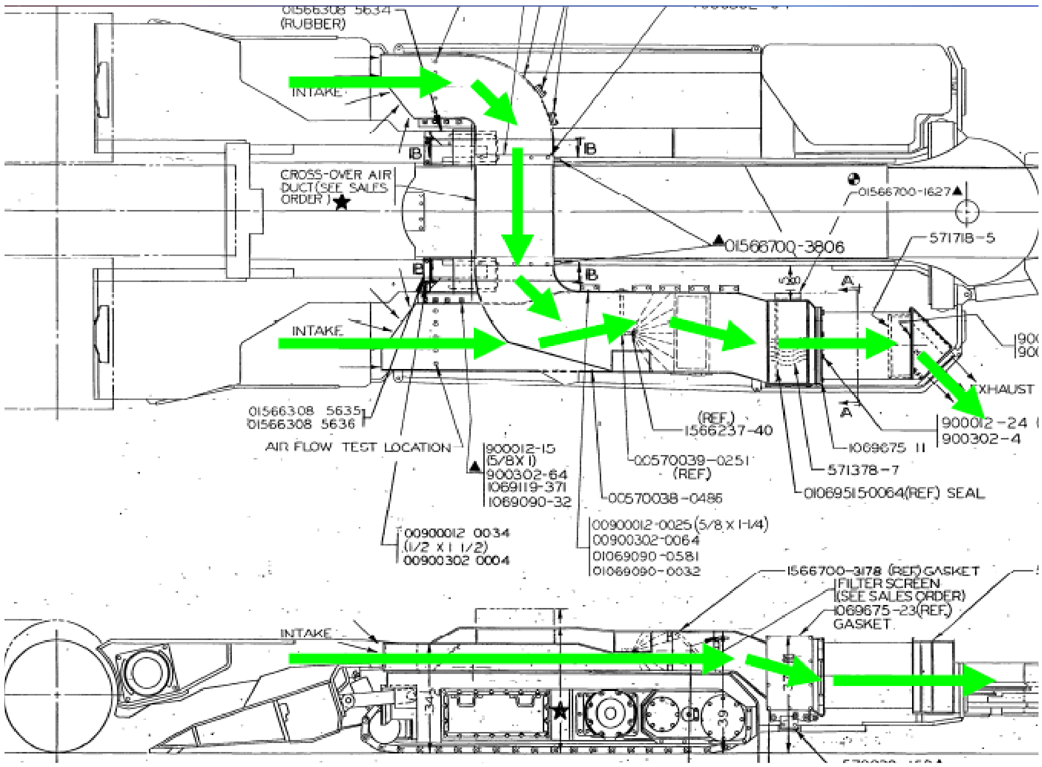


Figure 30. Continuous miner with flooded-bed scrubber. (Joy Global Inc., 2015)

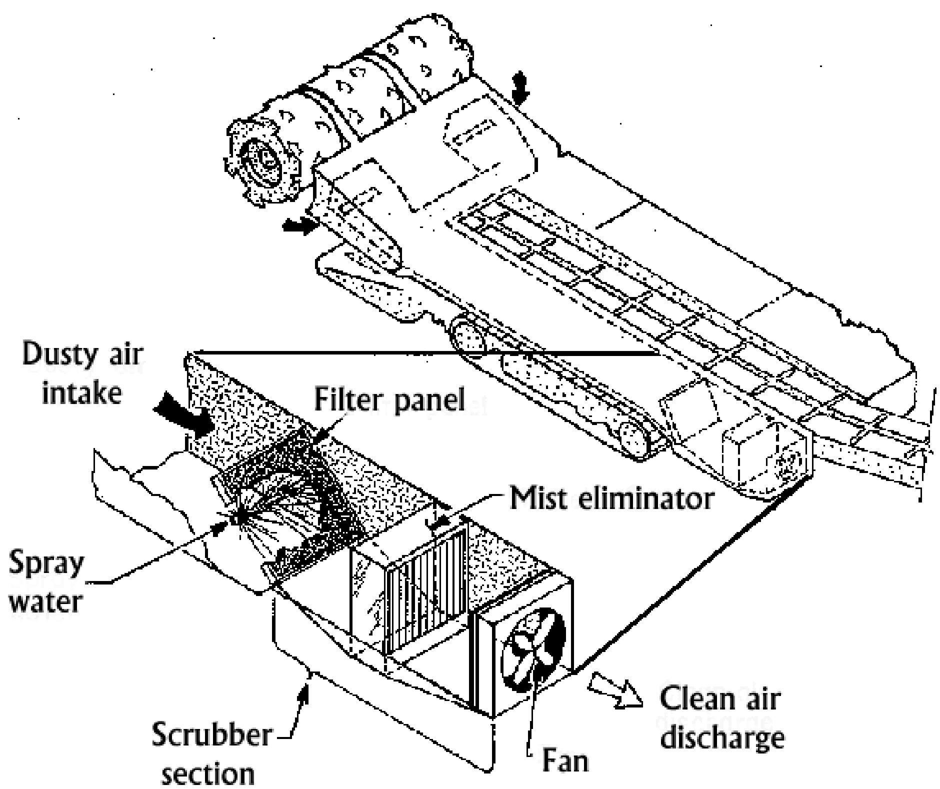


Figure 31. Components of a flooded-bed dust scrubber. (Colinet, 1997)

D. EQUIPMENT

The equipment needed to perform the unit operations are discussed next for the continuous mining section. The equipment includes one or two continuous mining machines, depending on whether or not a super section is used, face haulage equipment (at least two, possibly three or four for a super section), one or two roof bolting machines (two possibly for a super section), and a feeder-breaker connected at the tailpiece of the belt conveyor.

Continuous Mining Machines

The cutting of coal from the face is done by the continuous mining machine (CM for short). The specifications range for dimensions and capacity of the CM to match the seam height, desired cutting height, and/or the purpose for development mining. The CM operator runs the machine using a remote controller. The operator must be mindful of his position relative to the machine when operating it. A regulation has been finalized recently to assist the operator, via an on-board proximity detection system, in knowing when he is not in a safe place relative to the machine. A picture of one model of a Joy Global Inc. CM is shown in Figure 32. A Joy Global Inc. CM is also shown in an 'action' picture in Figure 33. Details on specifications will be given later on several models.

Figure 32. A Joy Global Inc. continuous mining machine. (Joy Global, Inc., 2015)

The production related specifications for five Joy Global Inc. CM models are given in Figure 34. The seam height, maximum tons (per minute), and maximum horsepower capability are important in selecting a model. Brochures, available from Joy Global Inc., have further dimensional and operating information/data for selection purposes. An example diagram with terminology for a CM is given in Figure 35.

Figure 33. A Joy Global Inc. action picture of a CM. (Grayson, 2015)

Model	Tons/Tonnes Minimum/Maximum	Horsepower/kW Maximum	Seam Heights Feet / Metres
12HM	35/21	893/661	4.6/1.4 – 19.7/6.0
12CM	32/29	792/594	4.6/1.4 – 15.4/4.7
14CM	31/28	870/644	2.3/0.7 – 10.2/3.1
17CM	21/19	556/416	2.3/0.7 – 5.2/1.6

Figure 34. Production related specifications for five Joy Global Inc. CM models (Joy Global, Inc., 2015)

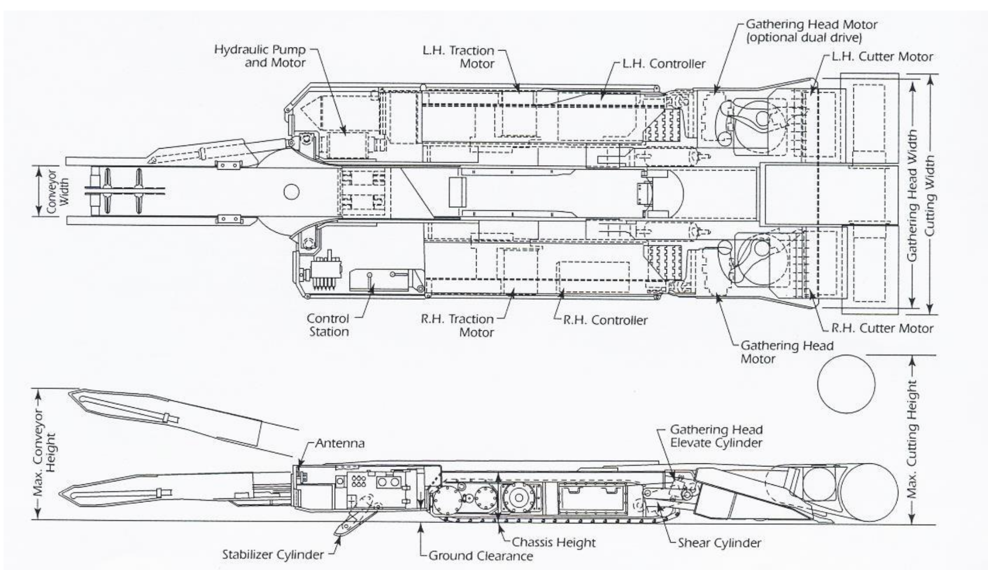


Figure 35. Example diagram with terminology for a CM model. (Joy Global, Inc., 2015)

Face Haulage Equipment

Different options exist for taking coal from the CM conveyor to the feeder-breaker at the tailpiece of the section belt conveyor. Rubber-tired vehicles predominate as the face-to-conveyor transport. These vehicles can be articulated or not; electricity powered through a cable or by batteries; or rarely diesel powered. In thin seams, approximately 54 in or less, continuous haulage systems are sometimes used connecting the CM to the feeder-breaker. Figures 36 and 37 illustrate a Joy Global Inc. shuttle car and the terminology associated with it, respectively.

Figure 36. One model of shuttle car manufactured by Joy Global, Inc. (2015)

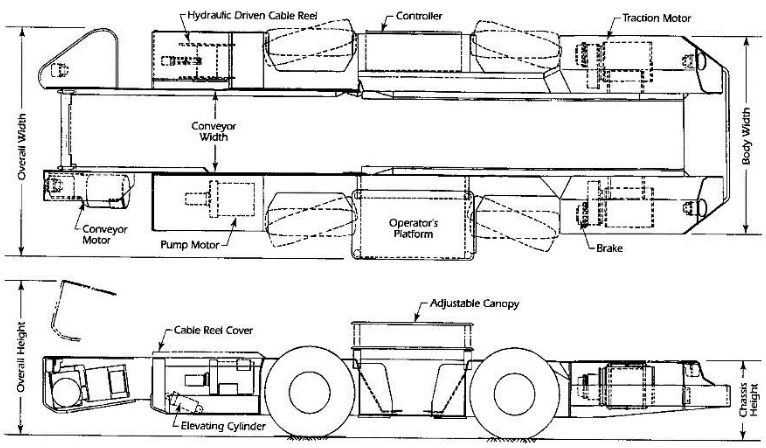


Figure 37. Terminology used for shuttle car by Joy Global Inc. (2015).

Two articulated battery-powered models haulage vehicles are shown in Figure 38.

Figure 38. Two articulated battery-powered models of face haulage vehicles (Grayson, 2014).

Two types of continuous haulage systems are shown in Figures 39 and 40. An associated cutting sequence is shown in Figure 41. Again, this type of face haulage system in mostly with in coal seams of 54 in or less.

Figure 39. One type of continuous haulage system with a flexible belt. (Joy Global, Inc., 2015)

Figure 40. A second type of continuous haulage system with a chain conveyor. (Joy Global, Inc., 2015)

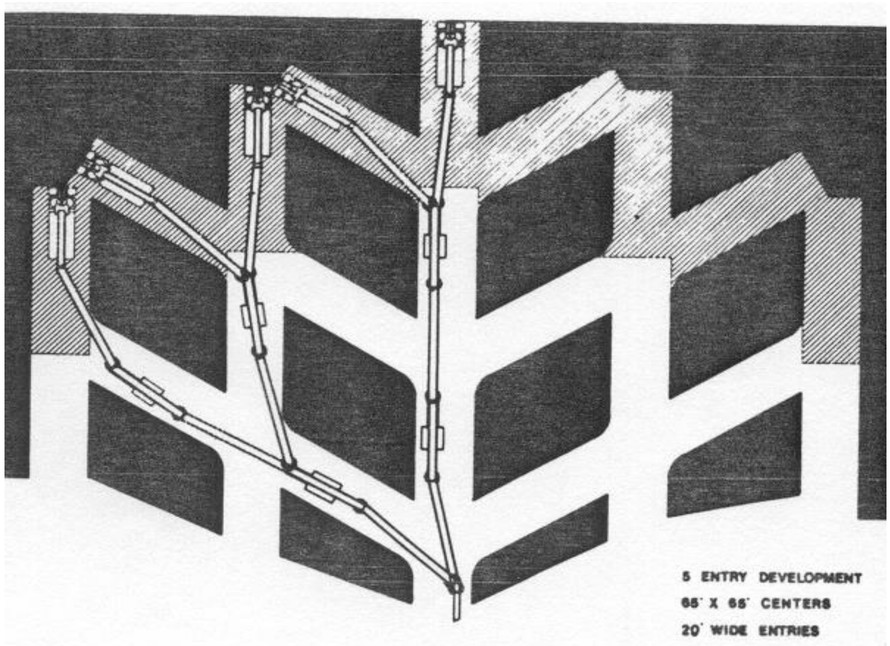


Figure 41. A cutting sequence used for a continuous haulage system. (Grayson, 2014)

Pillar Extraction Support Equipment

Once development mining is completed in a continuous mining section, often pillar extraction is done by some companies. This is called retreat mining. To make retreat mining safer, powered, remote-controlled roof supports were developed, which resemble a chock from past longwall mining days. An illustration of mobile roof supports is given in Figure 42 (J.H. Fletcher & Company, 2015). These supports replaced the use of posts (as breaker posts) and cribs, and now miners can stay away from locations near where the induced falls occur.

Figure 42. Illustration of mobile roof supports being used for retreat mining.

Side and front pictures of a pre-implementation mobile roof support are given in Figure 43. (J.H. Fletcher, 2015)

Figure 43. Side and front pictures of a pre-implementation mobile roof support.

Different cut sequences can be used for retreat mining, when approved as part of the roof and rib control plan. One example is shown in Figure 44 where mobile roof supports are used.

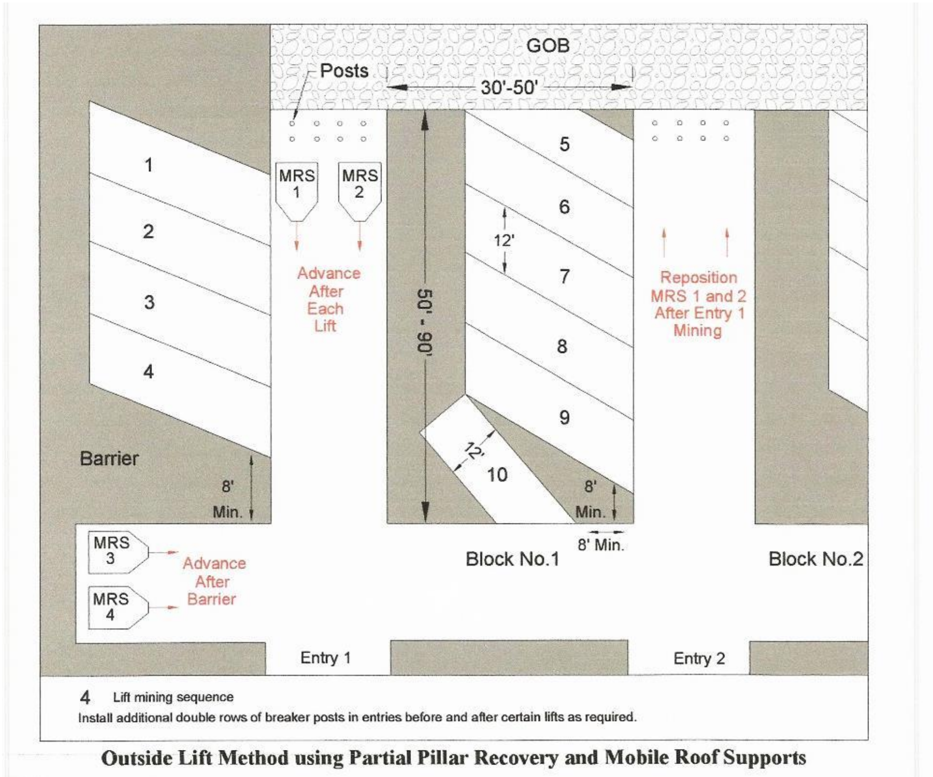


Figure 44. An example cut sequence for retreat mining when using mobile roof supports. (Marshall Miller & Associates, Inc., 2006)

Roof Bolting Machines

In order to accomplish roof and rib support safely without out exposure to unsupported roof, modern roof bolting machines are equipped with a temporary roof support system. A single-beam or double-beam (H-beam) type can be purchased. Canopies are also provided at the operator positions. One other innovation is the walk-through (the middle of the machine) type of roof bolting machine, where the bolters work from the middle of the machine rather than at outside locations. This avoids their constant exposure to coal ribs, which can sometimes be a hazard.

An example of a dual-boom, H-beam type roof bolting machine is shown in Figure 45, in action at the working face. A different type with one beam is shown in Figure 46. It is important to remember than the operators of the roof bolting machine must for the approved plan for installation of bolts, and if unanticipated adverse conditions occur, then adequate supplemental support must be installed at the roof and rib locations where the problem exists.

Figure 45. A picture of a dual-boom, H-beam type roof bolting machine (J.H. Fletcher & Company, 2015).

Figure 46. A picture of a single-beam roof bolting machine (J.H. Fletcher & Company, 2015).

Feeder-Breaker

Coal that is mined and loaded into face haulage equipment may have a size range with an upper limit above 3 in. In order to reduce the number of oversize pieces, a breaker is included with the feeder, which receives the payload from the face haulage vehicle and then feeds the material onto the belt conveyor. The breaker largely ensures that the desired top size for pieces of coal is met, say 3”. A picture of a feeder-breaker is shown in Figure 47.

Figure 47. An example Stamler feeder-breaker (Joy Global Inc., 2015).

E. HAULAGE

Personnel, supplies, equipment, and repair parts must all be taken into and out of the section; different types of vehicles can be used for these purposes. Also, coal must be hauled down the section and dumped on a sub-mains or mains belt conveyor, which is the exclusive way to do it. The different haulage methods are discussed next.

Belt Conveyor

The belt conveyor is designed for the maximum anticipated tonnage, usually maximum tons per a selected time period, e.g., minute, hour, etc. A multi-ply belt carcass is used, often with fiber reinforcement. The conveyor structure may be rigid frame or hanging from the roof; most often continuous mining section conveyor structure hangs from the roof.

The top of the belt is troughed using three-member idlers, which are spaced for good support of the loaded belt (often 4 ft. to 6 ft. spacing for section conveyors). The bottom of the belt is supported by un-troughed return idlers, and because there is no coal loaded on it, the spacing will most often be 8 ft.

The belt width, troughing of the top belt and the speed of the belt determine the designed flow of coal on the conveyor. In a CM section, a 36-in belt width is very common, and a common belt speed range is 200 to 300 ft. per minute (fpm). Figure 48 shows a typical rigid frame belt conveyor setup for a CM section.

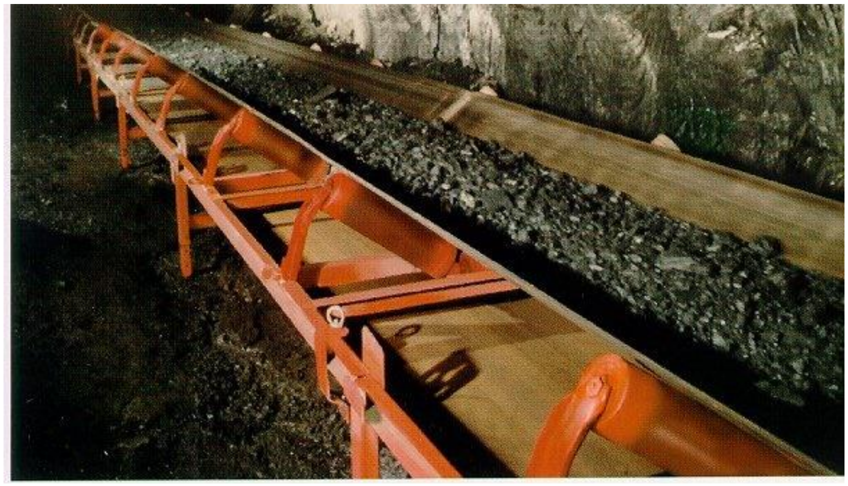


Figure 48. Typical rigid-frame belt conveyor set-ups for a CM section and main line (Grayson, 2014)

Mantrip and Personnel Carriers

Vehicles to transport personnel for various work functions may be track-based or rubber-tired, and may be powered by battery, trolley wire, or diesel fuel. A few examples are given in Figures 49 – 54.

Figure 49. Diesel mantrip/personnel carrier (Brookville, 2015).

Figure 50. Battery personnel carrier (Brookville, 2015).

Figure 51. RM Series personnel car (A.L. Lee, 2015).

Figure 52. Battery inspection jeep (Brookville, 2015).

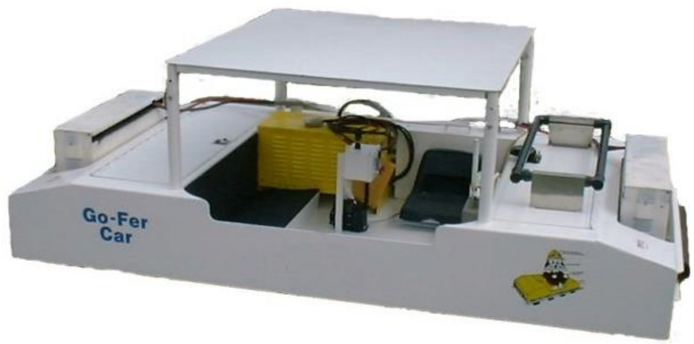


Figure 53. Figure Go-fer Car (A.L. Lee, 2015)

Figure 54. Battery Rail Runner (A.L. Lee, 2015)

Locomotives

Locomotives of various types are used to haul supplies, materials, repair parts, and equipment in and out of the sections and the mine. They, too, may be battery, trolley wire, or diesel powered. Three examples are given in Figures 55 – 57.

Figure 55. Diesel locomotive (Brookville, 2015).

Figure 56. Battery locomotive (Brookville, 2015).

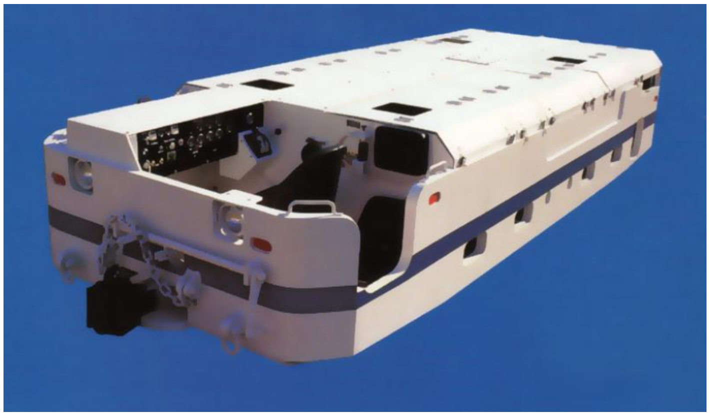


Figure 57. Diesel locomotive (A.L. Lee, 2015).

F. ASSISTANT MINE FOREMAN RESPONSIBILITIES/DUTIES

Operations comprise the bulk of the foreman's shift. Production cycles typically bring together several of his duties and produce overlapping responsibilities/duties that must be managed simultaneously. These responsibilities/duties pertain to the following aspects:

- Operations, including production, maintenance, and downtime;
- Compliance with state and federal mining regulations;
- Monitoring the health and safety of miners, including training; and
- Labor relations, including communications and administrative accountability.

The responsibilities/duties are comprehensively listed below, in chronological order, according to general practice (exceptions do exist for some of them).

Before Shift - Outside

1. Arrive an hour or so early to allow ample time for good coordination.
2. Dress immediately to avoid interference with coordination later.
3. Read over previous shift reports for production, maintenance, examinations, violations, and supply requisitions for the section.
4. Get verbal report on pager/phone or face-to-face from the previous shift's foreman, regarding:
    a. Status of equipment/maintenance work needed.
    b. Equipment locations in the mining cycle.
    c. Status of physical conditions and operational problems.
    d. Work which needs accomplished/finished.
    e. Existing or potential compliance problems.
    f. Employees staying in the section between shifts for any reason.
    g. Sign mine examination book with report for the section, if status was obtained by phone/pager.
5. Monitor check-in of personnel on crew.
6. Make a request for fill-in or extra personnel.
7. Convey status of section to shift foreman; coordinate on-shift requirements/needs.
8. Ensure crew members are properly equipped and ready for duty.
9. Communicate conditions on section to crew members.
10. Coordinate at mine map with previous shift foreman, shift foreman, mine foreman, and superintendent. Discuss important matters.
11. Notify fill-in/extra personnel of job assignment and whether special equipment is needed.

After Reaching Section and Before Operations Begin

1. Ensure that entire crew enters the mine as scheduled.
2. Ensure mantrip is checked for all safety and operational requirements, as

mandated by regulations and practice.

3. Check safety of crew on mantrip: no body parts outside vehicle; operator(s) wear glasses.
4. Start traveling inby to section, calling dispatcher as required.
5. Ensure travel at a safe speed, at a proper distance between trips, and in full control relative to conditions.
6. Note unsafe road, roof, rib, support conditions, if observed, and report to shift foreman and/or dispatcher.
7. If traveling on track haulage, keep switches in proper position, according to practice.
8. De-board in a proper and safely guarded place.
9. Have mantrip parked at designated location.
10. Ensure walkway is well kept and safe.
11. Give safety talk, as required.
12. Check roadways and intersections on the way to the first face for roof, rib, and floor conditions, adequate rock dusting, clean-up, and dustiness.
13. Examine all working faces and write date, time and initials at each; look for:
    a. De-energized equipment and location.
    b. Air quantity at inby end of brattice/tubing.
    c. Condition of ventilation checks, brattice/tubing.
    d. Methane, roof/rib/floor conditions.
    e. Safety and compliance.
    f. Note conditions that need correcting.
    g. Supplies on equipment.
    h. Sights.
    i. Clean-up and dustiness.
    j. Necessary cycle moves (fan, if used; cables; and stoppings needed)
14. Given instructions to crew members, as needed; note revisions to initial plan and conditions requiring caution in correcting them.
15. Determine status of equipment, cables, etc. from crew members who checked them in a general way.
16. Set power on equipment if ready and operable; begin repair of equipment that is not operable and coordinate for additional mechanics/electricians, if needed.
17. Coordinate dumping of supplies which were loaded during face run; ensure adequate amounts were loaded; load any necessary unplanned supplies for use; check servicing of equipment.
18. Report any unplanned occurrences to shift foreman/dispatcher.
19. Record the nature and duration of unplanned occurrences (delays).
20. Give task training, as needed, description of escapeways and firefighting duties to fill-in and/or extra workers. Ensure workers follow safe performance of tasks

according to Job Task/Safety Analyses.

On Shift

1. Production Related
    a. Monitor initial move to first cut; ensure sight line is in place and used; ensure extra brattice/tubing is available, as necessary, along with spads, wire, tools; ensure face haulage vehicle operators use correct haul roads (closest change-out point) and have enough cable (if cable exists) for entire cut. Ensure workers are coordinated on plan.
    b. Monitor loading of first couple of cars to ensure smooth start and note any problems (physical, operational, or mechanical).
    c. Report production start time to dispatcher, also state probability for loading well and any anticipated problems.
    d. Return to cut before time for place change and coordinate move to next cut. Note any changed conditions, if any, and give warnings to workers who will enter places, as necessary.
    e. Plan ahead for cable moves, fan moves (if used), keeping up with sight lines, clean-ups, rock dusting, etc. Coordinate as necessary.
    f. Report any problems or needs immediately to dispatcher and shift foreman.
    g. Repeat steps a – g for each cut, as necessary.
    h. Keep track of all delays (downtime) to production immediately and accurately for use on shift report.
2. Health and Safety Related
    a. Discreetly monitor crew members in the performance of their jobs and ensure safe work practices are used faithfully. Follow up with reinstruction, as necessary.
    b. Ensure workers use proper attire, with no loose clothing, and safety equipment/tools for conditions encountered.
    c. Ensure workers use proper tools and equipment for the job and also properly operate them.
    d. Observe workers for inconsistent performance and possible drug or alcohol effects. Counsel privately with discretion, as necessary. Note that personal problems may also affect performance.
    e. Use caution and give warnings in a new or rate situation; observe closely for hazards and anticipate supplemental actions that may be needed to combat the hazards, as approved in related plans.
    f. Ensure employees report all accidents, equipment damage, and close calls. Report them in turn to the shift foreman at an appropriate time.

    g. Ensure that an injured employee goes out of the mine if serious injury is possible. Call dispatcher to arrange transportation.

3. Labor Relations Related
    a. Maintain control of operations while being firm and consistent in handling workers, when necessary.
    b. Communicate with workers and other managers; inform workers of actions and decisions; coordinate with workers and managers for fulfilling needs of section.
    c. Don't participate in spreading gossip.
    d. Respect abilities of workers and solicit their input in decision-making for unusual or rare situations.
    e. Talk over incipient grievances privately; sort out details and try to settle in accordance with company policies/contract. Talk over personal problems with workers who seek help.
    f. Ensure accurate accounting of overtime in pay checks for workers.
    g. Keep track of unexcused absences of crew members, caution them privately and tactfully regarding indiscretions; remind them of potential disciplinary provisions, when necessary. Check on leave/vacation days when asked.
    h. Keep alert and energetic. This mental conditioning 'rubs off' on crew members.
    i. Equitably assign extra and downtime tasks to workers.
    j. Make only promises you know you can keep, follow through promptly and accurately on them; be fair to all crew members.
4. Compliance Related
    a. Ventilation/Escapeway Provisions – Daily
        i. 20-minute methane examinations at working faces.
        ii. 2-hour examinations at faces.
        iii. Air readings at faces and returns.
        iv. Miner examiner report on section following examinations.
        v. Ensure escapeway signs and directional devices are in place.
        vi. Ensure adequate rock dust maintained in escapeways and rest of section.
        vii. Ensure escapeway map is up to date on section.
        viii. Ensure dust control methods are maintained: sprays on equipment; dust collectors; roadways; washed-down equipment; belt tailpiece and feeder.
        ix. Keep water pumped down on section.
        x. Maintain checks on section.
    b. Roof/Rib Control Provisions - Daily

i. Ensure bolting pattern is maintained within tolerance.
ii. Use supplemental support as conditions warrant.
iii. Ensure required torqueing checks of bolts.
iv. Ensure Automated Temporary Roof Support on bolter makes contact with roof.
v. Ensure any necessary posting and/or cribbing patterns are maintained according to plan.
vi. Use greater bearing surface (headers, larger plates, straps, etc.) when needed, according to roof/rib control plan.
vii. Maintain sight lines and cutting widths to ensure consistent pillar and entry widths.
viii. Use rib support where needed.
ix. Check roof rock by drilling one foot longer hole each cut.
x. Ensure maximum cutting width is not exceeded.
xi. Ensure sum diagonal distances at an intersection is less than maximum allowable distance.
xii. Report unintentional roof falls for investigation and mapping; rehabilitate according to posted plan.
xiii. Discard immediately resin tubes which exceed shelf life.
xiv. Follow intersperse pattern for change from resin to conventional or other bolting, or vice versa.
xv. Do not exceed approved maximum cut depth.
xvi. Ensure proper tools available (sounding device, torque wrench, scaling bar, etc.).
xvii. Follow cut sequence within tolerance.

c. Electrical/Permissibility Provisions – Daily
    i. Ensure equipment plugs and receptacles at power center are marked and matched.
    ii. Ensure restraining clamps or chains are on trailing cables.
    iii. Ensure warning signs for high voltage are posted.
    iv. Ensure necessary rubber mats are in place at power center breakers.
    v. Ensure permissibility is maintained for equipment, pumps, heaters, etc.
    vi. Ensure temporary splices, where allowed, are changed within 24 hours.
    vii. Ensure no splices are made in cables within 50 feet of entry into continuous mining machine.
    viii. Ensure all bonds are intact on rails along track (rail to rail, cross bonds each 200 feet, switches).

ix. Watch for arcing between machines.
x. Ensure cables are hung and guarded where needed.
xi. Keep equipment clean.
xii. Maintain fire extinguishers and rock dust (240 pounds) at permanent installations.
xiii. Ensure equipment is safe to run (guards, etc.).
xiv. Ensure end of trolley wire is secured and guarded.
xv. Ensure trolley wire is guarded at man doors and mantrip unloading stations.
xvi. Use insulated hangers for hanging cables.

d. Pre-Shift Examination Provisions – Daily
   i. Ensure that the pre-shift examination for the next shift is done in accordance with regulations. Report results outside and countersign the examination when outside.

e. Miscellaneous Provisions – Daily
   i. Ensure workers are fully task trained on equipment that they are asked to operate, and it is documented.
   ii. Ensure accumulations of coal/dust are cleaned up.
   iii. Ensure two sources of communications are working properly.
   iv. Ensure potable water is available on section.
   v. Ensure required first aid supplies are available.
   vi. Ensure required fire-fighting equipment, emergency response equipment and devices, refuge chamber are available as approved.
   vii. Ensure required self-contained self-rescuers are available as designated in plan.
   viii. Maintain clean and unobstructed walkways.
   ix. Ensure any necessary manholes (shelters) are cut and cleaned.

f. Weekly Provisions
   i. Make necessary bleeder station and air monitoring point examinations and enter into examination book.
   ii. Ensure weekly permissibility examinations are completed for all equipment on section and properly documented.
   iii. Ensure methane monitor calibration is completed and documented.
   iv. Conduct weekly safety meeting, if appropriate (more frequently as required).

g. Monthly Provisions
   i. Ensure smoking article check is made and documented.

h. Quarterly Provisions
   i. Ensure fire drills are done as approved and documented.
   ii. Ensure walking of escapeways is done as approved and

documented.

i. Annual/Semi-Annual Provisions
    i. Ensure 8-hour retraining is done for crew members.
    ii. Report any ideas for revisions to roof control or ventilation plans.

End-Of-Shift Duties

1. Stop cutting in sufficient time to leave section in full compliance with regulations.
2. Leave section in good condition for next foreman; report all problems fully and accurately (equipment down, poor conditions, etc.).
3. Do not leave section early.
4. Ride out in mantrip as you rode in – orderly and controlled.
5. Ensure no horseplay occurs while waiting for elevator.

Post-Shift Duties

1. Make shift report professionally and completely.
2. Coordinate with in-going section foreman, shift foreman, mine foreman, and superintendent at mine map; report workers left in the mine, labor relations problems and operational problems; mark section map up to date; note near-term section needs such as power center move, track advance, planking, stoppings, belt move, etc.
3. Complete pay sheet accurately, including overtime hours, and save it or place in proper distribution box.
4. Coordinate with the maintenance department regarding any mechanical problems, even if small, even if previously reported.
5. Write report in and sign mine examiner book and assistant mine foreman book; counter-sign if report was previously entered in mine examiner book.
6. Complete and sign additional report books as required (smoking articles, safety talks, bolt torque, bleeder and monitoring points, escapeway travel, fire drill, weekly exam results, etc.).
7. Perform any follow-up promised to crew members.

2. LONGWALL MINING

Once the gate roads of a planned longwall panel (section) are developed, the next longwall setup is begun. The setup entries are cleaned, trimmed as necessary, and well supported before work is begun on setting up equipment, cables, hydraulics, etc. on the future face. Before the current longwall panel is finished, much work is done in placing a new pan conveyor line, including adding the cables and hydraulics. In some cases, many of the shields are set up as well, usually about 20% of them, which were rebuilt when the panel currently be mined was being set up.

After a panel is finished, the major longwall move begins, which is an arduous and complicated task. Good safe work practices and constant awareness of potentially dangerous conditions and situations must be exercised by each worker. When mining in the current longwall panel is complete, then tear-down of the equipment, cables, hydraulics, etc. is done. Usually about 80% to 100% of the shields must then be moved to the new longwall face. The shearer is usually sent out for rebuild, and a replacement shearer is placed on the new panel. Once all of the face equipment is set up, then the headgate equipment is placed, and troubleshooting of the electric and hydraulic systems is done to ensure proper functioning. Any support required to maintain headgate and tailgate stability is added, in compliance with the approved roof and ground control plan. Figure 58 shows a longwall face that is completely set up.

Figure 58. Picture of a completed longwall panel setup (Mining World, 2008).

Basic parameters of the longwall panel and face and its equipment are given in an annual longwall survey conducted by Coal Age magazine. Each year details on approximately fifty coal faces across the country are given. As an example, summary statistics on features are listed next for the 2012 longwall survey (Coal Age, 2013).

- Face widths ranged from 650 ft. to 1,580 ft., with an average of approximately 1,050 ft.; panels continue to get wider.
- Panel lengths ranged from 2,400 ft. to 21,500 ft., with an average of

approximately 12,000 ft.; they continue to get longer.

- The vast majority of the gate road entries (during development) number three, while six panels have four entries (in high methane regions of Alabama, Virginia and West Virginia) and four panels have two entries (for ground control purposes in western mines).
- Overburden thickness ranged from 300 ft. to 3000 ft.
- Seam thickness ranged from 45 in (only plow operation) to 240 in (in the west).
- The actual cutting height ranged from 48 in to 156 in.
- The depth of cut (web) for the shearer ranged from 28 in to 42 in, with an average of 38 in. (The plow's cutting depth was 7 in.)
- Only two-leg shields were used in all panels, and yield pressure settings for legs ranged from 700 tons to 1,300 tons, with eleven of them set at 1,200 tons or higher.
- Face conveyor widths ranged from 860 mm to 1,342 mm, with the most popular being 988 mm and 1,000 mm. The operating speed of the pan conveyor ranged from 229 fpm to 450 fpm, with the most popular being 371 fpm.
- Only twin-inboard chains are used in the pan conveyors, with diameters ranging from 34 mm to 50 mm. The most popular diameters were 42 mm and 48 mm.
- The number motors used to power the pan conveyor were predominately three, with installed horsepower ratings ranging from 1,200 hp to 5,700 hp. The most popular choice was three motors with 1,900-hp ratings each.
- Jumbotrac and Ultratrac haulage racks were the most popular.
- CAT stage loaders were most popular. Stage loader widths ranged from 1,000 mm to 1,732 mm, while stage loader speeds ranged from 297 fpm to 580 fpm. All stage loaders were equipped with a crusher.
- Service Machine and Line Power were the most popular choice for electric controls.
- The most predominately face voltage was 4,160 VAC, with seven panels opting for 2,300 VAC.
- Installed horsepower on the shearer ranged from 1,290 hp to 2,805 hp, with 1,500 to 2,000 being the most popular.

Once the setup of the face and troubleshooting is complete, unit operations can begin. Likely, a few more bugs will need to be fixed in the first week or two of operations, but then standard operations will ensue, at least until changing conditions affect them. Unit operations will be described next.

A. UNIT OPERATIONS

The mining cycle is relatively simple, as compared to the continuous mining cycle, but many important considerations are necessary to ensure the cycle continues to operate efficiently and as designed. The shearer operator uses a remote-control device to ‘drive’ the shearer to follow a cutting sequence. There are multiple ways to cut the face, but, except under non-compliance conditions, bi-directional cutting is usually done on the longwall face in Pennsylvania. That is, cutting occurs from the headgate to the tailgate and also from the tailgate to the headgate.

The bi-directional mining sequence is shown in the following Figure 59. An 18-step sequence is shown in Appendix B (pp. 604-605) of the cited source (Peng, 2006). Once the shearer cuts into the headgate (end of the tailgate to headgate ‘pass’), next the headgate area is cleaned up of loose coal. Once this is done the cutting sequence starts over again. At first, the pan conveyor or headgate operator coordinates the staggered movement of the face conveyor toward the face (called ‘snaking’) while the shearer operator cuts the shearer gradually into the face until a full drum width (web) is attained. Following this step, the shearer will be operated to cut back to the headgate to ‘pick up’ the uncut portion of coal not at full drum width. Approximately, a distance of 15 to 13 shields (75 ft. to 78 ft., depending on the width of the shield) is cut in this process. The shearer then cleans it way back to the full drum depth and cuts all the way to the tailgate. A similar process is used to complete the tailgate to headgate pass.

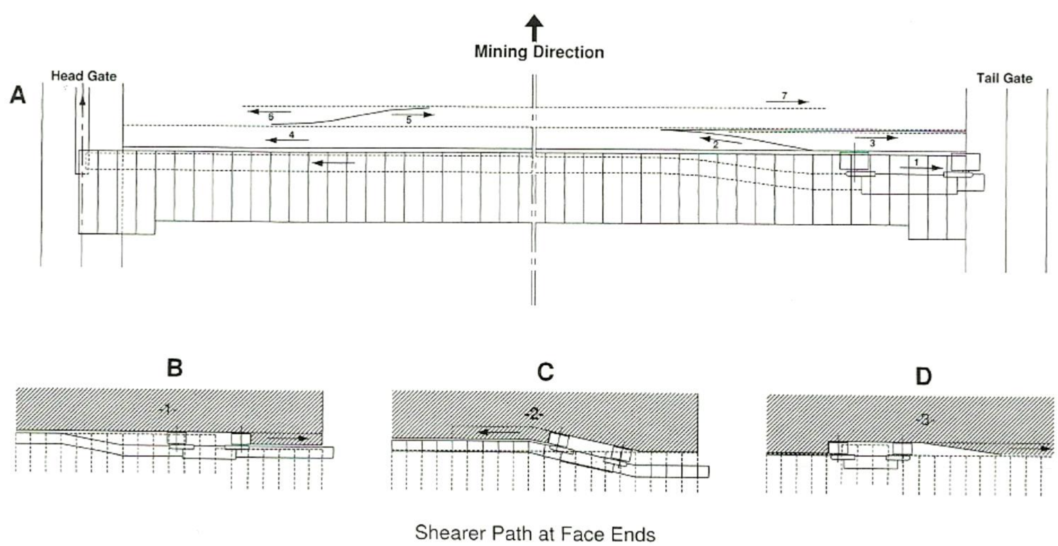


Figure 59. Cutting sequence for bi-directional mining on the longwall face (Peng, 2006, p. 293).

Considerations are necessary for maintaining relatively smooth operations without significant interruptions. A list of them follows:

1. Alignment along the longwall face must be maintained; when poor alignment occurs because of failure to follow the sight line, then high stress points can be developed along the face and cause areas of potentially significant roof failure.

2. Areas of significant change in the geology of the coal, roof, and floor on the face and in the gate entries must be identified as quickly as possible and additional ground control and other measures should be taken to minimize the impact on operations and safety. In some conditions, additional measures may not help much, such as when the coal seam pinches out because of a previously unidentified sandstone washout, in which case more drastic interruption of mining will occur.

3. When elevated methane levels above the legal limit are found at the federally/state approved ventilation monitoring points (face, air monitoring stations, etc.) and/or by the shearer-mounted methane detector, then immediate operations may need to be ceased and ventilation adjusted to dilute, render harmless and carry away the excess methane.

4. An appropriate regular, periodic schedule should be used to inspect and change water sprays and cutting bits on the shearer in order to effectively keep dust generated during cutting isolated according to plan and in compliance with regulations, and to minimize the development of ‘hot spots’ on bits (flat areas on the bit body or shank).

5. Federal/state mandated air readings need to be taken as specified in regulations.

6. Maintenance-related items should be checked regularly, such as the hydraulic pressure on shields, water pressure on the shearer, proper functioning of the communications and shutdown system, fire suppression systems, etc. Weekly permissibility checks on equipment should be done and recorded. Noticeable deterioration of machinery component should be identified and replaced before a major breakdown occurs, and scheduled to best prevent major delay to operations.

7. Federal/state mandated on-section fire-fighting items at electrical equipment, as required, should be checked.

8. Mandated first aid supplies should be checked regularly.

9. Emergency escape devices and marking of the escapeways should be checked regularly, including the on-section mine map for currency.

10. Crews should participate in the quarterly fire drills and walk the escapeways, as mandated.

11. Periodic accounting should be done for necessary supplies and materials for operations, and those needed should be ordered for delivery.

B. GROUND CONTROL

Because of the use of two-leg shields, longwall mining does provide excellent support for miners working on the face, unless drastic geologic conditions are encountered. Generally, no roof bolting is required on the face until the end of the panel. Longwall mining is able to mine economically at greater depths than continuous mining, which would require tremendous-sized pillars and narrow entries.

Of course longwall equipment is much more expensive than continuous mining equipment and periods for longwall moves, without any production, are arduous and sometimes long (under adverse conditions). Also, longwall mining cannot handle drastically changing geologic conditions (want areas or variable thickness of the coal seam) flexibly, as can continuous mining. Uncharted gas wells cause a serious operational delay. It also cannot be used where massive roof rock is encountered at relatively shallow depth.

One of the worst roof conditions occurs in an underlying seam if there are remnant pillars in a previously mined, overlying seam. Pressures are transferred through these pillars to the underlying seam through a punched stress effect. (see Figure 60). Longwall mining becomes extremely difficult if not virtually impossible under such conditions.

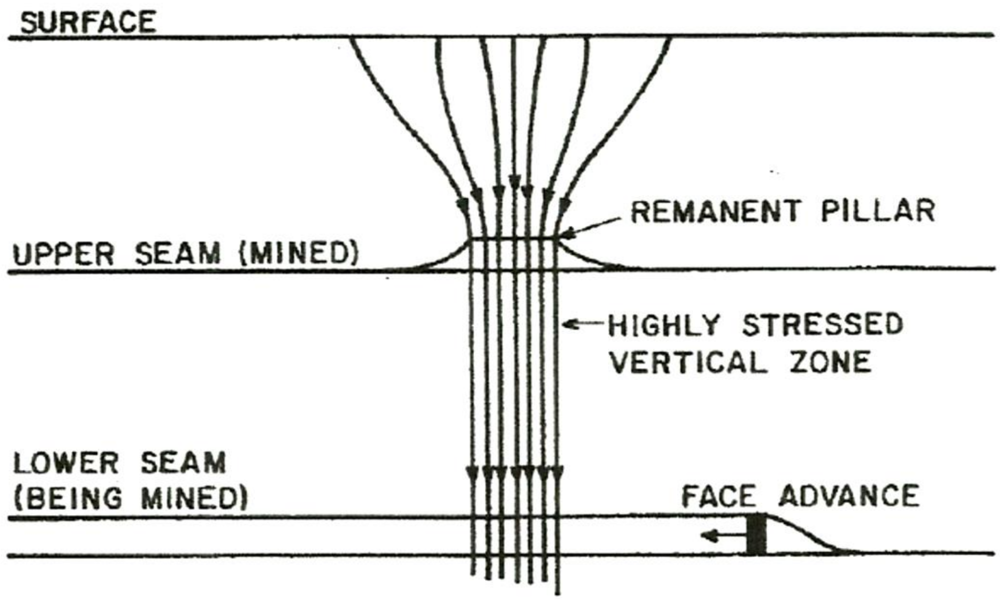


Figure 60. Stress concentration punch effect of remnant pillar in multiple seam mining (Stefanko, 1983).

Roof Pressures in Longwall Mining

The key to successful longwall mining is proper ground control (roof, floor and ribs) through the selection of adequate support systems. If proper ground control is to be achieved, some understanding is required of the influence of natural conditions and of roof pressures (primarily).

The basic requirements for good ground control are a firm bottom and top to resist support penetration. The roof rock must have relatively good caving characteristics. This generally excludes massive strong rock beds which require excessive open areas before the roof rock breaks.

Roof behavior is a function of actions and reactions. Increasing pressures with depth and geological disturbances influence incipient fractures in rock. Because of a lack of understanding of these facts, early longwall installations in the U.S. were failures. A combination of too-shallow seams with massive competent rocks that have not been preconditioned to cave created conditions where the roof hung in the gob instead of caving along the read prop line as desired to relieve the pressures over the props. The result was that the props yielded and went solid, and the rams telescoped completely within their bases. We now know that the props should have been much heavier under the conditions of shallower depths and more massive rocks in order to provide a greater resisting action and force caving at the gob line.

In longwall mining with caving, as the coal seam is removed and the long face advances, the immediate roof sags away from the stronger higher strata. Even though the amount of sag is small, it relieves the immediate roof of all load of the overburden. As the advance of the face continues, the roof span increases until caving occurs (see Figure 61), thereby creating a shelf over the emplaced supports. This cantilevered beam, or shelf, held within limitations and controlled by powered supports properly designed for the conditions, forms a protective work area. As the supports are advanced, a new long straight breaker line is established as the rear of the supports along which the overhanging shelf is sheared and caved.

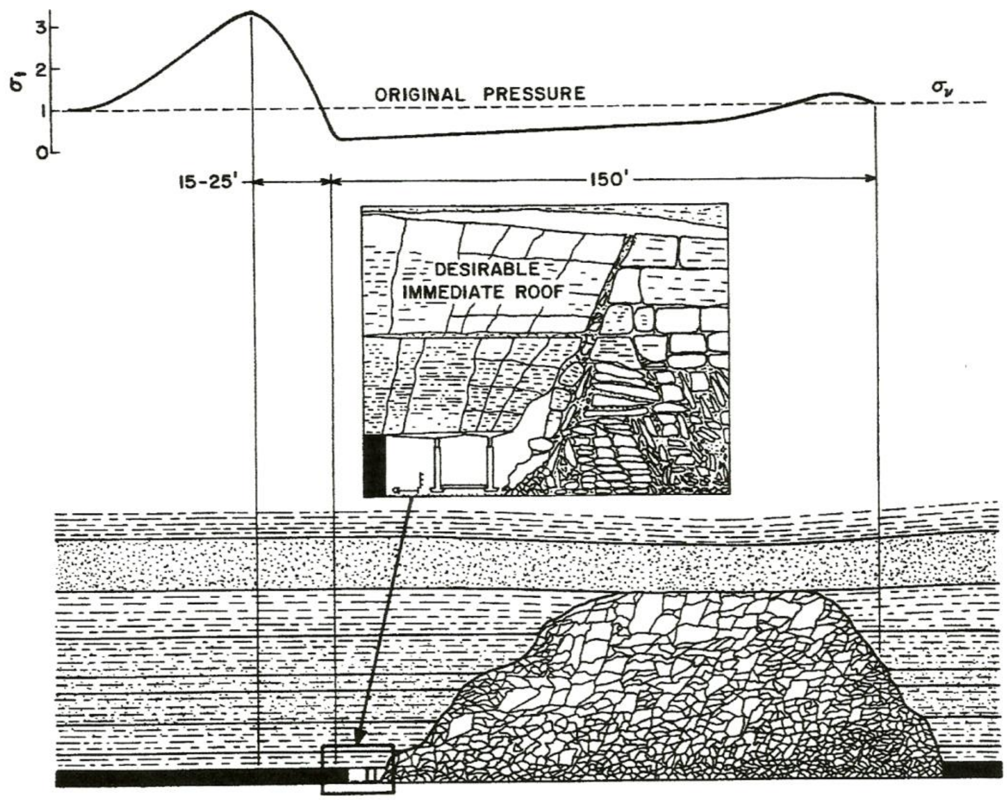


Figure 61. Stress profile on a longwall face section. (Stefanko, 1983)

The caved material expands to fill the void, and the upper roof forms a span between it and the line where the immediate roof separated from it near the face of the coal. Part of the overburden pressure from the spanned area is transferred through the higher strata to the solid area over the advancing face. Superposed on the original stress condition, this face abutment pressure will be high, causing incipient cleavage in weaker strata over the coal when it is at great depth.

The immediate roof, acting as a cantilever beam wedged between the higher strata and the coal, hinges at the edge of the coal face and causes stronger induced cleavage planes to form in the immediate roof. This beam reaction causes the coal along the face to fracture and soften, making it easier to mine.

The cleavage planes which have been formed in the immediate roof assist shearing at the rear edge of the supports. Without this weakening effect, it is more difficult to provide sufficient fulcrum strength at the rear edge to cause a cave (see Figure 62A).

In earlier installations in the U.S., strong roof with limited induced cleavage required stronger waste edge support than provided by jacks imported from Europe. Failure to recognize this fundamental principle doomed these earlier efforts to failure (see Figure 62B).

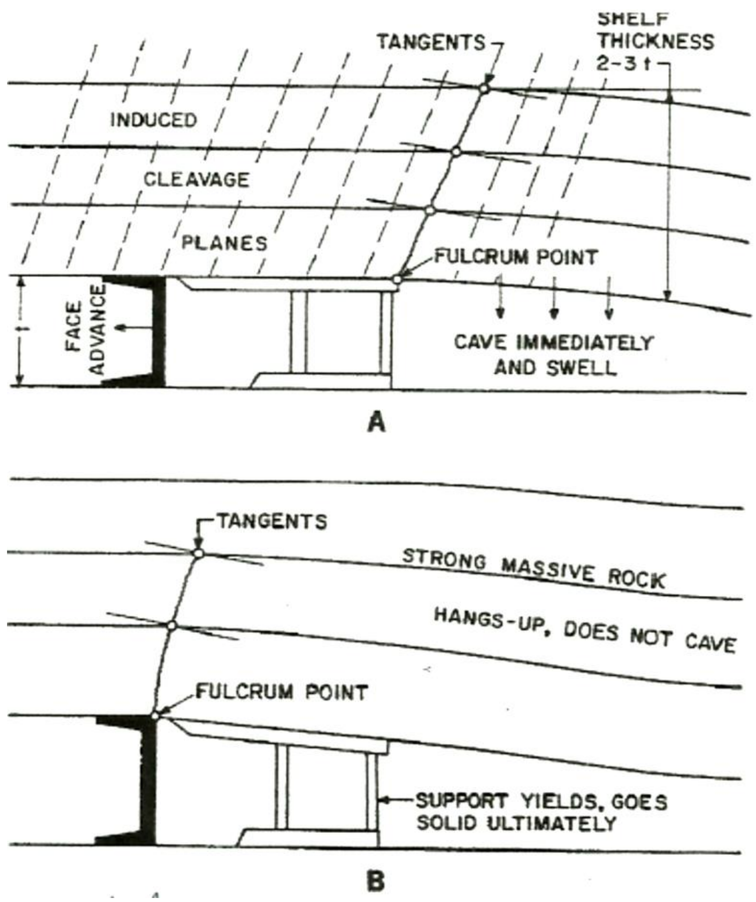


Figure 62. Roof behavior over longwall faces; A: desirable, B: undesirable. (Stefanko, 1983)

The immediate roof or shelf must be of sufficient thickness so that in caving it will expand to fill the void and permit gradual settling of the higher strata. A roof, weakened by excessive fracturing, will produce a greater volume of gob when it caves, so that a shelf thickness of not less than three times the seam thickness will usually suffice. On the other hand, a strong roof, under light cover where there has been little fracturing from the front abutment pressure, will fall en masse and will require a thicker shelf to create the volume of gob needed. Other factors being equal, the lighter the cover or the thicker the seam, the greater must be the shelf depth which conversely requires a heavier support.

In U.S.-style retreat longwall mining, the face is retreated away from barrier or bleeder entries. There usually are unmined chain pillars on both sides (gates) of the gob and the solid coal face itself. In effect, then, as the face is retreated and until caving begins, the mined-out void is covered by the equivalent of a plate. As the weight of the overburden loads the plate, the plate bends on all four sides, transferring the weight of the unsupported strata to pillars or coal on the ribs, and these become abutments.

As the size of the opening increases, the load on the abutments keeps increasing until something gives – the roof, if all goes as planned. The goal is to get a good first fall or cave and then to maintain caving at the waste edge of the supports while preventing the overriding of the supports and the breaking of the roof at the face. Under reasonably normal conditions, supports can be selected with the characteristics necessary to keep the immediate top from breaking over and ahead of them to the face, and at the same time, provide resistance points at which it can break and cave behind them. The severest stress on the roof supports is immediately prior to the first fall. If the supports can withstand this, they normally will resist periodic or intermittent weighting thereafter. With average overburden and no heavy rock members in the upper strata, the face may retreat 200 to 300 ft. before there is much caving. With stronger strata, this distance may increase to 400 to 500 ft. although caving will occur sooner with wider faces. Some spectacular (and dangerous) air blasts and vibrations have resulted from first falls.

Under ideal conditions of caving, most of the roof pressure is taken by the solid coal face and the gob, the function of the supports being to hold the minor immediate roof. While the immediate roof need not be held completely in place, sagging should be limited to prevent breaking along the face. Furthermore, lateral movement of the shelf area toward the gob must also be limited. Advancing the supports immediately after mining is the surest way of achieving this result, assuming there is sufficient basic support capacity to handle the main supporting job.

Quick release and advances, adequate capacity, and a sufficient number of supports are necessary to limit sag, but excessive support pressure should not be used because this can cause lifting of the roof and subsequent fracturing. In addition to interfering with the advancing of supports, loose coal and refuse under the bases of support and roof coal can crush out and destroy the holding ability of the support jacks.

The faster the face is advanced, the less time, as a rule, for troublesome conditions to develop, and, consequently, the better are the working conditions. But with support jacks of sufficient capacity maintained for proper function, mining at a face can be shut down for short periods with minimum deterioration of the top. Depending on conditions, additional support may need to be added if there will be long shutdowns.

Figures 61 and 62A indicate a desirable roof control situation. However, combinations of depth, strength of immediate top, and cleavage planes can cause poor roof control. When the overburden is shallow and does not cause strong shales and sandstones to prefracture in advance of the coal face, the roof breaks in large slabs or hangs out in the gob; the slabs eventually shear off from their own weight, causing the supports to go solid, resulting in a temporary loss of the face. If the condition persists, the remaining panel must be abandoned to another location or new, more massive supports must be installed.

At still greater depths, when massive sandstone comprises the immediate top and cleavage planes are parallel to the face, large blocks may interlock and create severe support pressures as shown in Figure 63, providing another situation in which supports may go solid. It should be apparent now why greater support jack capacity is required with stronger, more massive roof rock at shallower depths and why cleavage planes must be carefully considered in the design phase. Also, it should be apparent that the thicker the seam and thus the taller the support jacks, the more likely they are to upset and more difficulty is encountered in achieving support stability.

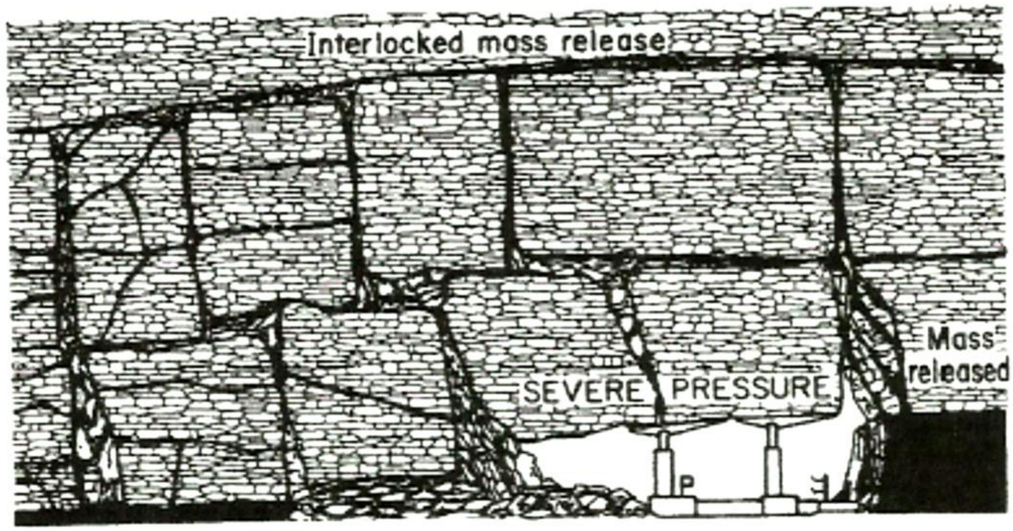


Figure 63. Effect of parallel face cleavage in massive rock at great depth. (Stefanko, 1983)

C. VENTILATION

Most often, except in deeper coal seams with high methane emission levels, three entries are driven in gate roads. In development of the gate road, one entry is an intake, there is a neutral belt entry, and one entry is a return. Once the development entries are completed, then the ventilation must be changed for the newly defined longwall panel.

Most often the primary intake air is taken up the two non-belt entries of the headgate. Check curtains are placed a little inby the longwall face, and the air is coursed across the longwall face. By federal regulation, a minimum of 30,000 cfm is generally required generally at a designated shield number on the face; however, the required quantity will be increased if necessary to control methane and/or dust. At the tailgate end of the face, the air is generally course inby the longwall face and outby the face in the nearest entry. This will generally be adequate to dilute, render harmless, and carry away low to moderate amounts of methane emissions. In cases where high methane levels are emitted, intake air may also be taken up the nearest entry in the tailgate to the end-of-face area and then beyond that point inby.

Air monitoring stations, or points, are established, in coordination with the federal and state agencies, to ensure consistent dilution and control of methane and dust. It is important to ensure that good roof control after mining is maintained, so that open airways persist. Additional supports must be added for this purpose when necessary.

D. EQUIPMENT

There are two major manufacturers of longwall mining systems in the U.S. now: Joy Global, Inc. and Caterpillar (CAT). An example of an outside-the-mine assembled Joy Global system is shown in Figure 64; it includes the shearer, riding on a pan conveyor, the two-leg shields, and the headgate units which interface with the section belt conveyor.

Figure 64. Outside-the-mine assembled Joy Global, Inc. system. (2015)

A picture of an installed Joy Global system at the face is shown in Figure 65, and a picture of an installed CAT system is shown in Figures 66 and 67. They both are state-of-the-art systems with full electronic control.

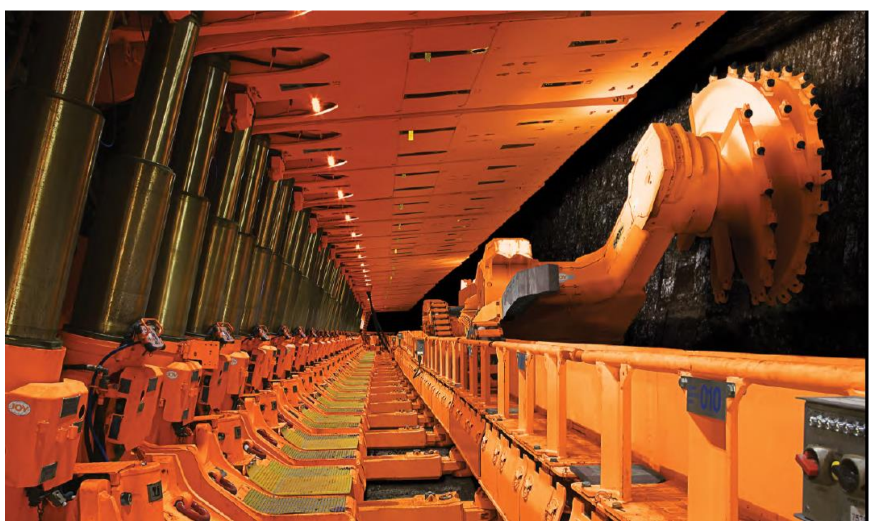


Figure 65. Installed Joy Global, Inc. System. (2015)

Figure 66. Installed CAT longwall system. (Grayson, 2014)

Figure 67. Installed CAT longwall system, showing the shearer. (Grayson, 2014)

Details on systems installed in longwall faces across the U.S. were given above in the 2012 longwall survey as a bulleted summary with general statistics.

E. HAULAGE

Haulage on the longwall face is done by the pan conveyor. The governing factors on productivity, in tons per minute, are the width of the coal-carrying part of the conveyor, along with its heaping capability which gives the cross-sectional area of a pile of coal, and the speed of the conveyor. This can then be translated into tons per hour as the mining cycle is factored in. The tons per hour of the section belt conveyor must be designed to match or exceed the peak output from the pan conveyor. Governing factors on belt conveyor production design are similar, although the structure is different, i.e., cross-sectional area and speed.

F. ASSISTANT MINE FOREMAN RESPONSIBILITIES/DUTIES

The responsibilities and duties of the assistant mine foreman are somewhat different for the longwall crew supervisor, as compared to those for a continuous mining section foreman. In the previous presentation on responsibilities and duties for a CM section foreman, the reader should simply translate multiple continuous mining section faces to a single longwall face, and reflect on the differences in regulations for the different methods, for example, ventilation requirements, ground control requirements, etc.

TABLE OF CONTENTS

MINING METHODS SECTION 4

A. MINE DRAINAGE

1. INTRODUCTION

Drainage in coal mines is extremely variable. In some mines, the drainage is excellent and water causes few problems; in others, lack of drainage results in severe inundations. However, even a small amount of water in low coal can cause minor discomfort and result in reduced productivity. With today's clean water regulations that forbid mine water discharge into streams without extensive prior treatment, coal mine drainage is causing even more problems for mine operators that it did in the past.

There are many possible sources of water that enter underground coal mines (Cummins and Given, 1973; Cassidy, 1973), as follows:

- In shallow mines with minimal overburden, there can be seepage of surface water.
- In deeper mines where caving is practiced and breaks occur to the surface, seepage of surface water can enter through the broken overburden.
- Since streams are thought to form at weakness planes or cracks in the earth, faulty rock occurring under the streams allow water to enter a mine.
- In a deep mine where caving is practiced, the fracture ellipse can break to an aquifer, draining it into the mine.
- Water may collect in exploration and drill holes and be drained when the holes are penetrated during mining.
- Connate (inherent) water occurs in coal seams and also in the bottom or roof and is freed in the process of mining.
- Abandoned mines may be accidentally cut into, freeing water that has collected there.
- Malfunctioning of the water-allaying, dust-suppression systems may create water spillage.
- Flow may occur from natural fractures or joint systems.

There are both direct and indirect problems associated with the influx of water into a coal mine; these have effects in as well as out of the mine. Some of the direct effects of water in a mine might include:

- The blocking of mine entries to the passage of air and haulage by the accumulation of water.
- Interruption of production and damage to the mine, possibly even loss of life due to inrushes o water.
- Increased costs caused by the need to remove the water.

- Interference with haulage; especially, water can cause poor traction for rubber-tired equipment and result in track deterioration as it washes away ballast.

Some of the more indirect effects of water in a mine include the following:

- The water may freeze in shafts and slopes during the winter, severely hampering or causing the suspension of operations.
- It reduces the efficiency of workers.
- It increases the need for maintenance of equipment, especially electrical cables and tires.
- It promotes deterioration of roof rock and reduces the stability of mine rock structures.

Finally, there are indirect effects of water that occur outside the mine, as follows:

- Moisture in the coal product increases shipping, treating and handling costs.
- Effluent from the mine may pollute surface waters.
- Water that drains into the mine can draw down wells in the area and thus lower the water table.
- Excessive lowering of the water table can result in surface subsidence.

To minimize the problems associated with an influx of unwanted water into underground mines, a four-step process of control is suggest: 1) prevention, 2) collection, 3) transportation, and 4) disposal. Each of these steps will be reviewed briefly.

2. PREVENTION

Each gallon of water that is prevented from entering the mine is one less gallon that will have to be collected, transported, and disposed of. While some common sense precautions such as avoiding the siting of shafts, boreholes, and other openings in the low spots on the surface; fluming; and other measures can prevent surface water runoff from entering the mine, preventing water influx from aquifers might be more difficult. Some of the more important ways of preventing water influx into mines include the following:

- Design of excavations to minimize breaking to aquifers or to the surface through appropriate width/depth ratios.
- Proper siting of shafts to avoid low spots and the use of pregrouting during sinking and lining of shafts.
- Providing adequate pillar support under bodies of surface water or underground aquifers, or, alternatively, providing mine design based on proper W/D ratios.

- Concreting all exploration drill holes so that water cannot accumulate in them.
- Keeping test holes properly drilled ahead of active workings when approaching a suspected abandoned mine and not relying just on the accuracy of mine surveys.
- Keeping dust-suppression water-allaying mine water systems in good working order to prevent accidental leaks.
- Initially mining the lowest spot in the mine and working up the seam, using the old workings as a sump.

3. COLLECTION

Even with the best mine design and all sorts of precautionary measures, water influx into mines cannot be entirely eliminated. The water that accumulates has to be collected before it can be transported and disposed of. In the collection process, it is important that gravity be used as much as possible to minimize the power required to transport the water.

In the past, it was common practice to begin mining at the lowest elevation of the seam and work up the pitch. In this way, water would flow away from the faces to the worked out areas which served as gigantic sumps with only excessive overflow being pumped. Drainage ditches were made in the bottom as a conduit for the flow. Today, this practice has been greatly modified. Since the bottom can contain pyrites, water coming in contact with it can be contaminated. Therefore, rather than open ditches, pipe, usually of the PVC variety, will generally be used to collect the water. Furthermore, since gobs will contain pyritic materials, it is usually undesirable to allow water to stand in them where oxidation of pyrites can occur. Thus, there is a much greater tendency than in the past to collect the water and get it out of the mine immediately. In addition, mining must be performed in such a manner that drainage will not occur from openings when mining is terminated.

In the face area, small pick-up pumps will be used periodically to move the water to a nearby small sump for removal from the section.

4. TRANSPORTATION

Even where gravity has been utilized to the greatest possible degree for collecting mine water, some means of transporting the water out of the mine is still required. Today, pumps are used almost exclusively for this purpose. Since pumping mine water is such an important subject, it will be given extensive treatment in a succeeding part of this section.

5. DISPOSAL

With the high effluent standards that must be maintained for streams today, disposal of mine water can be a very costly problem. There must be some knowledge of the material to be to be disposed of if it is to be handled properly.

Oxidation of pyrites in the coal seam and strata overlying and underlying the seam is the initial step in the formation of acid mine water. As oxidation continues, the material disintegrates, exposing new surfaces for further oxidation and acid formation. Thus time is an important factor, because the longer the acid-forming materials are exposed to the atmosphere, the greater the amount of acid that will be formed. While the chemical reactions expressed in the following are probably an oversimplification of this complex process, it begins as follows:

$$FeS_2 \text{ (pyrite)} + H_2O + 7O \rightarrow FeSO_4 + H_2SO_4 \qquad (1)$$

The rate at which this reaction occurs is variable and depends on such factors as the pyrite's properties and composition, temperature, and the pH of the water.

The second step, shown in equation (2), most certainly depends on aeration and temperature but also is considered to be influenced by bacterial oxidation, although this is not well understood.

$$2\ FeSO_4 + O + H_2SO_4 \rightarrow Fe_2\ (SO_4)_3 + H_2O \qquad (2)$$

Finally, the ferric sulfate hydrolyzes, forming more acid and precipitating ferric hydroxide and basic sulfates. The approximate formula for this reaction follows:

$$Fe_2\ (SO_4)_3 + 6\ H_2O \rightarrow 2\ Fe\ (OH)_3 + H_2SO_4 \qquad (3)$$

These final two products create a serious problem in the handling and disposal of mine water. The gelatinous yellow boy precipitate, $Fe(OH)_3$, is especially difficult to dispose of. Consisting of approximately 90% water, it has considerable volume and must eventually be buried in a suitable place. Sulfuric acid, H_2SO_4, is a highly corrosive material and must be neutralized before it can be placed in streams. Thus the principal methods for treating mine water are designed to neutralize acidity and remove iron by processes involving the use of lime, limestone or other basic compound and by demineralization.

Most mine water treatment processes involve lime neutralization. Lime is generally available, it has a high alkalinity, and it is less costly than other bases except limestone and waste material. Complete lime treatment processes include neutralization using hydrated lime, aeration to oxidize the iron from a ferrous to a ferric state, and sludge settling and sludge disposal. Usually a series of ponds on the surface that require considerable space will be used to carry out this process. As a pond is filled with sludge (yellow boy) it must be cleaned out.

Because of lower cost, limestone treatment is often more appealing than the use of hydrated lime. However, unless sufficient oxidation is provided, this type of system is not satisfactory for treating ferrous-iron water, as the oxidation rate is too low. Also the limestone becomes coated with ferric oxide hydrate and calcium sulfate and loses its neutralizing effectiveness. However, this latter problem has been overcome by placing limestone in rotating drums through which mine water is passed. The particle abrasion that occurs during rotation is sufficient to remove the coating from the surface of the stone and permit the carbonate reaction to continue. Treat of an increased volume of water has been accomplished by utilizing the rotating drums as an autogenous wet grinder to produce a 400-mesh limestone slurry that is mixed with the mine discharge.

B. PUMPS AND PUMPING SYSTEMS

1. PRINCIPLES AND DEFINITIONS

Mine drainage systems contain such items as sumps, suction pipes, pumps, discharge pipes, and appropriate fittings such as valves and ells (see Figure 68) to move water from a point in the mine to the surface. Energy is introduced into the system via the pump to overcome the following total dynamic water heads (H):

- Total dynamic suction lift which is comprised of two factors: the vertical distance of the pump above the liquid (static suction lift) and the frictional resistance of the suction pipe.
- Total dynamic discharge head which may comprise three factors: the vertical distance of discharge above the pump (static discharge head), the frictional resistance of the discharge pipe, and the pressure at the end of the discharge line.

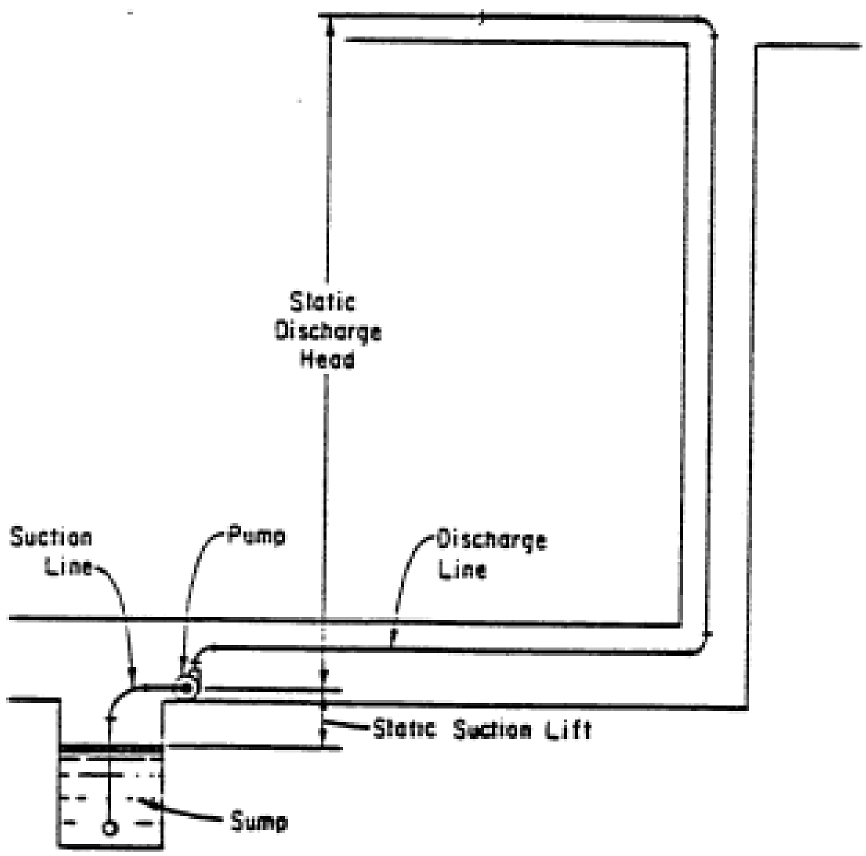


Figure 68. Generalized diagram of a mine drainage system. (Bise, 2003)

The total dynamic head (H) can be expressed mathematically as follows:

$$H = H_s + H_f + H_{sh} + H_v \qquad (4)$$

where H_s is the total static head representing the difference in elevation between the level of the water at the source to the point of discharge. H_f is the frictional head produced by resistance of water flow in the pipes, H_{sh} is shock losses due to changes of water flow produced by fittings, and Hv is the velocity head of the liquid moving at a given velocity in the equivalent head through which it would have to fall to acquire the same velocity. It is expressed by the following formula:

$$H_v = V^2/2g \qquad (5)$$

where H_v is the velocity head in feet, V is the velocity of the water in feet per second, and g is the acceleration due to gravity in feet per sec^2.

Many empirical formulas have been used for calculating the frictional losses in a pipe using the following formula:

$$H_f \propto (f L V^2)/D \qquad (6)$$

where f is the pipe coefficient of friction, L is the length of the pipe, V is the velocity of the water, and D is the diameter of the pipe.

Results have been expressed in convenient tables such as those derived from the work of Williams and Hazen (see Table 1) for old iron and steel pipe. In the table, the equivalent length method for determining shock losses in fittings is shown. For example, in a 6-in pipe, the shock loss in a single 90o ell is equivalent to the frictional loss that would occur in 16 ft. of straight 6-in pipe. Other such tables are available for pipes of different composition, for example PVC pipe.

Finally, pressure is required to accelerate the water from rest to discharge velocity which is generally not recovered. Usually, however, this is so small that it is ignored in the calculations.

Water horsepower (hp) is the theoretical minimum amount of power required to drive the water and is expressed as follows:

$$Hp = (8.33\ Q\ H)/33{,}000 \text{ ft. lb./min} \qquad (7)$$

where Q is the water flow in gallons per min, H is the total dynamic head in feet, and 8.33 is a conversion factor for gallons of water to pounds.

Reducing to a single constant, the formula becomes: $Hp = QH/3960$.

Brake horsepower (bhp) is the amount of power which must be delivered to the input shaft of the pump and is expressed as follows:

$$bhp = QH/(3960\ E) \qquad (8)$$

where E is the efficiency of the pump expressed as a decimal and is a ratio of water horsepower (output) to brake horsepower (input) times 100%.

Table 1. Friction Loss in Feet for Old Steel Pipe, C = 100. (Bise, 2003)

Flow, gpm	Pipe sizes in in. ½	1	2	3	4	5	6	Flow, gpm
2	7.4							2
4	27.0	2.14						4
6	57.0	4.55	0.20					6
8	98.0	7.80	0.33					8
10	147.0	11.70	0.50	0.07				10
15		25.00	1.08	0.15				15
20		42.00	1.82	0.25	0.06			20
30		89.00	3.84	0.54	0.13	0.04		30
40		152.00	6.60	0.91	0.22	0.08		40
50			9.90	1.38	0.34	0.11	0.05	50
60			13.90	1.94	0.47	0.16	0.07	60
80			23.70	3.30	0.81	0.27	0.11	80
100			35.30	4.98	1.23	0.41	0.17	100
150			17.60	10.60	2.61	0.88	0.37	150
200			129.00	18.00	4.43	1.50	0.62	200
250				27.20	6.76	2.27	0.94	250
300				38.00	9.30	3.17	1.30	300
350				50.60	12.50	4.22	1.74	350
400				64.70	16.00	5.40	2.22	400
450				80.50	19.90	6.65	2.76	450
500				97.80	24.10	8.15	3.36	500
600				137.00	33.80	11.70	4.70	600
700					45.00	15.20	6.25	700
800					57.60	19.40	8.00	800
900					71.60	24.20	9.95	900
1000						29.40	12.10	1000
1200						41.10	16.90	1200

Another way to estimate the friction and shock losses for a mine pumping system is to use the well-known Hazen-Williams empirical equation, which follows:

$$H_f = [0.2083\ (100/c)^{1.852} \text{ x } Q^{1.852}]/d^{4.8655}$$

where H_f is the friction loss in ft. per 100 ft. of pipe, c is the Hazen-Williams roughness constant for different materials of pipe, Q is the quantity of flow in gpm, and d is the inside diameter of the pipe. For popularly used PVC pipe, c = 150. The equivalent length of head loss for fittings in a particular segment of PVC pipe can be found in an associated table, for example, a 90^o elbow with a sweep radius has an equivalent length value of 12.0 ft. for a 4" PVC pipe and 7.0 ft. for a 3" PVC pipe, while the values are 13.1 and 11.1, respectively, for a sharp radius 90^o elbow.

Problem. An example problem will be given next on a mine pumping system. Refer to

Figure 69 and assume 70% efficiency of the pump. Calculate the total dynamic head the pump must be able to overcome and the bhp required.

<u>Solution</u>.

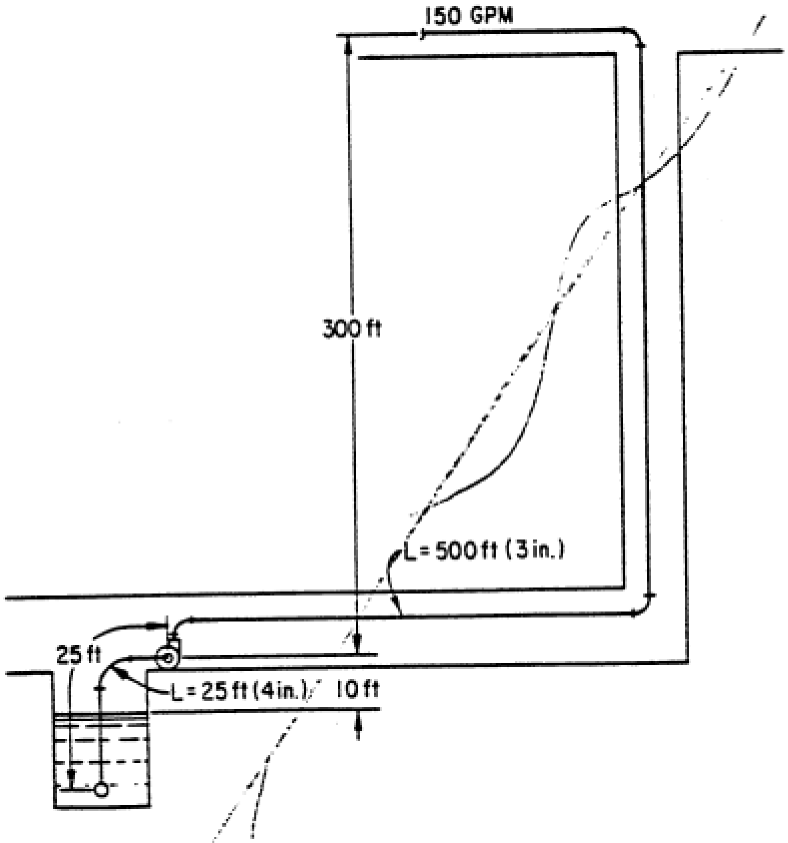


Figure 69. Pumping system for example problem.

First note that the static discharge head is 300 ft. and the static suction head is 10 ft.

Next determine the velocity head, H_v, at the discharge point on the surface. The velocity in the 3-inch discharge pipe is v = Q/A, where v must be in feet per second. So Q = 150 gpm must be converted to cubic feet per second and A must be in square feet. Online conversions of units are easy to find today, so look up 'convert gpm to cubic feet per second' and you will find 0.334 cubic feet per second. Calculating the area, A, for a circle uses the equation A = 3.1416 r^2, where r is the radius; when this is done and converted to square feet, A = 0.049 square feet. Thus the velocity is 0.334/0.049 or 6.82 feet per second. A table on the web (Engineering Tool Box, 2015)) shows that a velocity of 6.82 feet per second gives velocity head, Hv, of 0.724 ft.

Finally, the friction and shock losses in the two pipes (3-inch discharge and 4-inch suction) must be determined using the Hazen-Williams equation given previously. For the 3-inch pipe, the calculation follows:

$$H_f = [0.2083\ (100/c)^{1.852} \times Q^{1.852}]/d^{4.8655}$$

For PVC pipe, c = 150, Q = 150 gpm, and d is 3 inches, substituting numbers for the variables and calculating the friction head loss is now easy to do, as follows:

$$H_f = [0.2083\ (100/150)^{1.852} \times 150^{1.852}]/3^{4.8655}$$

$$H_f = 0.2083(0.4719)(10{,}718.2)/209.62$$

$$H_f = 5.03 \text{ ft. head loss per 100 ft. of pipe}$$

The length of the 3-inch pipe is given as 500 ft., but the shock loss of the fittings (two 90° sweep-radius elbows) must be added to this value. From an online table (Engineering Tool Box, 2015), the equivalent length for this type PVC elbow is 7.9 feet, thus for two elbows it will be 15.8 ft.

The friction head for the entire 3-inch pipe is then calculated as follows:

$$H_f = 5.03 \text{ ft.} \times (500 + 15.8) \text{ ft.}/100 \text{ ft.} = 25.94 \text{ ft.}$$

For PVC pipe, c = 150, Q = 150 gpm, and d is 4 inches, substituting numbers for the variables and calculating the friction head loss is now easy to do, as follows:

$$H_f = [0.2083\ (100/150)^{1.852} \times 150^{1.852}]/4^{4.8655}$$

$$H_f = 0.2083(0.4719)(10{,}718.2)/849.81$$

$$H_f = 1.240 \text{ ft. head loss per 100 ft. of pipe}$$

The length of the 4-inch pipe is given as 25 ft., but the shock loss of the fitting (90° sweep-radius elbows) must be added to this value. From an online table (Engineering Tool Box, 2015), the equivalent length for this type PVC elbow is 7.9 feet.

The friction head for the entire 4-inch pipe is then calculated as follows:

$$H_f = 1.240 \text{ ft.} \times (25 + 7.9) \text{ ft.}/100 \text{ ft.} = 0.408 \text{ ft.}$$

The total friction head (3-inch and 4-inch pipes) is then 26.348 ft.

Finally, the total dynamic head is calculated as follows:

$$H_{TDH} = H_s + H_f + H_v = 310 \text{ ft.} + 26.348 \text{ ft.} + 0.724 \text{ ft.} = 337.1 \text{ ft.}$$

Next pumps, including principles about them and the two major types, reciprocating and centrifugal, will be covered.

2. PUMPS

a. Principles

Since pumps are the key elements in mine drainage systems, their principles of operation and their limitations must be fully understood. A general misunderstanding exists concerning the most elementary working principles of pumps. Perhaps the use of the term suction has created the misunderstanding. Water is not pulled or sucked into a pump but it is actually pushed into the pump by atmospheric pressure. Regardless of the type of pump, a partial vacuum is created at the inlet of the pump as a result of the movement of the pump impellers, pistons, or gears, allowing atmospheric pressure to push the water into the pump. This principle results in some serious and critical limitations on pumps. At sea level, atmospheric pressure is equal to 14.7 psi, or to the weight of a column of water 34 ft. high and in inch square. Theoretically, then, in a perfect vacuum atmospheric pressure could force water 34 ft. high to reach a pump; however, a perfect vacuum can never be obtained due to leaks in pipes and pump casings, so under practical conditions a suction life of less than 34 ft. must be used. Thus in all pumping calculations atmospheric pressure is a factor, but since both intake and discharge are under the effects of atmospheric pressure, one plus and one minus, the effects are cancelled. The practical implications of this are that special care must be taken during pump installation to make sure that the total dynamic suction lift does not exceed more than about 20 ft. for trouble-free operation.

There are basically two types of pumps according to principles of operation, i.e. displacement and centrifugal, but there are many variations within each group. In the former group, the reciprocating pump is the most popular for mine use in gathering water.

b. Reciprocating Pumps

A reciprocating pump has a smooth-bore barrel or cylinder in which a piston moves back and forth. It is a positive displacement pump; that is, the piston travel is such that the water must find a suitable outlet or the pump casing will be ruptured by the buildup of pressure. Valves control the openings which lead from the pump cylinder to the section and discharge pipes; when the piston moves in one direction, a partial vacuum is created in the cylinder allowing the atmospheric pressure to open the suction valve and water is drawn into the cylinder. On the return stroke of the piston, the suction valve closes and the discharge values leading to the discharge pipe are forced open by the cylinder pressure created by the piston, and the water is forced out of the pump.

By a suitable arrangement of valves at either end of the water cylinder, a piston pump, in making a single stroke, can force water out of one end while drawing it in at the other end, and, on the return stroke, it can repeat the process but in the opposite direction. Such a pump is called double-acting, while one that requires a complete cycle of back-and-forth motion to fill the cylinder with water and then discharge it is called a single-acting pump.

Figure 70 illustrates the liquid end of a double-acting piston pump with the piston shown at both ends of the stroke. The double-acting cylinder has two separate water chambers, each with its own suction and discharge valves, with the piston operating through the center of both. Movement of the piston to the right, as shown in Figure 70C, forces water out of the right-hand chamber into the discharge pipe through discharge valve DB while, at the same time, opening suction value SA, causing the left-hand chamber to fill (C left); on the reverse stroke (C right), water is forced out of the left-hand chamber through discharge valve DA and suction valve SB is open, causing the right-hand chamber to fill. This construction allows pumping in both directions of the piston stroke, hence the name double-acting.

This type of pump is most efficient for relatively small capacities, high heads, and automatic operation. It is self-priming and its pressure is limited only by the strength of the pump and motor. Therefore, as a safety factor, some bypass arrangement is necessary to avoid blocked discharge which would destroy the pump. Since the flow of water is pulsating or irregular, air chambers are used on either the discharge or suction side, or both, to compensate for this irregularity and smooth out the flow.

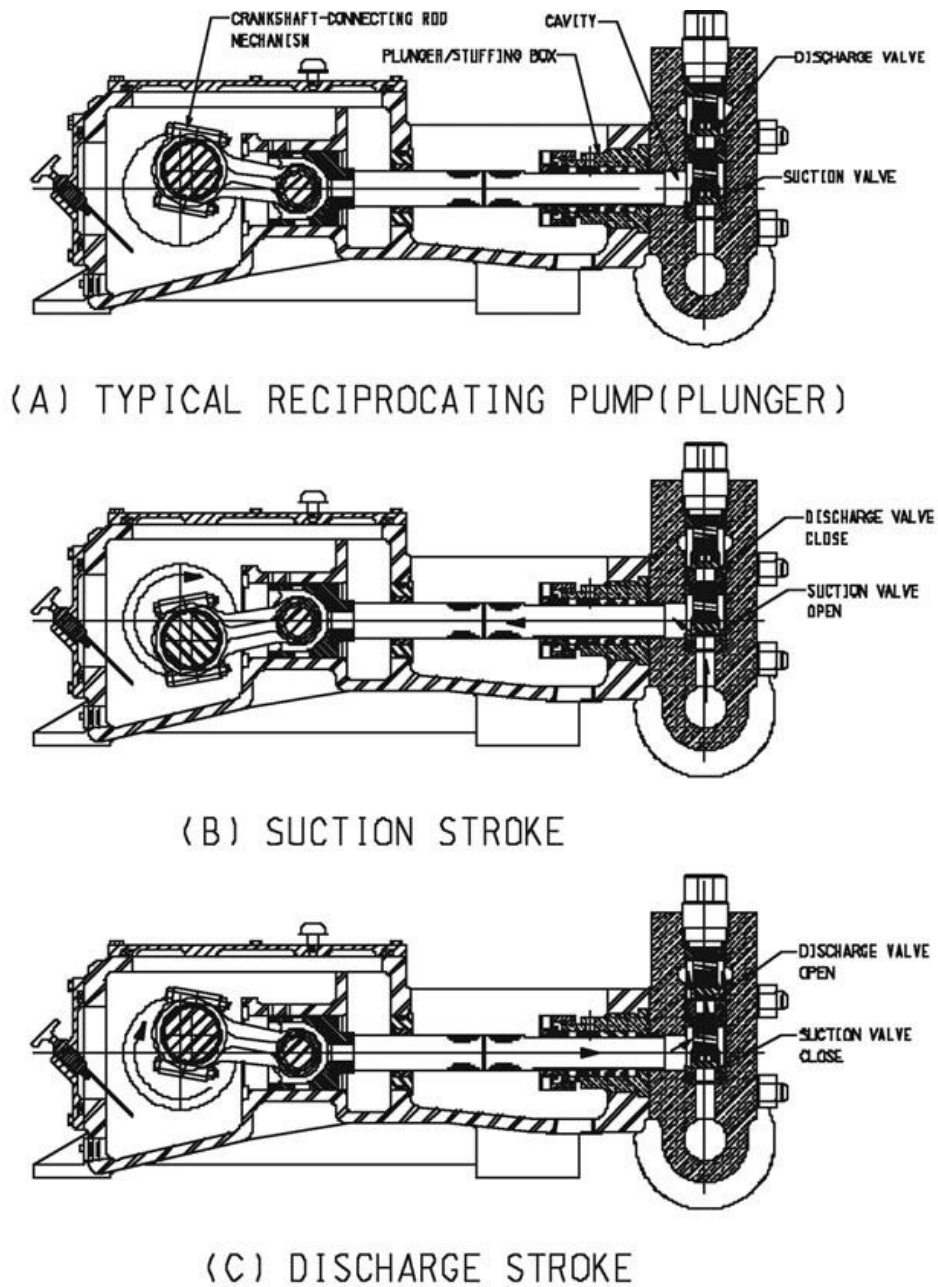


Figure 70. Illustration of a reciprocating pump and its function (Pumps and Systems, 2015).

The reciprocating pump was more popular in the past than it is now, because pumping at high heads ca be accomplished more simply today with centrifugal pumps. While the reciprocating pump is more efficient than the centrifugal pump, it is a high-maintenance device because of its valve action, it is very heavy and bulky, and it is more difficult to protect against corrosive and abrasive liquids. It is not suitable for handling muddy or gritty water since this will scour the cylinder lining.

The reciprocating pump is generally used underground in the form of auxiliary pumps that gather water and pump it to main sumps or that receive water from other pumps and transfer it to the main sump. The reciprocating pump's self-priming feature makes it efficient for intermittent operations, but in general, its disadvantages so outweigh its advantages that for mine service it is becoming less popular.

c. Centrifugal Pumps

The centrifugal pump is by far the most popular for mine pumping because it can handle large quantities of water that can be highly contaminated with foreign matter. This type of pump derives its name from the fact that it involves a force which is created by a body's moving in a circular path, i.e. a centrifugal force. In this pump, the liquid is forced to revolve and a centrifugal force is exerted by it in the case around the revolving wheel or impeller which is equal to the discharge pressure or head (see Figure 71).

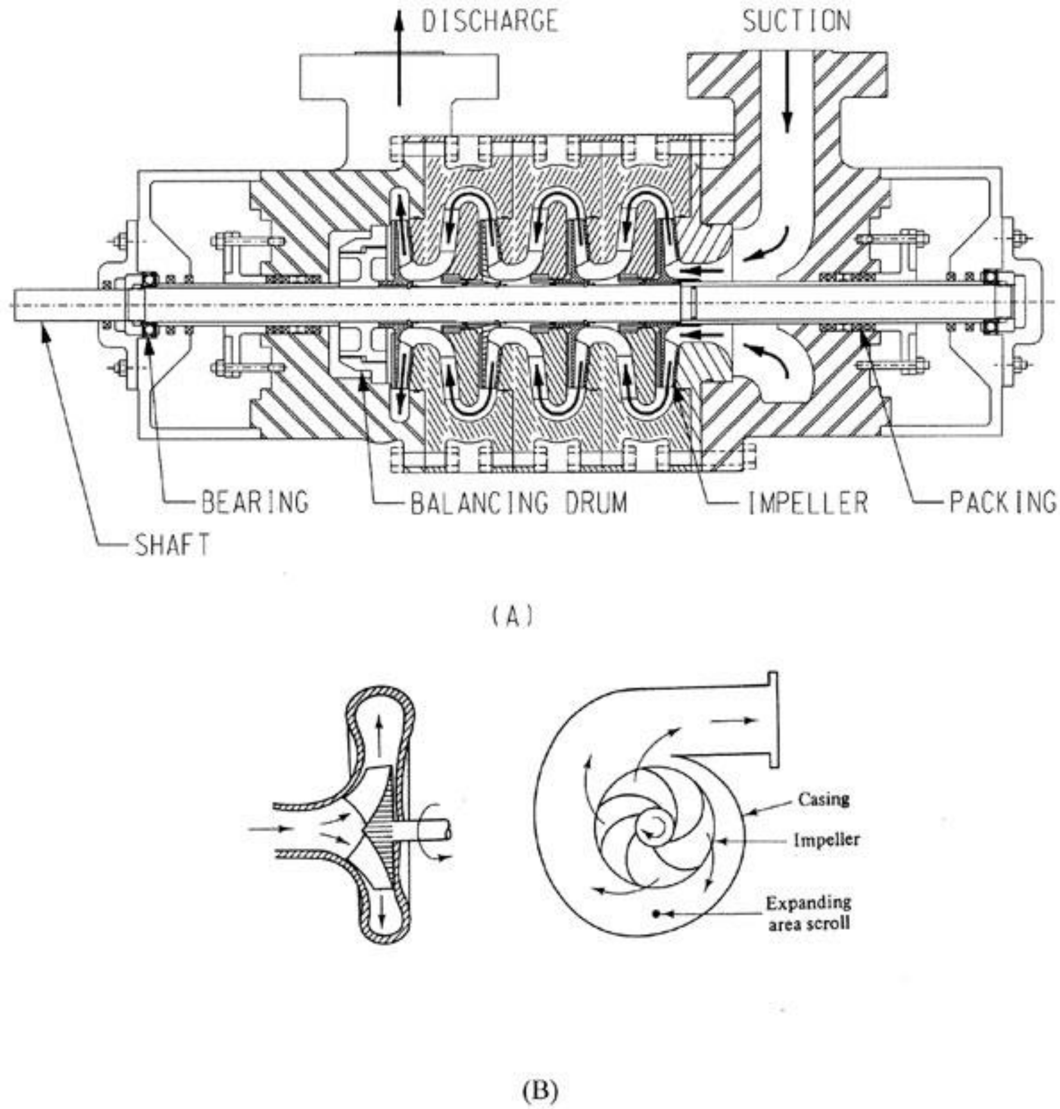


Figure 71. Illustration of centrifugal pump and the interior of it (Pumps and Systems, 2015).

The force developed by a centrifugal pump is understood easily by thinking about a boy swinging a bucket of water around above his head while no water is spilled, provided the bucket is moving around fast enough. The force that holds the water against the bottom of the bucket is centrifugal force. Suppose a hole was punched in the bottom of the bucket. A stream of water would flow out, impelled by the centrifugal force. This stream would be continuous if the boy's arm were a pipe supplying water to the bucket. The boy's arm simulates a suction pipe, the bucket is the impeller throwing a stream, and the casing of the pump has been introduced to guide the stream in one particular direction to the discharge outlet. The more rapidly the bucket is rotated, the greater the centrifugal force, creating a higher pressure at the outlet, and resulting in an elongated stream.

With no pistons, valves, or close clearances, a centrifugal pump is not positive acting. In other words, a particular impeller is good for only so much pressure or head and, if the actual pressure or head is higher than this, the impeller will merely churn the liquid.

The liquid forced out of the casing by the impeller movement creates a partial vacuum permitting atmospheric pressure to force more water into the pump suction inlet and the operation is continuous. It should be apparent that before this vacuum can be created, a pump casing must be full of water, or primed. Thus priming is an important feature in the starting of centrifugal pumps.

Among the many outstanding features of a centrifugal pump are the following:

- Simple construction,
- No close clearance required,
- Only small amount of floor space needed,
- Low first cost,
- Inexpensive and easy to maintain,
- No excessive pressures created, even with discharge valve closed,
- Practical maximum suction lift is 15 ft.,
- Smooth, non-pulsating flow,
- Impeller and shaft are the only moving parts,
- Quiet operation,
- No valves or reciprocating parts,
- Operates at high speed for belt drive and direct connection to motors, and
- Can easily be made corrosion-resistant.

While the centrifugal pump is traditionally considered a low-head pump, it can be staged, i.e. connected in series to operate against extremely large heads.

Figure 72 shows how a specific pump can be tested and its characteristic curves derived. A dynamometer is placed into the input of the shaft to measure brake horsepower (bhp). On the discharge side, a gate valve to vary the water flow is placed together with a pressure gage and a flow meter. With the gate valve completely closed and the pump running, the dynamometer records the brake horsepower and the pressure gage measures the head, the flow obviously being zero, providing the first point on the head-capacity and brake-horsepower curves. The gate valve is then opened slightly and the measurements are made again, and this time the flowmeter provides a positive indication. This sequence is repeated until the gate valve is fully opened and a maximum flow is being recorded. Drawing curves through the points gives the head-capacity and brake-horsepower curves, as shown in Figure 72. The efficiency curve can easily be calculated from the two sets of values.

The product of the pressure head (H, ft.) and the flow (Q, gpm) gives the water horsepower (WHP), or the theoretically minimum horsepower required to produce the desired result [see equations (7) and (8)]. The efficiency which is output over input or E = WHP/bhp can be found using the following formula:

$$E = Q\,H/(3960\ \text{bhp}) \qquad (9)$$

The curves shown in Figure 72 indicate what the pump will do with one specific impeller diameter, in this case the maximum diameter. However, centrifugal pumps are flexible in that one casing or volute can be used with various impeller diameters cut down from the maximum to make separate curves of several impeller diameters for each pump; however, this would result in a multiplicity of graphs which would be awkward and confusing. Therefore, curves as shown in Figure 72 are constructed which tell at a glance what the pump will do at a specified speed with various impeller diameters, from the maximum to the minimum.

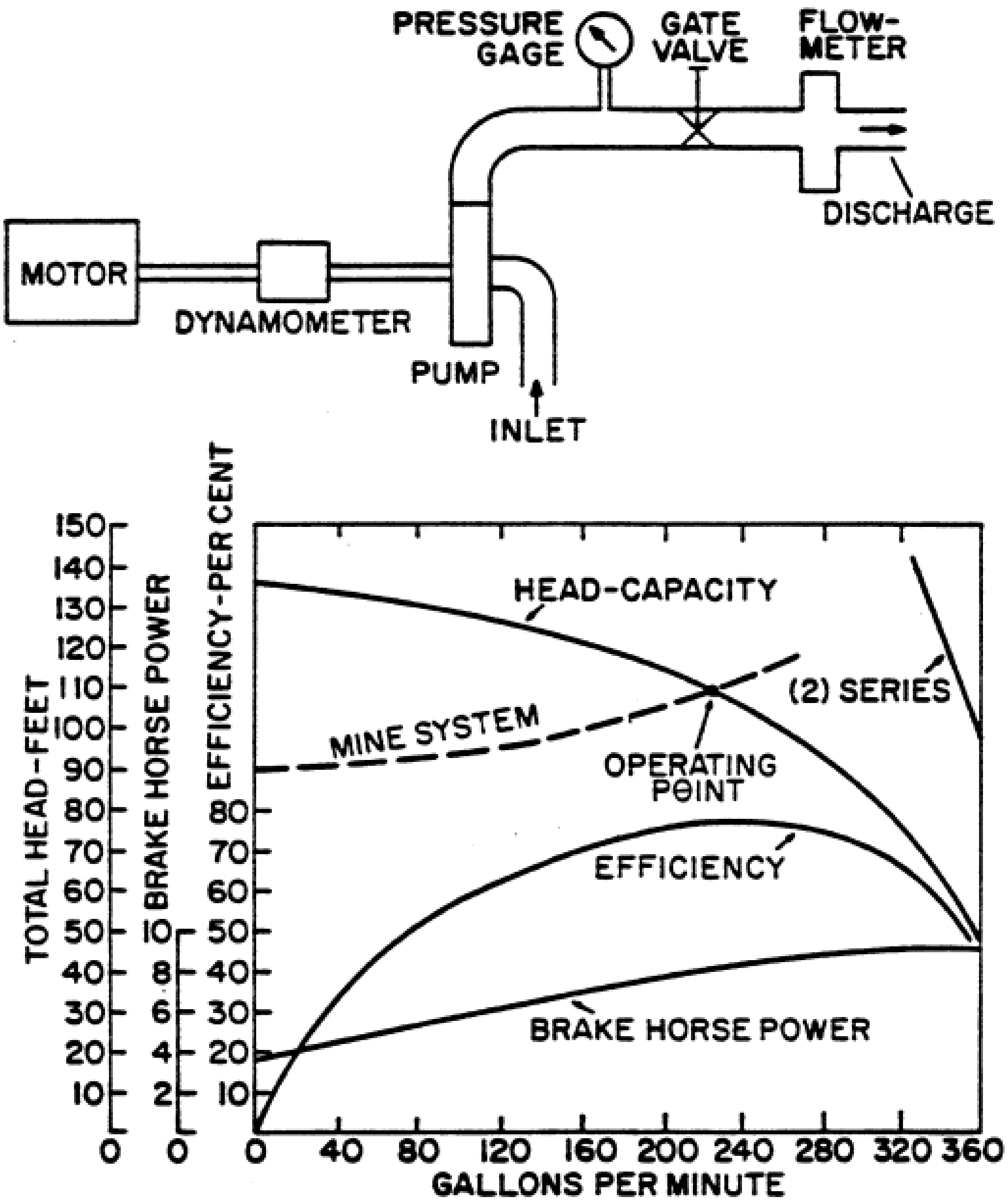


Figure 72. Centrifugal pump test arrangement to determine characteristic curves (previous study guide).

Each impeller is actually tested and separate test curves made up similar to Figure 72. All the head-capacity curves are then correlated on a single sheet. It can readily be seen, however, that, if an attempt was made to also include all the horsepower and efficiency curves, there would be such a conglomeration of lines that it would be impossible to pick out the individual curves. This is greatly simplified by taking several points of efficiency on each of the head-capacity curves and connecting them to make a composite efficiency curve, and the same can be done with the horsepower curves. For example, in Figure 73, 70% was obtained with the 5 7/8 inch diameter impeller at 162 gpm, 120 ft., and at 304 gpm, 82 ft. This same efficiency was obtained with the 5 3/9 inch diameter impeller at 150 gpm, 98 ft. and 264 gpm, 69 ft., and with the 5 inch diameter impeller at 148 gpm, 82 ft., and 225 gpm, 60 ft. Connecting all the points of the same efficiency results in the various efficiency curves.

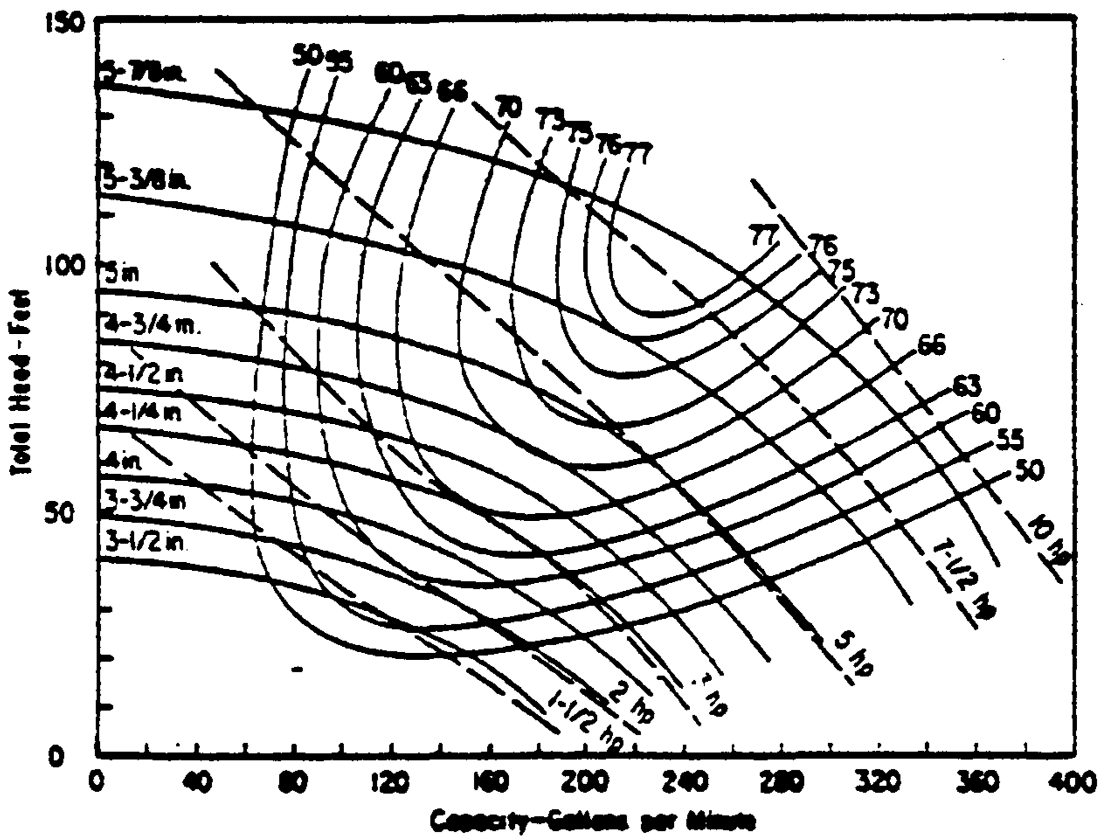


Figure 73. Pump characteristics for various impeller diameters (previous Study Guide).

Similar treatment can be given to get the various horsepower curves. Note that, arbitrarily, impellers of specific diameters have been tested. This is done as a check but does not mean that intermediate-diameter impellers cannot be furnished. For instance, if a job required 180 gpm, 105-ft head, referring to Figure 72, the impeller would be cut to approximately 5 5/8 inch diameter, the efficiency would be approximately 74%, and a 7 hp motor would be required.

There is a very practical ramification of this flexibility. Rarely will a pump be purchased for a single installation at a constant specified head and quantity flow throughout its life. Instead, a pump will be expected to work over a wide range of flows during its lifetime. Therefore, a pump casing could be purchased to accommodate the largest impeller size, but during the early stages of operation when lower quantities of flow are handled at smaller heads, a small-diameter impeller would be used, and then increasing the diameter size as needed later. In addition, even greater flexibility can be obtained from centrifugal pumps by staging them and by increasing their speed of operation.

3. STAGING

In staging, pumps are placed in series, that is, the discharge of one is put into the suction side of the other. Taking this one step further, it is possible to place two impellers on a single shaft so that one will discharge into the other (see Figure 73, left), and the pump can be provided with a fluted casing. The practical implication of this is shown in the graph in Figure 74.

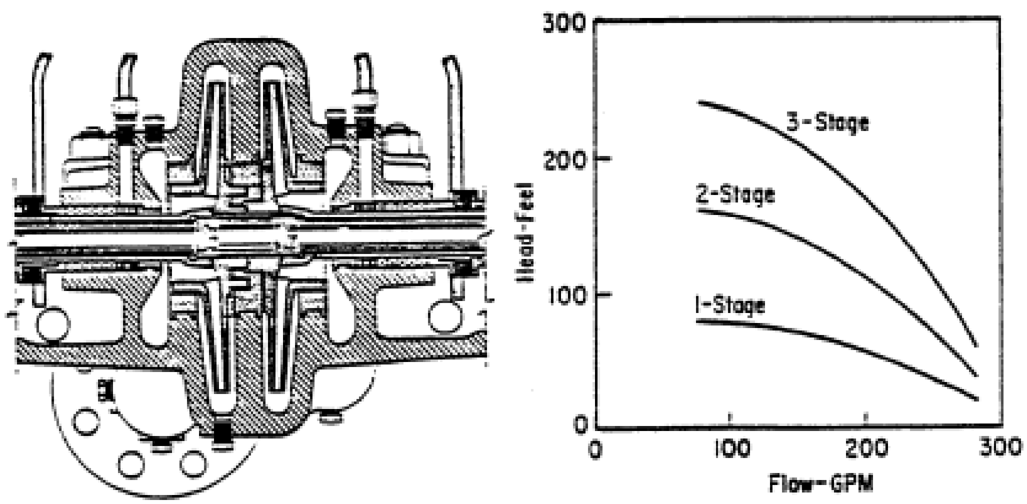


Figure 74. Two-stage centrifugal pump showing principle of staging (previous Study Guide).

If the head-quantity characteristics of a single impeller are known from a test, then the heads can be vertically added for additional stages such as 2 and 3 (see Figure 73 graph). In other words, the same quantity of water passes through each stage, and each stage boosts the head by an equal amount. The overall envelope then has the characteristic of a multi-stage pump. Obviously, there is virtually no limit to the number of stages that can be provided in a single casing. However, once such a pump is provided with four or more stages, the one complex casting used for the pump casing will be so heavy and bulky that a crane will be needed to lift it off for maintenance. Therefore, multi-stage pumps consist of a series of volute-shaped split casings with impellers arranged in pairs, back to back, each pair being, from a thrust standpoint, a double-suction impeller and therefore free from end-thrust. Such a four-stage pump is shown in Figure 75. Each of the individual top halves of a multi-stage pump cast is iron, bronze, or stainless steel is very strong, yet can be lifted by two men.

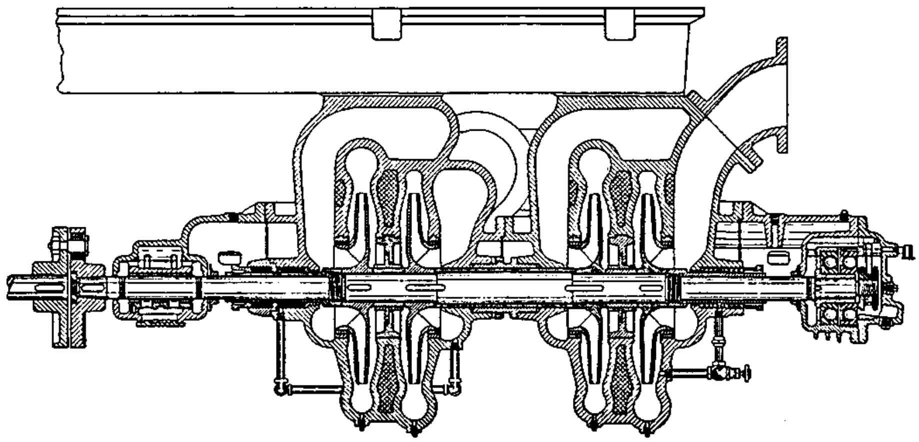


Figure 75. Four-stage centrifugal pump with split casing (previous Study Guide).

So far station duty pumps with horizontal shafts have been discussed. The same principles apply to pumps that can be used vertically in boreholes (see Figure 76). Essentially, the pump shown in Figure 76 is a centrifugal pump set on a vertical shaft in a borehole. Two different types of pumps are employed for each use. The newer type shown in Figure 76 is a submersible pump. That is, the motor is placed into the hole with the pump at the end of a pipe casing. This requires water-tight construction of the motor but has the advantage that the motor is cooled by the circulating water. Of course, a power cable is required in the hole for the motor.

Submersible pumps are built in a wide variety of sizes, often with many stages, and can achieve very high flows and heads with equivalent horsepower requirements (see Figure 77). A submersible pump can be dropped into a quarry or strip pit for dewatering purposes as well as be used underground. A small, light-weight, construction-type submersible pump, with a sealed electric motor, equipped with a suitable intake inlet guard to protect the impellers are commonly used to remove water from the production sections and other underground mine locations.

In the past, conventional deep-well pumps were employed with the motors installed on the surface at the necks of the boreholes and connected through shafting to the pumps at the bottom of the holes. Alignment problems and inefficiency have made this arrangement less popular today.

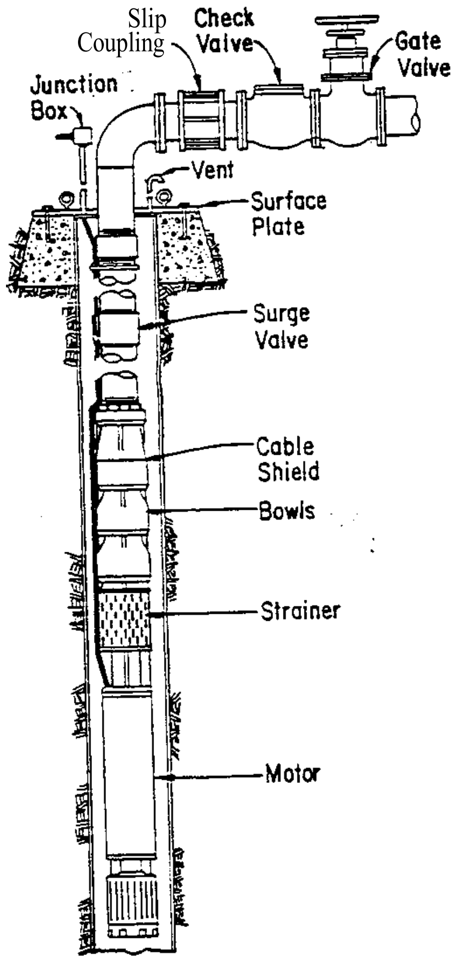


Figure 76. Cross-section of a submersible pump installation (previous Study Guide).

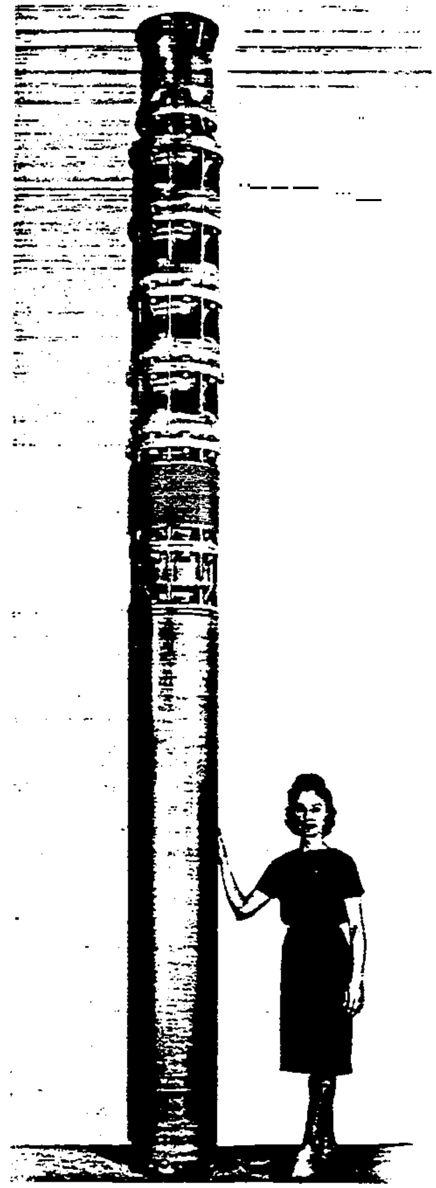


Figure 77. Large submersible pump: 300 hp, 2500 gpm, 350-ft head (previous Study Guide).

4. SPEED CONTROL

Because of the varying conditions under which a pump may be required to operate, flexibility is a desirable feature. An obvious way of varying head and/or capacity is by a speed adjustment. For a positive displacement pump, it should be apparent that with increased speed, more cylinders of water are displaced; that is, the quantity of water handled will increase linearly with the increase in speed, but the head will remain constant since it is a function of the physical characteristics of the pump. However, a reciprocating pump is a slow-speed device, and an increase in speed is possible only within very small limits.

The centrifugal pump (impeller) speed is easily adjusted, and, within reasonable limits, it is possible to derive new characteristic curves from the old ones by applying the following pump laws:

- Capacity varies in direct proportion to speed,
- Head varies in proportion to the square of the speed, and
- Horsepower varies in proportion to the cube of the speed.

A simple example will demonstrate the laws involved in a speed change.

Problem: Let's assume that the pump in Figure 72, for which characteristic curves are available, is to have its speed changed from 3450 rpm to 3900 rpm. What will be the new characteristic curves at the higher speed?

Solution: Any point may be chosen on the head-quantity curve and the brake horsepower and efficiency may be obtained at that point by reading down to the respective curves. The pump laws are applied to the arbitrarily chosen points to provide the new characteristic points which can be appropriately plotted to give the new characteristic curves at the speed of 3900 rpm, using the speed ratio (SR) of 3900/3450 = 1.13, as follows:

Q_o	H_o	HP_o	E_o	$Q_n = Q_o \times SR$	$H_n = H_o \times SR^2$	$HP_n = HP_o \times SR^3$
85	130	5.5	52	96.1	166.1	7.94
170	120	7.2	72	192.2	153.3	10.40
255	100	8.5	77	288.0	127.8	12.28
327	70	9.2	64	369.0	89.4	13.29

For example, the first point arbitrarily chose on the head-quantity curve is Q = 85 gpm, H = 130 ft. Reading down from this point to the efficiency curve provides an efficiency of 52% while continuing downward provides a bhp reading of 5.5 hp. Applying the pump laws for a speed ratio (SR) of 1.13 gives the following calculations:

$$Q_n = Q_o \times SR = 85 \times (1.13) = 96.1$$

$$H_n = H_o \times SR^2 = 130 \times (1.13)^2 = 166.1$$

$$HP_n = HP_o \times SR^3 = 5.5 \times (1.13)^3 = 7.94$$

Plotting Q = 96.1 at H = 166.1 gives the first point of the new characteristic H-Q curve. To the appropriate horsepower scale directly under this point, plot bhp = 7.94, which will provide the first point on the new bhp curve. Since the new efficiency will be:

$$E_n = \frac{Q \times SR \times H \times SR^2}{HP \times SR^3}$$

the SR's cancel out in both numerator and denominator so that the efficiency remains the same at the new transposed point (Q = 96.1 at H = 166.1) as at the old point (Q = 85 at H – 130) and the old value 52, is merely moved over under the new point to obtain the first point on the new efficiency curve. Note that while the efficiency of a point remains the same for the transposed point resulting from a speed change, the old efficiency curve is noted used; it is basically shifted to the right to derive a new curve. Similar treatment of the other arbitrarily chose points will provide points for the new characteristic curves which can be drawn as shown previously.

Some useful conversion factors for mine drainage problems are given in Table 1, along with an example on their use (from previous Study Guide).

Table 1. Conversion Factors for Mine Drainage Problems.

Liquid Measure, in gallons	Volumetric Measure in cubic inches	Volumetric Measure in cubic feet	Weight Measure, in pounds
1	231	0.134	8.342
7.481 (use 7.5)	1728	1.000	62.5

Example: How much water will a sump hold that is 65 feet long, 8½ feet wide, and 6 feet deep?

Solution: The volume of the sump is:
65 x 8.5x6 =3315 cubic feet.

The sump capacity in gallons will be:
7.5 x 3315 =24,862.5 gallons.

The weight of this water will be:
62.5 x 3315 =207,187.5 lbs or 103.5936 tons.

5. FLOW OF WATER THROUGH PIPES

The flow of water through a pipe is attended by certain phenomena which must be thoroughly understood if further study in drainage is contemplated. For example, water from a reservoir (quiet water) as it enters a pipe will suffer contraction, just as in the case of flow through an orifice. Then as it flows through the pipe, its velocity is retarded by friction of the water particles against the pipe walls. These factors cause a decrease in the velocity of the water and the stream being discharged at the end of the pipe is much smaller (if the pipe is long) than the cross-sectional area of the pipe. Initially the velocity of the water in the pipe is the same as that at the entrance, but this velocity decreases as the length of the pipe increases (provided the head of water and the pipe diameter are unchanged). The pressure forcing the water through the pipe is that exerted by the standing water in the reservoir. Pressure is measured as so many feet of vertical water column above the pipe outlet level and is called the head. The decrease in velocity of the water in the pipe is sometimes called loss of head and is caused by friction.

The loss of head at the entrance of a pipe is not important if the pipe has great length. It could be proved that a pipe 1,000 ft. long and 1 ft. in diameter under a head of 100 ft. has an entrance loss equal to 0.228 ft. or one-fourth of one percent of the total head. For this reason entrance loss can, for our work, be neglected.

The loss of head from friction in the pipe is usually large, and in long pipes it becomes so great that the discharge is only a small fraction of that which might normally be expected from such a head of water. There are a number of formulas for velocity and quantity of flow through pipes, but the simplest are the following:

<u>For Pipes from 3 to 8 Inches in Diameter</u>

Velocity: $v = 1.13\sqrt{Dh}$

Quantity: $q = 0.89\sqrt{D^5h}$

<u>For Pipes Above 8 Inches in Diameter</u>

Velocity: $v = 1.27\sqrt{Dh}$

Quantity: $q = \sqrt{D^5h}$

In these formulas, velocity, v, is in feet per second; quantity, q, is in cubic feet per second; pipe diameter, D, is in feet, and head, h, is in feet per thousand feet of pipe. An example will show the method of using these formulas.

Example: A pipe line is used to drain a reservoir. The line is 2,000 feet long, is 12 inches in diameter, and the difference in elevation between the pipe outlet and the water in the reservoir is 40 feet. Find the velocity and quantity of water which will flow under these conditions.

Solution: D = 12 in = 1 ft.

h = 40 ft. in 2,000 ft. = 20 ft. in 1,000 ft.

$$v = 1.27\sqrt{D\,h} = 1.27\sqrt{1 \times 20} = 5.68 \text{ ft. per sec}$$

$$q = \sqrt{D^5 h} = \sqrt{1^5 \times 20} = 4.47 \text{ cu ft. per sec}$$

q can also be found by multiplying the cross-sectional area of the pipe by the velocity, as follows:

$$q = 5.68 \times 3.1416 \times D^2/4 = 5.68 \times 0.7854 = 4.46 \text{ cu ft. per sec}$$

Note: $3.1416 \times D^2/4$ is the formula for the area of a circle.

These formulas are for rough inside surface pipes made of iron or steel. Regardless of how smooth the inside surface of a pipe may be when placed in service, it soon becomes rough and the above formulas will then apply. The results obtained by using smooth-bore formulas will give disappointing results if tests are made to verify the results after the pipes have been in service over time. PVC type pipes require different formulas, which will have a different lead coefficient.

The loss of head due to bends and elbows in the pipeline will vary. Easy curves will cause slight losses, while sharp curves will cause more excessive losses in head. If there are several such curves, the losses may become serious, especially where the velocity is high. A 90-degree turn will cause seven times the friction loss of a 40-degree turn. It is good policy to reduce the degree of bend when changing the direction of a pipe line and the elimination of all bends possible is still better practice.

The importance of using one pipe of sufficient size to carry the water instead of several small pipes is emphasized in some of the problems, which follow; it is also illustrated in Table 2, which shows the loss of head by friction.

Table 2. Loss of Head in Pipe Flow Due to Friction (previous Study Guide).

LOSS OF HEAD DUE TO FRICTION IN FEET

Velocity feet per second	Inside diameter of pipe, in inches 1	2	3	4	5	6	7	8	9	10
2.0	2.37	1.185	.791	.593	.474	.395	.338	.296	.264	.237
5.0	12.33	6.170	4.110	3.080	2.460	2.050	1.760	1.540	1.370	1.230
7.0	22.89	11.450	7.620	5.720	4.570	3.810	3.260	2.850	2.520	2.280

The tremendous increase in friction loss as the velocity increases is due to the fact that the loss increases as the square of the velocity. The value of using larger pipes is shown in the instance where a 10-inch pipe will carry water at a velocity of 7 feet per second with less friction loss than a 1-inch pipe can carry water at a velocity of 2 feet per second.

Problem: How many 2-inch pipes will carry the same amount of water as one 10-inch pipe, all being of the same length and under the same head of water?

Solution: The formula for quantity, q, above shows that the quantity varies with the square root of the fifth power of the diameter. Therefore, designating x as the number of 2-inch pipes necessary, we have the following:

$$q = x\sqrt{2^5 h} = \sqrt{10^5 h}$$

Setting the two right-hand equations equal, and noting that the h cancels out, then we solve the following equation for x:

$x = \sqrt{10^5/2^5}$ = 55.9, or 56 pipes needed for equivalency of flow, q.

Problem: Will two 3-inch pipes be better than one 5-inch pipe in carrying water in a discharge line 1000 feet long?

Solution: The statement of the problem indicates a constant head and equal lengths of pipe. The quantity of water, then, will vary as the square of D to the fifth power. Let q be the flow in one 5-inch pipe and q_1 be the flow in two 3-inch pipes. Then:

$$q : q_1 \text{ as } \sqrt{5^5} : 2\sqrt{3^5}$$

which gives a ratio of 55.9 : 31 (55.9 to 31).

Then, since the quantities bear the ratio of 31 to 55.9, for each 31 gallons of water flowing in the two 3-inch pipes there will be almost 56 gallons of water flowing in the one 5-inch pipe under the same head and length of pipe. The latter installation, one 5-inch pipe, will also be more economical in first cost and in the cost of laying the pipe.

6. INSTALLATION AND OPERATION OF RECIPROCATING PUMPS

Installation

Location. The pump should be located as near the source of supply as possible. It should not be set too high above the surface of the water that is to be pumped. At sea level a pump will not work well if it is more than 22 feet vertically above the sump and one foot less for every 1,320 of elevation above sea level.

Foundation. The pump should be securely fastened to a firm and level foundation. This avoids strain on the pipe connections and keeps the pump in correct relation to its driving unit. For larger pumps a concrete foundation is advisable.

Piping. All piping should be in as near a straight line as possible. Sharp turns should be avoided. Al joints and connections should be tight. The pipe should be large enough to pass the water without too high of a friction head. The pump manufacturers usually specify a minimum size pipe, but this can be checked as previously explained under flow of water through pipes.

Suction Pipes. The suction pipe in particular should be absolutely tight as a small leak here will greatly reduce the efficiency of the pump. Where the suction line is laid on a grade, care should be used to have this grade uniform to avoid air pockets. As a general rule, the suction line should have a drop of not less than 6 inches in each 100 feet toward the surface of the water.

Vacuum Chamber. The use of a vacuum chamber is desirable and usually necessary where the suction line is long, or has a high lift, or where there is a suction head or pressure. In any case, it steadies the flow and the air cushion takes up part of the shock which would otherwise come on the working parts of the pump.

Foot Valve. Where the suction lift is over 12 to 15 feet vertical, or where the suction line is over 100 feet on a slope, a foot valve should be used on the end of the suction pipe. This will keep the suction line and pump filled with water while idle and avoids the necessity for priming the pump each time it is started. However, the condition of the water to be pumped should be investigated before installing a foot valve on the suction line of a mine pump. The corrosive action of the sulfuric acid in most mine water may cause the foot valve to leak, in which case the water would leave the suction line and pump. If this occurred, the pump would have to be primed before it could operate. The investigation would find when a new foot valve would be needed.

Strainer. A strainer of liberal area which can be examined occasionally should be placed on the end of the suction pipe to prevent foreign substances from being drawn into the pump and clogging the valves. Periodic cleaning of the strainer is necessary if the pump is to operate in a trouble-free manner.

Water Relief Valve or Safety Valve. A relief valve of ample size set at a pressure slightly higher than that at which the pump is to work should be placed in the discharge pipe between the pump and the first shut off valve in the line. This will avoid damage in case the pump is started with the valve closed or the pipe line gets blocked.

Gate Valve. Always use gate valves instead of globe valves. They are more positive in their action and offer less resistance to flow. It is convenient to have gate valves in both discharge and suction lines in case the pump has to be shut down for inspection.

Drain Pipes. Drain pipes should be provided to carry off any water which accumulates around stuffing boxes.

Operation

Priming. If the pump and suction line are full of air, it may be necessary to prime the pump. If priming pumps are not provided, it will be necessary to take of the discharge valve covers, lift the valves and fill the cylinders with water, and allow the air to escape through the petcocks or plugs provided for that purpose.

Glands. Do not screw the glands down too tightly, especially when starting the pump, because the packing will swell and be too tight, thereby causing unnecessary friction. Tighten the glands only enough to stop leakage; having them too tight may split the cylinder.

Packing. New piston packing should be soaked in warm water for several hours before it is fitted in place; even then it should not be crowded in too tightly. It should be renewed before it becomes too hard and cuts the piston.

Valves should be examined occasionally to see that they are seating properly. When metal valves are used, they should be ground into their seats occasionally to insure their seating is proper.

Lubrication and Bearings. Care should be used to have all bearings properly lubricated to prevent their burning out or wearing loose. When a bearing becomes loose, it should be given attention immediately because a loose bearing, if neglected, will cause pounding, excessive wear, and breakage.

By-pass. Pumps driven by electric motors or mechanism which do not start well under full load should have by-passes which can be opened before starting and gradually closed as the pump gains speed.

Freezing. If the pump is located where there is danger of freezing, it should be completely drained by means of the plugs or petcocks provided for this purpose, if it is to stand idle for any length of time.

7. INSTALLATION AND OPERATION OF CENTRIFUGAL PUMPS

In general, the points mentioned in connection with the installation of reciprocating pumps will also apply to the installation of centrifugal pumps. The latter, however, are usually more permanent in character, so a few additional points on installation are necessary.

Permanence. Centrifugal pump installations are usually located at one point for the life of the mine so greater care should be taken when making the installation. The pump room should be a concrete or masonry lined room with a concrete floor. Light should be plentiful and every attempt should be made to keep the room clean.

Power Equipment. A.C. current is used, with the voltage ranging from 400 up to 2200 volts. With such high voltage present in a confined space, it is necessary to plan the location of each piece of equipment to utilize the available space to the utmost, and also to arrange the starter in a way where it will be difficult for anyone to tamper with.

Location. Since centrifugal pumps usually pump to the surface, the sump near which they are placed may be located at many points other than near the shaft bottom. Power can be taken to the motor(s) through a borehole, if necessary, and the water can be discharged through another borehole.

Safety. Since high voltage will exist in many centrifugal pump rooms, it is important to place a gate and a warning sign across the entrance(s), and this room should be kept locked. Screens/guards at the side and back of electrical equipment are advisable.

Attendant. It may be necessary to have the operation of main pumping stations monitored remotely by an attendant at a control room, especially when there is a significant threat of flooding areas of the mine if an electrical or mechanical failure should occur. Further, reliable devices (electrodes) can be used whereby the pumps will automatically start when the sump level rises to a certain height, and the pumps will be automatically stopped when the water in the sump has been lowered to a certain level. In such cases remote monitoring by an attendant can ensure designed performance. Occasionally, a pumper will ensure that the lubrication, control panels, and other essential features are functioning properly.

Operation

Priming. Do not attempt to run a centrifugal pump empty. It must be primed. Several methods are used for priming, as follows:

- If the pump is below water level, water will flow into the suction pipe and casing due to gravity when air is let out through petcocks provided for that purpose.

- A by-pass around the check valve in the discharge line may be used for priming when the discharge pipe is kept full of water.

- An ordinary cistern pump connected to the opening at the top of a small pump (up to 4 inches) can be used to exhaust the air and thus fill the pump. On larger sizes a vacuum pump is used.

- The pump may be filled with water from an outside source.

When priming by filling the pump with water either from an outside source or a by-pass, the foot valve will have to be closed and air cocks opened. In fact, in any method except exhaust pump or ejector, the air cocks have to be opened and the water should not be allowed to flow in faster than the air is let out, otherwise air pockets will form.

Starting. After the pump has been primed, it is ready to start. The gate valve in the discharge line should always be closed when starting. To remove the rest of the air after the pump has started, open the air cock and leave it open until a steady stream of water flows from it, then close it. Do not allow the pump to run very long with the gate valve closed or it will heat up. Glands should not be drawn up tight; a reasonable amount of leakage is necessary through the stuffing boxes and glands.

8. PRESSURE OF LIQUIDS ON SURFACES

The law governing the pressure of liquids on surfaces is as follows: The pressure of a liquid on the surface of its container or on the surface of any body in the liquid is equal to the weight of the column of liquid above the surface being considered.

For instance, the pressure exerted on a square foot of area of the bottom of a tank 6 ft. high and completely filled with water will equal the weight of a column of water one foot square and 6 feet high. The pressure on any area, large or small, which is under water will equal the weight of a column of water having the same area and with a height equal to the depth of the water over the area. The pressure exerted by this column not only acts downward but also acts horizontally and with equal force; thus the pressure on the sides of the tank at the 6-ft depth will also be equal to the weight of a 6-ft column of water. The pressure against any surface, whether horizontal, vertical, or inclined, always acts perpendicularly to that surface.

It is sometimes desirable to find the pressure exerted on a certain area of surface which is not horizontal like the bottom of a tank but is inclined or vertical. The pressure exerted on any one point of such an inclined surface would equal the weight of the water column over that point. However, since the surface is inclined, there may be considerable differences between the height of the water column at various points over the entire area. To obtain the total pressure, then, would mean taking the average of the pressure at various points on the surface, and the accuracy of this value of total pressure would depend on the accuracy of the readings taken, that is, they should be scattered uniformly over the surface. All of this is unnecessary for our purpose for by using the pressure on one point, called the center of gravity of the surface, we have the average of all the pressures; multiplying this average pressure by the area of the surface gives the pressure exerted on the entire area.

Any horizontal surface, of course, has equal depths of water over its entire area and the center of gravity does not enter into the calculations. If the inclined surface is a parallelogram (including rectangles, squares, rhomboids, and rhombuses), the center of gravity is at the intersection of the two lines drawn from the midpoints of opposite sides; the center of gravity of a rectangle, for example, will be located on the line midway between the two vertical sides and halfway from the top to the bottom. The center of gravity of a triangle is located one-third of the distance along the line drawn from the midpoint of the base to the opposite vertex. The center of gravity of a circle or an ellipse, or of any figure which approximates these (hexagon, octagon) is at the geometrical center of the figure. If the quadrilateral figure is not a parallelogram (trapezoid or a trapezium) its center of gravity is found as follows (follow the diagram on the trapezoid in Figure

78): Divide the area into two triangles with a diagonal between opposite corners; find the center of gravity of each triangle and draw a line between them; repeat, drawing a diagonal between the two remaining corners and finding the center of gravity for each triangle; the point where the center of gravity lines intersect will be the center of gravity for the quadrilateral. This method could be employed for parallelograms also, but the first method is much easier.

Example: A gate in a dam is 8 ft. wide and 6 ft. high. The dam is filled to within 1 ft. of the top of the gate. Find the pressure exerted on the gate by the water.

Solution: Since the gate has the form of a rectangle, the center of gravity will be located on a line midway between the water level and the gate bottom or 2 ft. below water level. Then:

Wet area of gate = 8 x 5 = 40 sq. ft.
Pressure on gate = 2.5 x 40 x 62.5 – 6250 pounds

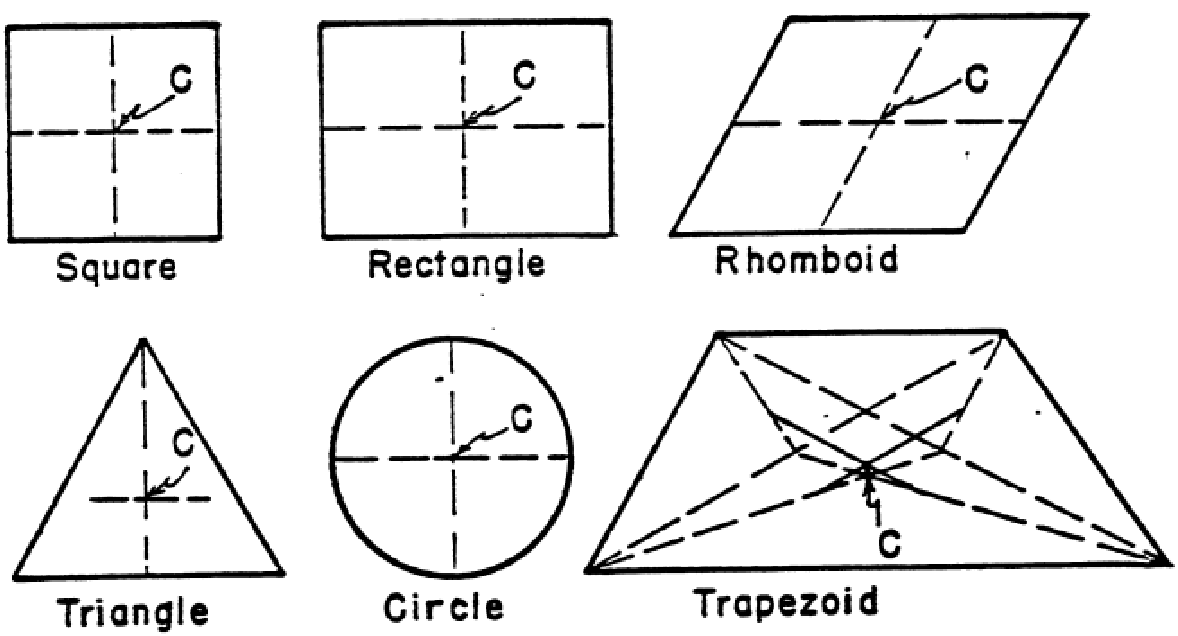


Figure 78. Centers of gravity of common geometrical figures.

Example: A rectangular shaft 600 ft. deep is full of water. What is the pressure per square inch and the total pressure on the shaft bottom (neglecting any atmospheric pressure) if the shaft has a cross-section 14 ft. by 18 ft.?

Solution: Weight of water column one inch square and 1 ft. high equals

$$62.5/(12 \times 12) = 0.434 \text{ pounds}$$

Then, pressure of water in shaft per square inch equals

$$600 \times 0.434 = 260.4 \text{ pounds}$$

Total pressure on shaft bottom equals

$$600 \times (14 \times 18) \times 62.5 = 9{,}450{,}000 \text{ pounds or } 4725 \text{ tons}$$

Example: Find the pressure per square inch on the bottom of a column pipe full of water and lying on a slope 300 ft. long, pitching 45^o.

Solution: Vertical height of water column equals

$$300 \sin 45^o = 300 \times 0.70711 = 212.133 \text{ ft.}$$

Pressure per square inch due to water column equals

$$212.133 \times 0.434 = 92.06 \text{ pounds}$$

9. PRESSURE AGAINST DAMS

The calculation of pressures on dams that are retaining quiet water is made in the same manner as the calculation of pressures against the walls of retaining vessels. That is, the total pressure, P, acting against the dam in Figure 79, which is for the purpose of illustration, 14 ft. high and 40 ft. long with 12 ft. of water being retained, will be:

$$40 \times 12 \times 6 \times 62.5 = 180{,}000 \text{ pounds}$$

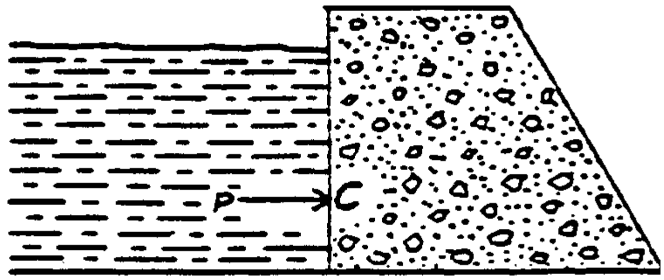


Figure 79. Action of water pressure on a dam (previous Study Guide).

In Figure 80 where the face presented to the water is inclined, the area covered by the water may be 20 ft. high along the slope. If the dam length is the same as the first illustration, and the water level has not been changed, the pressure in this case will be:

40 x 20 x 6 x 62.5 = 300,000 pounds

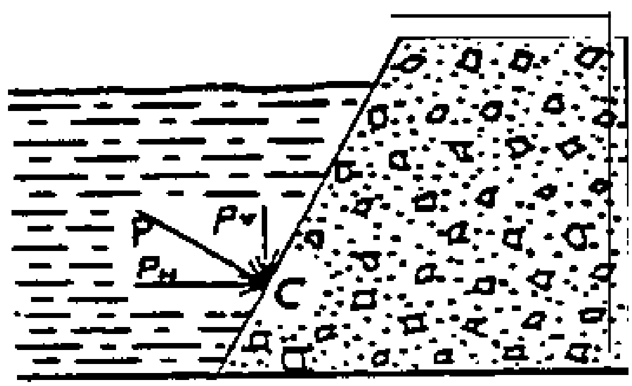


Figure 80. Action of water pressure on a dam with a sloped face (previous Study Guide).

In the first case, the pressure of the water acts perpendicularly on the surface of the dam and tends to overturn the wall or make it slide on its base. This turning or sliding effect could be calculated, if necessary, because the result would be the same as if the total pressure were concentrated along a line two-thirds of the total depth from the water, or at point C. For any single vertical strip of wall, this is called the point of application, or center of pressure. In the second case, the pressure of the water will have its point of application at the same depth, or at point C, and will be two-thirds of the wet face down along the face; the pressure will also act perpendicularly on the surface at this point. However, the pressure which acts in this manner will have a vertical component, P_v, tending to hold the dam in place and a horizontal component, P_h, tending to move the dam.

The pressure exerted against the dam depends on the depth of the water being retained and not on the expanse of water backed up by the dam. In the aforementioned illustrations, if the water extended back from the dam 1000 ft., it would exert no more pressure on the dam than if it extended back only 10 ft.

Distribution of Pressure

In the foregoing illustrations of the dams, the pressure exerted by the water equally distributed against the face of the walls even though its center of application of pressure is placed at a point two-thirds of the depth from the surface. This is due to the individual pressures becoming greater as the depth increases; in other words, the pressure on one square foot of dam surface at the top of the wall will be less than one square foot of surface at the bottom of the wall.

Since the pressure of the water against any surface acts perpendicularly on that surface, the greatest need for strength in a dam is near the bottom or base of the retaining wall. The greatest design thickness of wall is always at the base; any bracing of mine dams should be done with the idea of presenting the greatest resistance at the bottom. For the same reason, the retaining hoops of a circular tank are stronger and placed closer together near the bottom.

If we had two vessels with the same area of base, but one had straight sides while the other had curved sides (either outward or inward), the pressure exerted on the base, or on the sides at any certain height, by the same depth of water, will be the same. This parallels those cases in mines where an entry connected to a shift is filled with water, and the shaft is also full of water; the entry does not have the water directly over it, yet the pressure on its walls will be the same as on the walls of the shaft at the same depth. That is the reason why headings or gangways approaching water-filled old workings must be driven cautiously.

10. MINE DAMS

Dams are not, as a general rule, a part of a mine drainage system. They are most frequently used to hold back water in one section of a mine so that it can be flooded to extinguish a fire; or they may be used to hold back water in a section that has passed through water-bearing strata and thus prevent the water from causing trouble in other working sections. Sometimes a section, or even an entire mine, may be used as a reservoir to admit water to pumps under pressure. A plan view of a typical masonry dam across a mine entry is shown in Figure 81.

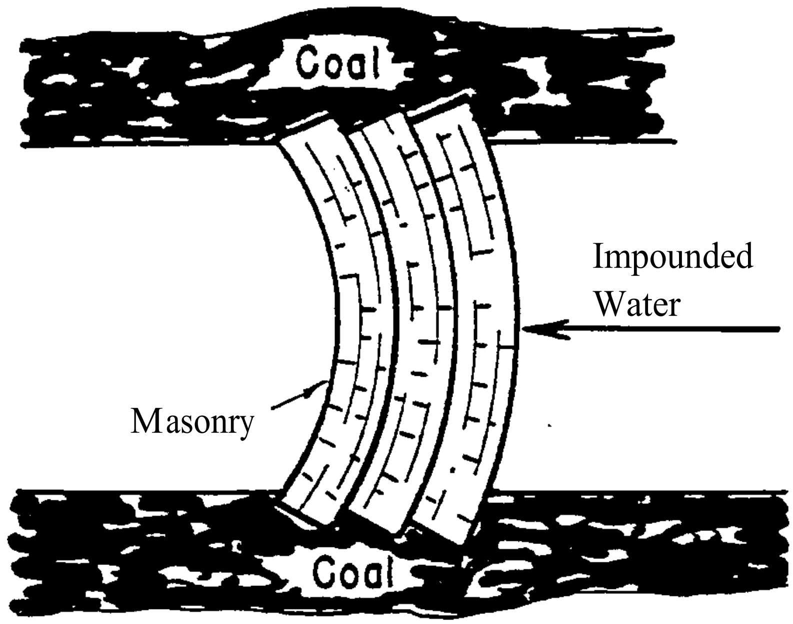


Figure 81. Plan view of a typical masonry dam (previous Study Guide).

Regardless of the use to which the dam is put, there are certain principles of construction which may be used in solving such problems, which follow:

- A dam is required by safety regulations not to be built at an elevation above any area in the mine where miners may work.

- Any dam must be calculated to resist the pressure of the greatest head of water that might be encountered. Safety demands that the dam have a good firm bearing in the floor and roof as well as in the ribs.

- Dams, in order to resist great heads, should be built of brick, masonry, plain concrete, or reinforced concrete. Brick and masonry cost the most, and are less used, while reinforced concrete and, at times, plain concrete are most often used.

- Reinforced concrete dams will be more suitable under different conditions than dams constructed of any other material. This is because reinforced concrete will take tension as well as compression, and the dam can be constructed to resist the same head of water on either side. Brick, masonry, and plain concrete will take compression only due to their low tensile strength; dams of these materials will resist a head of water coming only on the convex (outward curved) side. Reinforced concrete will resist compression best, with plain concrete next and brick-work last; masonry will give varying resistances to compression depending on the manner of construction.

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TABLE OF CONTENTS

MINE MAPS AND SURVEYING

M. MINE MAPS AND SURVEYING

Mapping and surveying are two important functions of a mining operation. The importance of having a comprehensive maps of a mine which incorporate all the important information about a mine and the features surrounding it, both laterally and vertically, cannot be overemphasized. All information on natural features [rivers, lakes, hills and valleys, faults, types of strata, etc.] and cultural features [roads, buildings, other mines, etc.] on which may depend the health and safety of miners, the safety and stability of the mine, and the safety and stability of the surrounding community, need to be collected and shown on various maps that are prepared for the mine.

For our discussion here, the map of a mine is the actual representation on paper the realization of a mine plan – the plan to develop and extract the coal seam by systematically driving entries and cross-cuts to develop pillars and extract them. Mines are laid out on a predetermined plan for such things as distances between entries and cross-cuts, distances between rows of bolts and bolts in a row, width and height of entries, advance with a miner during a cut, the ventilation requirements, etc., to ensure the health and safety of the miners and the stability of the mine. Two important purposes of a mine plan are to accurately represent the status of the mine workings as of date and to serve as a control tool to check against the approved mine plan. Therefore, it is important for the mine officials and miners to ensure that when operations such as coal cutting, roof supporting, ventilation, and other operations are carried out, they are done in a manner that adherence to the plan is strictly followed.

Mine surveying encompasses the tools, techniques and methods that are used to measure and calculate such things as angles, lengths, areas, volumes, and positions and mapping them. It is a science that is rooted in many other sciences such as physics, mathematics, engineering, and astronomy. Needless to say, to produce accurate maps, accurate surveying is essential.

Section 224 of the Safety Laws of Pennsylvania for Underground Bituminous Coal Mines specifies mapping requirements and surveying standards. The map of the mine should be made by a registered mining engineer or registered professional surveyor. A true copy of the map shall be kept in the mine office for the use of mine officials and the department. With regard to surveying underground, a survey station spad should be within 300 feet of the deepest penetration of the final faces of each mining section, butt or room. See Figure MM-1 for the explanation of these mining terms.

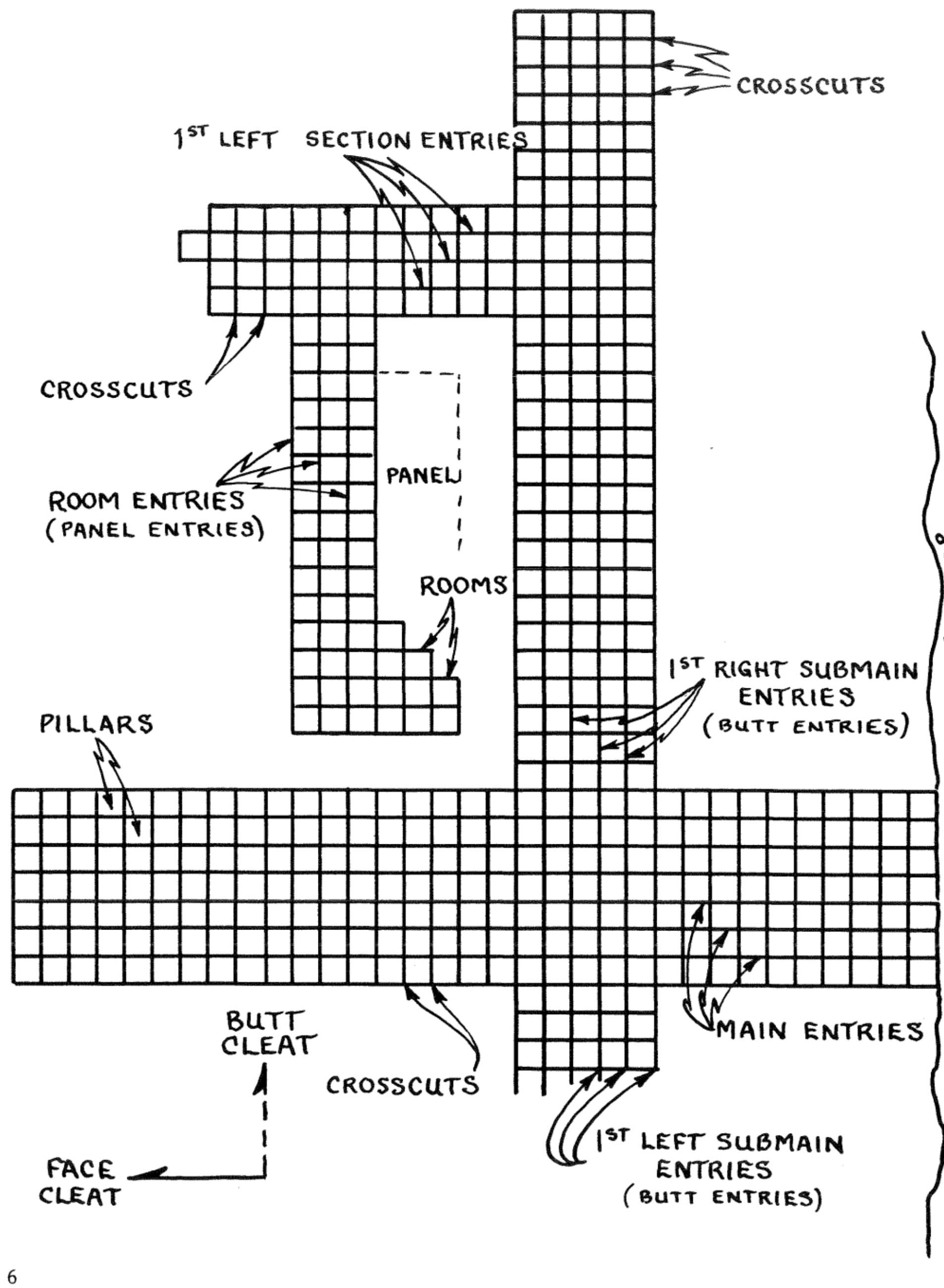


Figure MM-1: Explanation of Mining Terms
[Source: Coal Mine Map Guide, Mining Extension Service, College of Mines and Energy Resources,
West Virginia University, Study Guide Series No. 8]

In addition to the mapping and surveying requirements of an active mine, there are specific provisions with regard to the measurements and markings that are required both in underground and in mine maps before a mine is abandoned or closed for an extended period from active mining. The availability of accurate of maps of all mines, either active or abandoned, in a mining district is important for planning any future mine in the district. The experience of the mining industry in many countries and in several states in the U.S., is that, without such accurate maps, the mine workings of active mines have cut into these abandoned mines, causing enormous damage with or without accompanying loss of lives.

Today, with the advances in computer hardware and software, mapping and surveying are greatly automated with electronic instruments and 3-D programs. These have greatly simplified the tasks of conducting the surveys and updating the mine maps on a regular basis by the registered surveyor or the registered engineer. However, it is necessary to ensure that the mine map is kept up to date between the surveys by the surveyor and the required updating by the registered engineer or professional surveyor on regular intervals. Therefore, it is important for mine officials to be familiar with the symbols used in mine maps, to keep the maps of their working sections up to date and to know the simple field methods for extending the center lines in underground.

N. MINE MAP SYMBOLS

There are a large number of surface and underground features that are to be identified in a mine map as specified in Section 224. The scale of the map should not be less than 200 feet to an inch. A complete legend identifying all features represented on the map and a title block of ownership and changes of ownership with dates are the very first requirement. It is not the purpose here to go into all the map requirements and all the features that must be identified in a mine map. The mine officials must be familiar with all these requirements and should be able to identifying them on the map. While symbols for many of the features are standard, one must familiarize with the system used in the maps of the company/mine. A list of symbols is shown in Figure MM-2.

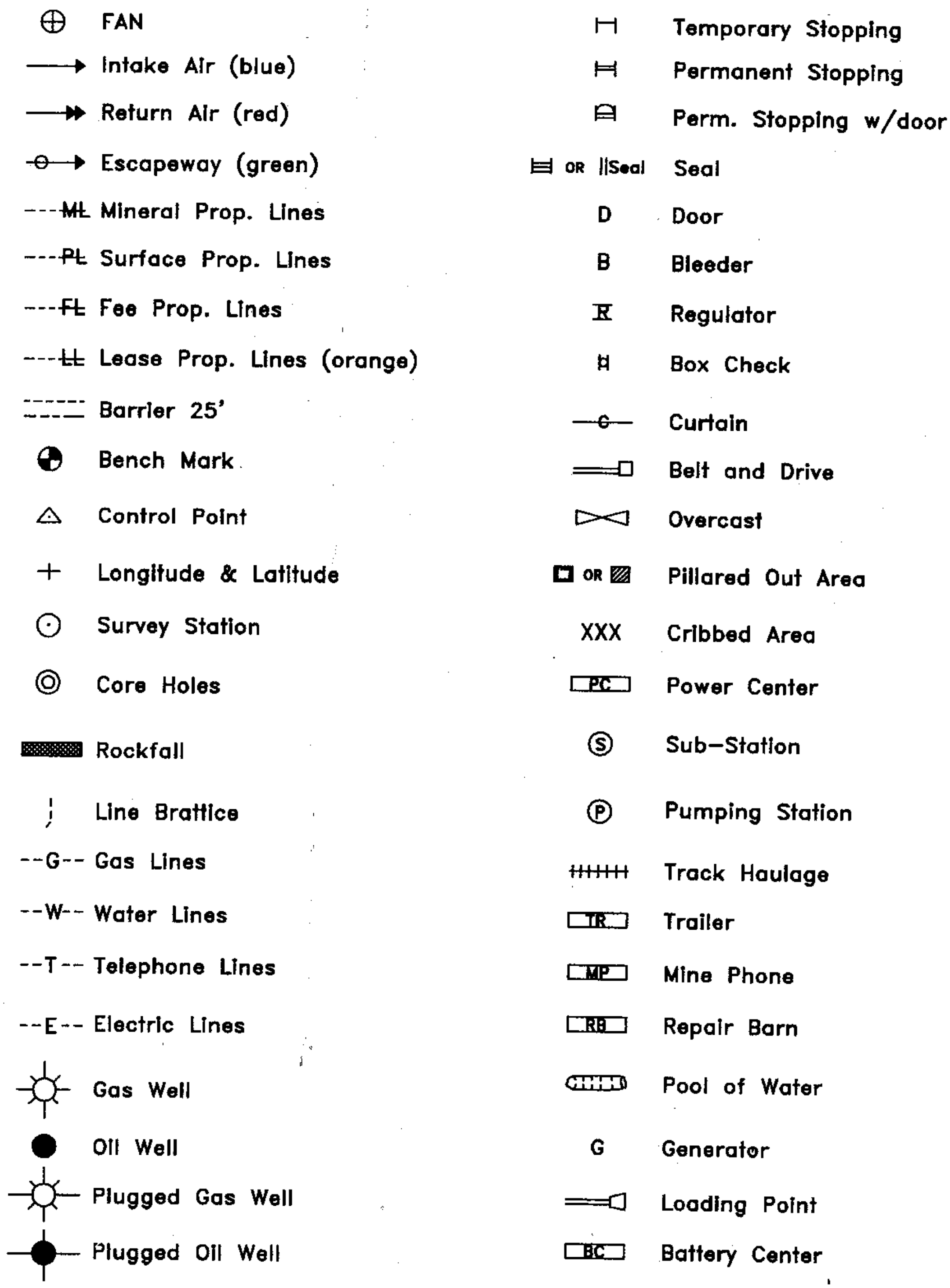


Figure MM-2: Mine Map Symbols
[Source: Coal Mining Reference Book, *Fifth Edition*, Revised
Kentucky Mining Institute, 2002]
Mine Surveying

There are a number of accuracy standards established for surveying as well as for establishment of survey spads, survey stations and check survey stations. For example, every entry must be surveyed at intervals not to exceed 300 lineal feet. Survey spads shall be established in each entry of all mains, sections, butts, rooms, and other excavations. Further, a survey station spad is required to be within 300 feet of the deepest penetration of the final faces of each mining section, butt or room. The survey station spads in the sections will be tied to a check survey station.

O. MINE SPAD

A mine spad is installed in the roof by drilling a hole, about 2 or so inches in depth and about 0.5 inches in diameter, along the center line of an entry by the surveyor. First, a wooden plug is inserted into the hole tight and then a metal key or spad is driven into the plug. The spad has a stamped survey station number [Figure MM-3]. At a short distance from the spad, along the center line, is installed another spad, called a pointer spad. The spad and the pointer spad are used in combination to extend the center line of an entry. When the entry is sufficiently advanced such that development will proceed in a new direction, a new spad and pointer spad will be installed to indicate the new centerline.

Figure MM-3: Mine Spad
[Source: Coal Mine Map Guide, Mining Extension Service, College of Mines and Energy Resources,
West Virginia University, Study Guide Series No. 8]

P. MINE PROJECTION

Shown in the Figure MM-4 is a projection of the part of mining section where four entries are being advanced. As shown here, the pillars are on 80 ft. centers. The entry and cross-cut widths, though not specified in the figure, can vary from 16 ft. to 20 ft. The single lines show the future development of the entry.

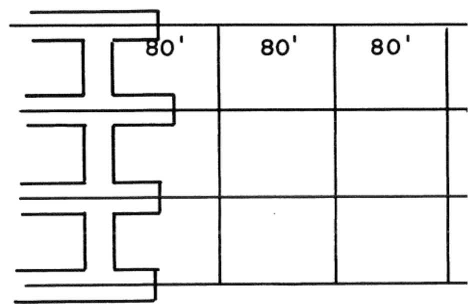


Figure MM-4: A Four Entry Mine Projection

It is important to keep the entries aligned properly and the cross-cuts turned at the right distances from the entries so that the pillars are properly oriented and are of the right dimensions. While in Figure MM-4, the cross-cuts are at right angles to the entries, i.e. at 90 degrees to the entries, cross-cuts may be at angles of 30, 45 or 60 degrees or at any other angle to the entries. In general, the surveyor will install the spads and mark the center lines of the entries and the cross-cuts. However, there are times when, in between scheduled surveys and updates, the mine official has to extend the center line of the entries and mark off the directions of the cross-cuts. With the use of a string and a marking chalk, and knowledge of elementary trigonometry, these tasks can be accomplished.

Q. EXTENDING THE CENTERLINE

The maintaining of the centerline of an entry is aided by the spad and the pointer spad. As shown in Figure MM-5, a wire with a weight attached to it is suspended from both the spad and the pointer spad, providing a line of sight. The mine official sights along this line and directs the assistant at the face to mark a line on the face, in line with the line of sight as the projection of the centerline on the face. Several points are also marked on the roof along the line of sight by the assistant. The straight line connecting these points along the roof to the face is painted and is the centerline of the entry. At the face, from the centerline, half the width of the entry is measured and marked on both sides of the centerline to indicate the extent to which coal can be cut.

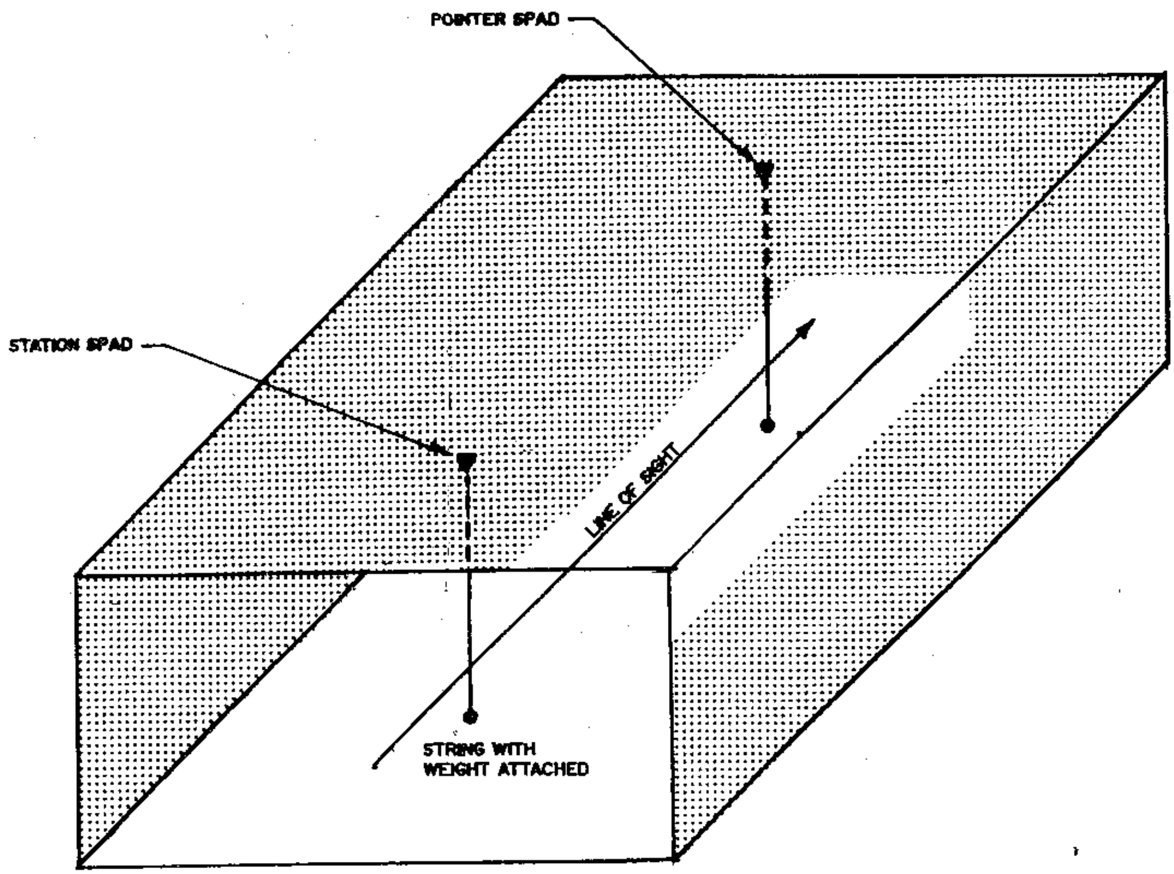


Figure MM-5: Extending the Centerline
[Source: Coal Mining Reference Book, *Fifth Edition,* Revised
Kentucky Mining Institute, 2002]

R. TURNING A PLACE WITHOUT A TRANSIT

It is convenient to have a survey crew to locate the centers for turning places. However, this is not always possible and the mine official has to locate the center at various angles from the existing centerline in an entry. Knowledge of simple trigonometry functions [Sine, Cosine and Tangent] and access to a string and chalk are all that are necessary to locate the center of the new place at any angle. In the Figure MM-6, the trigonometric functions of the angle θ are $\sin\theta = [Y/r]$, $\cos\theta = [X/r]$ and $\tan\theta = [Y/X]$.

Theory: Assume in the Figure MM-6 that the intersection of the two-axes [0,0] is the center of the entry which runs east-west and has advanced sufficiently to the east. It is necessary to drive an entry at an angle [θ] from [0,0] as shown. Consider a distance X along the entry from the intersection [0,0]. From the trigonometric functions, we can calculate, Y = X Tanθ, and r = [X/Cosθ]. Knowing Y and r, the direction of the entry can be easily fixed.

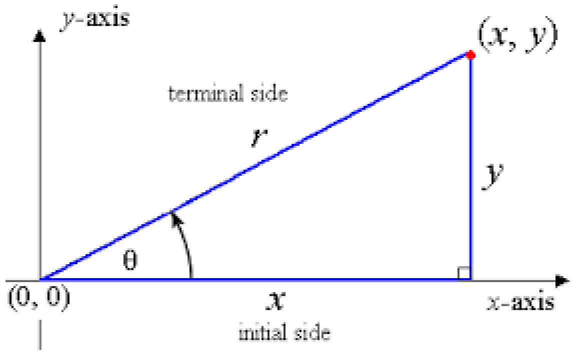


Figure MM-6: Trigonometric Functions

<u>Practice</u>:

In underground, first we would mark off a distance X from the point [0,0] along the existing centerline. With this point as the center, we would take our string and chalk, and draw an arc of radius Y, in a direction perpendicular to the existing centerline. We would then take our string and chalk, and with [0.0] as the center, draw an arc of radius r in the general direction of the proposed entry to intersect the previous arc, the point of intersection being [X,Y]. The line connecting the existing point [0,0] and the new point [X,Y] is the centerline of the new entry at angle θ. Project this line to the face to mark both the center line and the limits of the face on either side of this center line.

It can be readily seen that if a 90 degree turn were required [0,0], we can draw an arc of radius y from [0,0] in a perpendicular direction to the centerline and an arc of radius r, from the point at distance X from centerline to intersect the first arc. The line connecting the point of intersection and [0.0] point will be the centerline of the 90 degree turn. In fact, it will be the Y-axis. In the following, examples of the procedures of turning angles of 30, 60, 45 and 90 degrees are abstracted from the reference: Coal Mining Reference Book, *Fifth Edition*, Revised, Kentucky Mining Institute, 2002.

1. 30 DEGREE ANGLE TURN

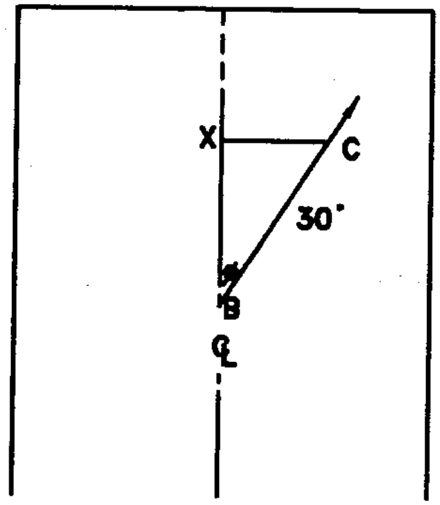


Figure MM-7: 30-Degree Angle Turn

In Figure MM-7, assume that the new entry has to start at point B on the existing centerline and has to be at an angle of 30 degrees to it. Starting at B measure a distance of approximately 7 ft. along the existing centerline and locate the point X. Note that XC = 7 x Tan[30] = 7 x 0.5774 = 4.0 ft., and BC = [7/(Cos[30])] = [7/0.866] = 8 ft. With X as center, draw an arc, 4 ft. in radius, perpendicular to the existing centerline. Then, with B as the center, draw an arc 8 ft. in radius, to intersect the previous arc at point C. Connect B and C giving the centerline of the entry at 30 degrees to the existing entry. Mark the centerline and ribs of the new face on the rib of the existing entry.

2. 45 DEGREE ANGLE TURN

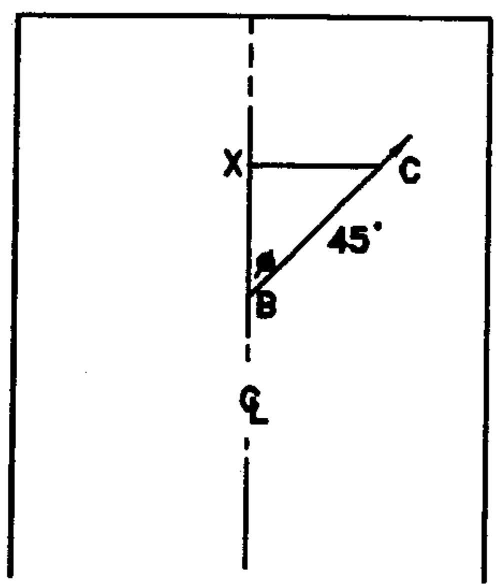


Figure MM-8: 45-Degree Angle Turn

In Figure MM-8, assume that the distance BX is approximately 5 ft. Note that Tan [45] = 1.0 and Cos [45] = 0.7071. As before, calculate the distances XC and BC. In fact, XC is 5 ft. and BC = 7 ft. With X as center, draw an arc, 5 ft. in radius, perpendicular to the existing centerline. Then, with B as the center, draw an arc 7 ft. in radius, to intersect the previous arc at point C. Connect B and C giving the centerline of the entry at 45 degrees to the existing entry. Mark the centerline and ribs of the new face on the rib of the existing entry.

3. 60 DEGREE ANGLE TURN

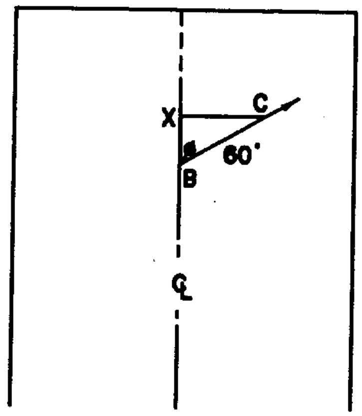


Figure MM-9: 60-Degree Angle Turn

Assume that the distance BX is approximately 4 ft. Note that Tan [60] = 1.732 and Cos [60] = 0.5. The distances XC and BC are therefore approximately 7 ft. and 8 ft. respectively. As before, the point C can be located at the intersection of the two arcs, 7 ft. and 8 ft. in radius and the centerline of the 60 degree angled entry can be marked.

4. 90 DEGREE ANGLE TURN

A number of methods is available for a 90 degree turn. One method was described in the very beginning of this section of turning angles. Here, the description of yet another simple method follows utilizing the 3-4-5 sides' rule of a right angled triangle.

As shown in Figure MM-10, consider that an entry at 90 degrees to the present entry is to be driven and that the new entry's centerline is on the point X on the centerline of the present entry.

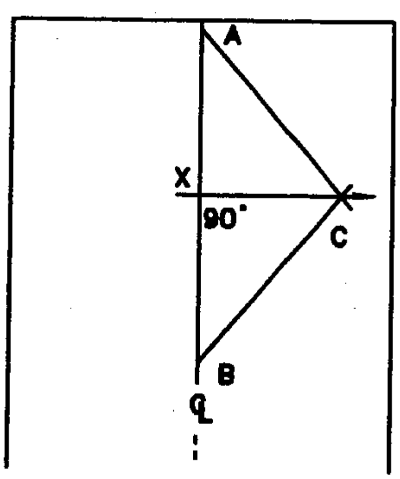


Figure MM-10: 90-Degree Angle Turn

From point X, mark off points A and B on the centerline and on either side of X and equidistant from X. For example, XA = XB = 3 ft. With A as the center, draw an arc, about 5 ft. in radius, in a direction perpendicular to the centerline at X. With B as the center, draw an arc, about the same length in radius, in a direction perpendicular to the centerline at X, intersecting the previous arc at C. The line connecting X and C is the centerline of the new entry. From the new centerline XC, measure and mark the centerline and the ribs of the new entry.

TABLE OF CONTENTS

VENTILATION SECTION 1

Common Formulas

The following tabulation provides some fundamental equations by which many basic mathematic problems can be solved.

FUNDAMENTAL MATH EQUATIONS

Hypotenuse of a Right Triangle	$C = \sqrt{A^2 + B^2}$
Circumference of a Circle	$C = D\ x\ \pi$
Area of a Circle.	$A = Area = \frac{\pi * D^2}{4}$ Or $Area = \pi\ x\ R^2$ Or $Area = D^2 *\ 0.7854$
Area of a Rectangle.	$A = H\ x\ W$
Area of the base of a Solid.	$A = \frac{\pi * D^2}{4}$
Surface of a Sphere.	$\pi\ x\ D\ x\ D = \pi * D^2$
Area of a Triangle.	$A = \frac{1}{2} B\ x\ H$
Area of a Trapezoid	$A = \frac{T+B}{2}\ x\ H$

EXPLANATION OF SYMBOLS

Formula for the hypotenuse of a right triangle C = Hypotenuse

Formula for the circumference of a circle and all formulas for determining the area of a solid

C = Circumference D=Diameter π = 3.1416 A = Area H = Height

W = Width B = Base T = Top B = Bottom

MINE VENTILATION

This chapter on mine ventilation has five sections. In the first section [Section 1], an introduction to several factors in mine ventilation and several fundamental principles of ventilation are outlined. In Section 2, several specific ventilation formulas and their relationships to the factors in mine ventilation are discussed. In Section 3, the structures and equipment that are used in mine ventilation are described. Tools, techniques and methods of making ventilation measurements are described in Section 4. In Section 5, the ventilation plans of room and pillar and longwall sections are presented.

Section 1

PURPOSE AND PRINCIPLES MINE VENTILATION

A. PURPOSE OF VENTILATION

1. PURPOSE OF VENTILATION

Ventilation of a mine means the supplying of a current of air to the various parts of the mine. Ventilation has a double purpose: (a) to dilute, render harmless, and carry off the accumulations of dangerous gases and dusts from the mine; (b) to supply fresh or uncontaminated air to the men working in the mine.

Miners are well aware of the necessity of supplying sufficient air to remove explosive mine gases, chiefly Methane, under normal working conditions, or to dilute such gases well below their lower explosive limits. An equally important reason for adequate ventilation is the dilution of noxious or toxic gases, such as Carbon Monoxide, Carbon Dioxide, Hydrogen Sulfide, Sulfur Dioxide, or the Oxides of Nitrogen, below the point where they may affect the health of the miners. Dusts, which may have an injurious effect on the health of underground workers, constitute a hazard which adequate ventilation may reduce. Still another health hazard which proper ventilation may reduce is the high-humidity, high-temperature condition which sometimes prevails in mines; when coupled with small quantities of toxic or noxious gases and either large or small amounts of dusts, temperature and humidity factors may contribute to the ill health of workers and lower their working efficiency. At times of emergency, such as during fires and after explosions, the air flow to the affected parts of the mine can be increased or decreased as conditions may require. In summary ventilation has far reaching effects on the safety, health, production and cost factors of mining coal.

2. THE VENTILATING CURRENT

The body of air which is moved through a mine is called the ventilating current. In order to have a current of air flow from the surface, through the mine, and back to the surface, we must have at least one continuous passage through the mine. Since a mine will have many air passages in each working section, there must be at least two intake passages and one return passage for the air in each section. The mine entries which are the passages for air also called as airways – intake airways and return airways. The air current travel is controlled by means of doors, stoppings, overcasts, and other devices. These are called air flow control devices and are discussed in Section 3.

3. FACTORS IN CIRCULATION OF AIR CURRENTS

The general principles of mine ventilation are as follows. Airways offer resistance to the flow of air. Therefore, to circulate an air current through the mine airways, power must be used to overcome the resistance of the airways. In order to circulate to circulate a certain quantity air, a certain amount of power has to be used. This application of power, in turn, produces a certain velocity and pressure. Also, as air is circulated in the airway, this also results in a certain amount of work.

We divide the above mentioned factors into three groups: Group A includes only the producing factor or the power required; Group B includes the resisting factors of rubbing surface and the unit of resistance for each square foot of that surface; Group C includes the resulting factors of velocity, quantity and pressure of the air current and the work done at a certain velocity, pressure and quantity.

B. VENTILATING PRESSURE

1. PRESSURE AND THE AIR CURRENT

Air, like any other fluid body, will always move from a point of high pressure to a point of lower pressure. If we had a tube filled with air and applied pressure to the air at one end of the tube, the air would flow through the tube towards the other end and a current of air would be established. The same result could be obtained by lowering the pressure on the air at one end of the tube; the air would flow into the tube at the higher pressure end and flow through the tube to the lower pressure end.

2. TOTAL AND UNIT VENTILATING PRESSURE.

The pressure causing an air current to flow is the difference between the pressures at the intake and return ends of the tube or the passage. The pressure at the intake end is always greater than that at the return end. The difference between these pressures is called the ventilating pressure.

The unit ventilating pressure is the amount of pressure per square foot of an airway's cross-section by which the intake pressure exceeds the return pressure. The product of the unit pressure, usually designated by p, and the area of the airway, designated by a, equals the force, F.

$$F = pa$$

Example MV-1: If an airway is 8 feet wide and 6 feet high, and the unit pressure is 5 pounds per square foot, what is the force?

Solution:

$$F = pa$$
$$F = p\,(5) x\ a\,(8\ x\ 6)$$
$$= 5\ x\ 48 = 240\ lbs$$

The force creating an air current can be appreciated every time a door between an intake airway and return airway is opened. In order to open the door the miner must overcome the total force being exerted on the door.

Example MV-2: If the unit pressure is 10 pounds per square foot and the door measures 5 by 8 feet, what force is minimum force is needed to open the door?

Solution: The force needed should exceed the force on the door. The force on the door is:

$$F = pa$$
$$F = p\,(10) x\; a\;(5x\; 8)$$
10 x 5 x 8 = 400 lbs.

The force needed should exceed 400 lbs.

3. MEASUREMENT OF AIR PRESSURES

The unit of ventilating pressure, which has been given above in pounds per square foot, is measured by the water column which it will support. If we take a box in the form of a cube, the water will weigh 62.5 pounds. The pressure on the bottom of the completely filled box will be 62.5 pounds per square foot, i.e. the pressure per square foot for each 12 inches of water column in 62.5 lbs.

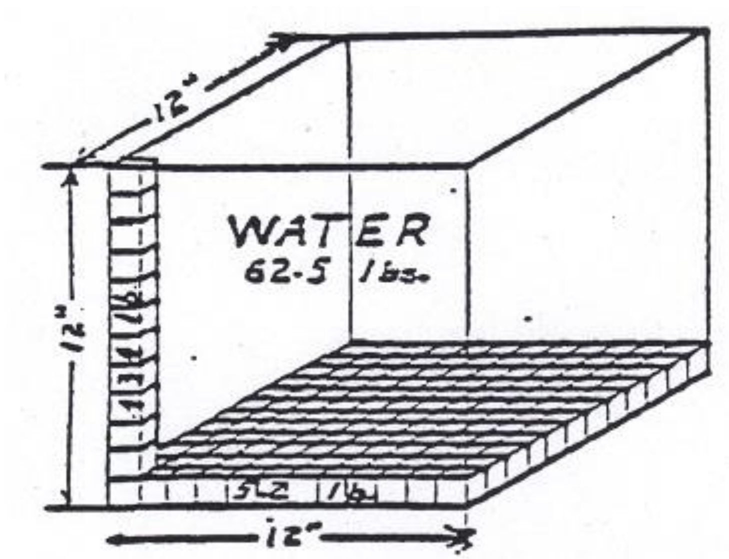


Figure MV1-1

If the water column were one-inch-high, the pressure would be 62.5/12 or 5.2 lbs. per square foot. If we take the weight of one foot of water column over one square inch of bottom, the pressure is 62.5/144 or 0.434 lbs. per square inch. The relationship between, pressure *p* and water gage *i*, is:

$$p = i \textbf{ x } \mathbf{5.2}$$

$$i = \frac{\boldsymbol{p}}{\mathbf{5.2}}$$

Example MV-3: The water gauge reading is 2 inches of water. What is the unit pressure in lbs. per sq. ft.?

Solution:

$$p = i \; x \; 5.2$$
$$= 2 \; x \; 5.2$$
$$= 10.4 \; lbs.per.sq.ft.(psf)$$

Force exerted on a unit area by a column of air is referred as air pressure or absolute pressure. The pressure caused on a point on earth by the weight of the air column above that point is called the Atmospheric Pressure. Atmospheric pressure is measured by a mercury barometer. The most common type of mercury barometer is known as cistern barometer consists of a cistern and a tube filled with mercury [Section 4]. As the atmospheric pressure increases or decreases, the height of the mercury level in the tube changes. At sea level and at a temperature of 85.4°F, the height of the mercury column over the mercury in the cistern is 29.92 in. This is equal to an air pressure of 14.70 lbs. per square inch [psi].

Pressure difference or differential pressure is the difference in air pressure between two points of interest. In mine ventilation, we are often interested in the pressure difference of the air on the two sides of a door or stopping or in the fan drift and the outside atmosphere. The differential pressure is measured with the help of a manometer [Section 4].

C. VELOCITY AND QUANTITY OF AIR

1. VELOCITY OF AIR

The velocity of an air current is its speed or rate of travel in a given direction. For example, if a handful of fine dust is thrown into a current of air and it is found that the particles of dust have been carried 100 feet in 10 seconds, the velocity of the air is also 100 feet in 10 seconds or is 10 feet per second [100 feet/10 seconds = 10 feet/second]. Usually air velocity is given in feet per minute and in this case would be 600 ft. per min. [10 feet per second x 60 seconds = 600 fpm].

2. QUANTITY OF AIR

The quantity of air passing through an airway in a certain length of time is the product of the velocity and the cross-sectional area of the airway.

$$q = v\ x\ a = a\ x\ v = av = va$$

where q = quantity of air in cu. ft. per min. [cfm]
v = velocity of the air ft. per min [fpm]
a = area of the airway or entry in sq. ft.

Example MV-4: If an airway, 8 x 6 feet, is passing air at a velocity of 600 ft. per min., what is the quantity of air being passed?

Solution:

$$q\ (Quantity)\ =\ v\ (velocity)\ x\ a\ (area\ of\ cross - section)$$
$$q\ =\ 600\ x\ (8\ x\ 6)\ =\ 600\ x\ 48\ =\ 28{,}800\ cu.ft.per\ min.$$

In passing through a mine, air becomes warmer or colder, depending on the difference in temperature between the outside and inside of the mine. The intake air, then, may decrease or increase in volume as it travels towards the return depending on whether it becomes, respectively, colder or warmer. The return air is variable in quantity, due not only to the temperature changes but also to additions of mine gases and to decreased pressure on the air as it nears the return end. Therefore, in discussing the quantity of air circulating in a mine, the quantity of intake air is always used unless the problem specifies the return air in its calculations.

The most common instrument used to measure the velocity of air in a mine is called an anemometer. There are several manufacturers of anemometers and there are several other instruments for measuring velocity. These are discussed in Section 4.

3. VELOCITY PRESSURE

It is well known that a blowing wind due to its velocity exerts pressure in the direction of airflow, and that this pressure increases non-linearly with the increase in air velocity. This pressure is called the velocity pressure (or kinematic pressure) of the air and in ventilation calculations, an approximate formula for velocity pressure is:

$$P_v = 5.2\ \left[\frac{v}{4000}\right]^2$$

$$i_v = \left[\frac{v}{4000}\right]^2$$

Where P_V is the velocity pressure in lbs. per sq. ft. [psf]

V is the velocity in ft. per min.[fpm]
i_V is the velocity pressure in inches of water gauge

Example MV-5: If the air flowing in a mine shaft has an average velocity of 1,800 ft. per min. at a particular cross-section, what is the velocity pressure of this air at that cross-section in psf and in inches of water?

Solution:

$$P_v = 5.2 \left[\frac{v}{4000}\right]^2$$

$$P_v = 5.2 \left[\frac{1800}{4000}\right]^2$$

$P_v = 1.053$ lbs. per sq. ft. [psf] or 0.2025 inches of water gauge.

$i_v = [P_v/5.2] = [1.053/5.2] = 0.2025$ inches of water.

Alternatively, we can use the formula for inches of water, as shown below.

$$i_v = \left[\frac{v}{4000}\right]^2$$

Substituting for velocity, *v = 1800*, in this formula, we get,

$$i_v = \left[\frac{1800}{4000}\right]^2$$

It can be easily seen that $i_v = 0.2025\ inches\ of\ water$.

Example MV-6: In the above example, consider that there is a bunton fitted across the cross-section of the shaft and the bunton is 10 ft. long and 6 inches wide. What is the force exerted by the air on the bunton?

Solution:

Area of bunton = $10\ ft.x\ 0.5\ ft. = \ 5\ sq.ft.$
Force = area x pressure = $5\ x\ 1.053\ =\ 5.265\ lbs.$

Note: A bunton is one of a number of struts reinforcing the walls of a shaft and dividing it into vertical compartments.

D. AIRWAYS AND RUBBING SURFACE

1. FORM AND SIZE OF AIRWAYS.

There are five forms of airway which are common to mines. These are: circular or round, square, rectangular, arched, and trapezoid airways. These forms should be familiar to any student who has studied arithmetic.

The perimeter of an airway is the distance around the airway and is measured in feet. Thus, a circular airway's perimeter will equal its circumference, which is π times diameter. The perimeter of a square airway equals four times the width. The perimeter of a rectangular airway equals twice the sum of the height and the width. The perimeter of an arched airway equals half the circumference of a circle plus twice the side height to the beginning of the arch. The perimeter of a trapezoidal airway equals twice the slant height of the side plus the top width plus the bottom width.

The area of an airway means the area of the cross-section and it is measured in square feet. The area of a circular airway equals $\frac{\pi}{4}$ times diameter squared. The area of a square airway equals the width squared. The area of a rectangular airway equals the product of the width times the rib height. The area of a trapezoidal airway equals half the sum of the top and bottom widths times the vertical height.

The rubbing surface of an airway equals the perimeter times the length of the airway. It is the entire inner surface of the airway under consideration and is given in square feet of surface.

Example MV-7: If an airway has dimensions of 8 by 10 feet (8 ft. high and 10 ft. wide) and is 3,000 feet long, calculate the rubbing surface?

Solution:

Rubbing surface (S) = perimeter (o) x length (l)

$$(S) = 2(8 + 10)\, x\, 3{,}000 = 108{,}000\, sq.ft.$$

2. COMPARISON OF AIRWAYS

There is a difference in the amount of resistance offered to the passage of an air current by the different forms of airways. Resistance affects the power necessary to pass the air, and the airway having the least natural resistance (discounting timber obstructions) will be the most economical for ventilation purposes. The resistance depends on several factors, one being the amount and character of the rubbing surface. The unit resistance (the resistance a square foot of surface to an air current with a velocity of one foot per minute) is termed coefficient of friction and is designed by letter 'k' (lb. min.^{2}/ft.^{4}). In mine ventilation, it is common to refer the coefficient of friction as "friction factor of the airway." Airways having the same coefficient of friction and length will have lesser or greater resistance, depending on the perimeter, for the same cross-sectional area. These factors are shown in the Table MV-1 which follows.

From this table it can be seen that where the cross-sectional areas are the same, the airway with the least perimeter will have the least rubbing surface (and least resistance) per thousand feet. The circular airway is the most efficient in this respect, followed by the arched airway then by the square, the rectangular, and the trapezoidal airways in that order. As the rectangular airway's dimensions approach those of the square airway, as in the second case, the rubbing surface becomes a minimum for that type.

Comparison of Airway Forms

Form of Section	Dimensions in feet	Length in feet	Perimeter in feet	Rubbing surface in square feet	Area in square feet
Circular Arched	11.23 dia. 9 wide 12.08 high	1,000 1,000	35.44 38.30	35,440 38,300	100 100
Square Rectangular	10 x 10 8 high 12.5 wide	1,000 1,000	40.00 41.00	40,000 41,000	100 100
Rectangular	9 high 11.11 wide	1,000	40.22	40,220	100
Trapezoidal	8 high (10 and 15) wide	1,000	41.76	41,760	100

In practice, mining conditions may influence the form of airway driven. Shafts may be circular or ovaloid. Occasionally, slopes may be arched. Square and rectangular airways are common. The anthracite mines of Pennsylvania probably have more different forms of airways than most bituminous mines. In addition to four of the common forms, anthracite mines may have airways in the form of rhomboids, rhombuses and trapeziums.

E. MINE RESISTANCE

1. RESISTANCE TO AN AIR CURRENT.

Every passage through which an air current is flowing offers resistance to the air current, regardless of whether the passage is smoothly lined or with rough sides. This resistance is due to the rubbing of the air current on the sides, top, and bottom of the passage; the air particles strike the walls and rebound into the path of other air particles, thus causing confusion and reducing the velocity of the air flowing near the walls. This resistance, then, must be recognized as entirely frictional resistance. The pressure that is applied to the passage overcomes the effect of this frictional resistance and other resistances that may exist for flow.

If the velocity of the air current increases, the air particles rebound farther from the wall, interfere with other air particles farther away from the wall or nearer the center of the passage, and tend to reduce the air velocity over a larger section of the airway's area. Therefore, more pressure or power will be required to maintain the velocity. As resistance increases, or power and pressure must increase to maintain the same velocity. Likewise, as resistance decreases, velocity may be increased with the same pressure, or the same velocity may be had for less pressure.

2. UNIT RESISTANCE AND THE RESISTANCE FORMULA

The resistance of an airway determines the pressure necessary to generate a certain velocity of air through the airway. The unit resistance (resistance per square foot of surface to an air current with a velocity of one foot per minute) is usually termed the coefficient of friction and is designed by the letter 'k' (lb. min.^{2}/ft.^{4}).

As the velocity of the air current increases, each particle of air strikes the rubbing surface with more force, and more particle of air are available for striking the rubbing surface. If we double the velocity, the air particle strike twice as hard and there are twice as many air particles to strike, so the resistance to the flow of air will be 2 x 2 or 4 times as great as before the velocity was increased. If the velocity were tripled, the resistance would be increased 9 times. The force that is required to create a velocity, V can be found by the following formula:

$$Force\ (F) = ksv^2$$

3. CALCULATION OF RESISTANCE AND VENTILATING PRESSURES

In order for a current of air to flow through a passage, the force applied to course the air through the airway must be equal to the force necessary to overcome the resistance of the airway. When this condition is reached, the velocity will be uniform. The calculations involving the factors of unit pressure p, area a, velocity v, water gauge i, and coefficient of friction k are as follows:

$$F = Pa = ksv^2$$

$$Pa = ksv^2$$

$$p = \frac{ksv^2}{a}$$

Remembering $s = length\ x\ area = l\ x\ o = lo, q = va$, and substituting for s and v,

$$P = \frac{ksv^2}{a}$$

$$= \frac{kloq^2}{a^3}$$

For a given airway, k, l, o and a are known. Therefore, the quantity $\frac{klo}{a^3}$ is a constant and can be calculated. This is called the airway resistance, R which has the units of lb. min.^{2}/ft.^{8}

$$R = \frac{klo}{a^3}$$

The resistance, pressure, and quantity relationship can now be stated as:

$$P = Rq^2.$$

In mining, it is common to express the pressure in in. of water

$$i = \frac{p}{5.2}$$

$$i = \frac{kloq^2}{5.2a^3}$$

Therefore, resistance can also be defined as $R = \frac{klo}{5.2a^3}$ with the units of $\frac{in.(min.)^2}{(ft.)^6}$.

Note that, as before,

$$i = Rq^2$$

The use of these factors and formulas in a mine ventilation example is illustrated in the following example.

Example MV-8: An airway, 8 feet by 10 feet, 3,000 feet long, is passing a current of air at a velocity of 500 feet per minute.

[1] Find the applied force and unit ventilating pressure, the water gauge, and the mine resistances in the two units using 2 x 10^{-8} lb. min.^{2}/ft.^{4} for k.

[2] Verify the values for ventilating pressure and water gauge using the calculated resistances.

Solution: First calculate all the relevant values for area, perimeter, and rubbing surface of the airway:

a = 8 x 10 = 80 sq. ft.
o = 2(8 + 10) = 36 feet.
s = 3,000 x 36 = 108,000 square feet.
k = 2 x 10^{-8} lb. min^2/ft.^{4}

Substituting these values in the formula for calculating pressure, $p = \frac{ksv^2}{a}$

$$p = \frac{0.00000002 \; x \; 108{,}000 \; x \; 500 \; x \; 500}{80}$$

$$= 6.75 \; lbs. per \; sq. ft.$$

Calculate the force as pressure x area: $F = p \; x \; a$

$$F = 6.75\ x\ 80 = 540\ lbs.$$

Calculate pressure in inches of water:

$$i = \frac{6.75}{5.2} = 1.3\ inches.$$

Calculate Resistance in $\frac{lb.min.^2}{ft.^8}$

$$R\ =\ mine\ resistance\ in \frac{lb.min.^2}{ft.^8}$$

$$R\ = \frac{klo}{a^3}$$

$$= \frac{0.00000002\ x\ 3{,}000\ x\ 2(8\ +\ 10)}{80\ x\ 80\ x\ 80}$$

$$= 0.0000000042 \frac{min.^2}{ft.^8}$$

Calculate Resistance in [in.min.^{2}/ft.^{6}]

$$R\ =\ mine\ resistance\ in \frac{in.min.^2}{ft.}$$

$$R\ = \frac{klo}{5.2a^3}$$

$$R\ = \frac{0.00000002\ x\ 3000\ x\ 2(8\ +\ 10)}{5.2\ x\ 80\ x\ 80\ x\ 80}$$

$$R\ =\ 8.076\ x\ (10^{-10}) \frac{in.min.^2}{ft}.^6$$

Calculate the quantity flowing from area and velocity

$$Q\ =\ 500\ x\ 80\ =\ 40{,}000\ cu.ft.per\ \mathrm{min}[cfm]$$

Calculate the pressure from the equation:

$$P\ =\ Rq^2$$

$$= 0.0000000042\ x\ 40{,}000\ x\ 40{,}000$$

$$= 6.72\ lbs.per\ sq.ft.$$

$$i = Rq^2$$

$$= 8.076 \, x \, 10^{-10} \, x \, 40{,}000 \, x \, 40{,}000$$

$$= 1.29 \, inches$$

Because the values of k and R are very small, there are small round-off errors. The answers are in good agreement.

F. POWER AND WORK IN PRODUCING AN AIR CURRENT

1. DEFINITION OF WORK

If a force acts through a certain distance, work if performed. The quantity or amount of work done equals the value of the force, in pounds, times the distance through which it is moved, in feet. A force of 100 pounds acting on a body may push it 200 feet; the work done equals 100 x 200 or 20,000 foot-pounds. Lifting a weight through a vertical distance likewise produces work just as work is done in exerting a force through a given distance.

The time involved in doing work does not affect the final result. In the foregoing example, the work done would have been the same regardless of whether it took one minute, one hour, or one year to complete the movement.

2. DEFINITION OF POWER AND HORSEPOWER

Time is involved in power calculations and power relates to the performance of a given work in a given time. A certain amount of work which can be done by a certain force in a certain length of time can be done in half that time if the force is doubled or if a second force has twice the power of the initial force. In comparing machines we must compare their ability to do the same amount of work by the time it takes each machine.

The work done in pushing a weight of one pound through a distance of one foot in one minute is called the unit of power in ventilation. The value of this unit, designated by u, is given in foot-pounds per minute (ft.-lbs. per minute).

The unit of power is usually too small to be used in ventilation calculations. Therefore, a larger unit, called the horsepower, is used instead. One horsepower, usually designated as hp, equals 33,000 ft.-lbs. per minute. A one hp. machine, in theory, can move 33,000 pounds one foot vertically in one minute, or 330 pounds 100 feet vertically in one minute, or 33 pounds 1,000 feet vertically in one minute.

3. CALCULATION OF WORK AND POWER IN VENTILATION

The work performed each minute by a current of air with a velocity of a certain number of feet per minute is equal to the product of the force causing the flow and the velocity. The formula expressing this relation is,

$$u = Fv$$

Or substituting $F = pa$, $$u = pav$$

And, substituting ksv^2 for pa, $$u = v(ksv^2) = ksv^3$$

Since 'u' is the work performed each minute, it is equal to the power on the air.

Example MV-9: An airway, 6 x 12 feet, is passing air at a velocity of 550 ft. per minute. The ventilating pressure is 7 lbs. per square foot. Find the power on the air.

Solution:

$$a = 6\ x\ 12 = 72\ sq.ft.$$

$$u = pav$$

$$= 7\ x\ 72\ x\ 550 = 277{,}200\ ft.-lbs.per\ minute.$$

$$h.p. = \frac{277{,}200}{33{,}000} = 8.4\ horsepower.$$

Example MV-10: Find the power necessary to pass air at a velocity of 550 ft. per minute through an airway, 6 x 12 ft. and 3,000 feet long, and coefficient of friction 0.02 x 10^{-7} lb.-min.^{2}/ft.^{4}.

Solution:

$$o = 2(6 + 12) = 36\ ft.$$

$$k = 0.02\ x\ 10^{-7}\ \frac{lb.min.^2}{ft.^4}$$

$$s = 3{,}000\ x\ 36 = 108{,}000\ sq.ft.$$

$$u = ksv^3$$

$$= 0.02\ x\ 10^{-7} x\ 108{,}000\ x\ 550^3$$

$$= 359{,}370\ ft.-lbs.per\ minute$$

$$h.p. = \frac{359{,}370}{33{,}000} = 10.89\ horsepower.$$

If the fundamental formulas are kept in mind, it is possible to substitute certain values from these in another formula to solve a problem giving only a few values. For instance, we could rearrange our formulas as follows:

$$u = pav$$

Or since $av = q$, then

$$u = pq$$

And $h.p. = \frac{pq}{33{,}000} = \frac{Rq^3}{33{,}000} = \frac{kloq^3}{5.2a^3}$

Example MV-11: In a mine 60,000 cu. ft. of air is being circulated at a 2-inch water gauge. What is the hp?

$$p = 5.2i = 5.2\ x\ 2 = 10.4\ lbs.per\ sq.ft.$$

$$u = pq = 10.4\ x\ 60{,}000 = 624{,}000\ ft.-lbs.per\ min.$$

$$h.p. = \frac{624{,}000}{33{,}000} = 18.909\ horsepower.$$

G. VENTILATION CALCULATIONS

1. ACCURACY in CALCULATIONS

Unless there is some particular reason for extreme accuracy in making ventilation calculations, the student may abide by the following rules with regard to accuracy:

Quantity or volume of air should be calculated to the nearest hundred cubic feet; that is 91,370 cu. ft. should be taken as 91,400 cu. ft. Actual measurement of volume in practice can hardly be estimated closer than this.

The same degree of accuracy is allowable with the rubbing surface. If an airway measures 6 x 8 ft. and 2235 ft. long, the calculation of the surface will be 6 x 8 x 2235 or 107,280 sq. ft. This should be taken as 107,300 sq. ft. Actual surface could be varied as much as 1% without affecting the practical value of the calculations.

Velocity readings should be read to feet per minute without decimals. Water gauge readings should be calculated to hundredths of an inch, or to two decimal places; actual readings, unless a draught gauge is used, can hardly be read closer than tenths of an inch. Pressures per square foot should be carried to two decimal places if the pressure exceeds 10 pounds per square foot.

Horsepower readings of less than 100 horsepower should be carried to three decimal places; if more than 100 horsepower, two decimal places are sufficient. In discarding figures, as in the case of carrying to two decimal places, if the third place figure is less than five, drop it and make no change in the second place figure. If the third place figure exceeds five, drop it but add one to the second place figure. The same applies to the third and fourth place figures where three decimal figures are desired.

TABLE OF CONTENTS

VENTILATION SECTION 2

SECTION 2

VENTILATION FORMULAS AND RELATIONS

A. FUNDAMENTAL VENTILATION FORMULAS

In the study of fundamental principles of ventilation there were certain formulas given. The importance of knowing these fundamental formulas lies in the fact that from them are derived all of the ventilation formulas used in calculations. A list of the fundamental formulas and the units associated with them are given in (Table MV-2). The definition of each of the symbols used in the formulas and the unit associated with each symbols are as follows.

a = sectional area of airway, in square feet $(ft.)^2$
l = length of airway, in feet $(ft.)$
w = width of airway, in feet $(ft.)$
h = height of airway, in feet $(ft.)$
o = perimeter of airway, in feet $(ft.)$
s = rubbing surface, in square feet, sq. ft.,$((ft.)^2)$
v = velocity of air current, in feet per minute, ft. per min., (fpm)
q = quantity of air, in cubic feet per minute, cu. ft. per min., (cfm)
F = applied force, in pounds $(lbs.)$
p = unit ventilating pressure, in pounds per square feet $(\frac{lb}{(ft.)^2})$
i = inches of water gauge; also given as w.g.
k = coefficient of friction $(\frac{lb.(min.)^2}{(ft.)^4})$
R = total resistance of an airway $(\frac{lb.(min.)^2}{(ft.)^8})$
R = total resistance of an airway$\frac{in.(\text{min.})^2}{(\text{ft.})^6}$
u = units of work, in foot-pounds per minute$(\frac{ft.lb.}{min.})$
h = horsepower; also given as hp. or H.P.

Table MV-2

FUNDAMENTAL VENTILATION FORMULAS

	Value to be found (unit)	Equation
(1)	Force (lbs.)	$F = pa$
(2)	Unit Pressure $(\frac{lb}{(ft.)^2})$	$p = \frac{ksv^2}{a}$
(3)	Unit Pressure $(\frac{lb}{(ft.)^2})$	$p = \frac{ksq^2}{a^3}$
(4)	Water Gauge (inches, w.g.)	$i = \frac{p}{5.2}$
(5)	Quantity (cfm)	$q = av$
(6)	Perimeter, rectangular airway $(ft.)$	$0 = 2\,(w + h)$
(7)	Rubbing Surface $(ft.^2)$	$s = lo$
(8)	Resistance $(\frac{lb.(min.)^2}{(ft.)^8})$	$R = \frac{klo}{a^3}$
(9)	Resistance$(\frac{in.(\mathrm{min.})^2}{(\mathrm{ft.})^6})$	$R = \frac{klo}{5.2\ x\ a^3}$
(10)	Velocity Pressure $(\frac{lb}{(ft.)^2})$	$p_v = 5.2\left[\frac{v}{4000}\right]^2$
(11)	Velocity Pressure (inches, w.g.)	$p_v = \left[\frac{v}{4000}\right]^2$
(12)	Units of Power$(\frac{ft.lb.}{min.})$	$u = pav$
(13)	Units of Power $(\frac{ft.lb.}{min.})$	$u = ksv^3$
(14)	Units of Power $(\frac{ft.lb.}{min.})$	$u = \frac{ksq^3}{a^3}$
(15)	Units of Power (hp. or H.P.)	$h = \frac{u}{33{,}000}$

In the following sub-sections of this section, the use of these formulas for comparing two or more different ventilation systems is illustrated. For example, the airway characteristics such as surface roughness, area, length, etc. determine the ventilation parameters of pressure and power required to pass a certain quantity of air. It is well known that the mine ventilation system changes with time due to such factors as changing length of airways, or increasing or decreasing quantity of air in the various workings. Therefore, it is important to know these formulas and their use for comparing different mine airways.

A simple method for comparing two airways involves the solving of formulas representing conditions in each airway at the same time, eliminating those factors which do not change or which are equal in value.

B. PRESSURE RELATIONS IN VENTILATION

All pressure relations can be explained by studying the following pressure formulas:

$$p = \frac{ksv^2}{a} = \frac{klov^2}{a}$$

$$p = \frac{ksq^2}{a^3} = \frac{kloq^2}{a^3}$$

1. Note that for a given airway, the values for k, l, o and a do not change. Therefore, as seen in the equations above relating pressure and velocity, and pressure and quantity, pressure will change only with a change in velocity or quantity. The actual relationship is that pressure required*[p]* increases to the second power of the velocity (v) or the quantity (q) increase, i.e. if the velocity or the quantity is doubled, the pressure will increase four fold.

2. The pressure required to generate a certain velocity of air, v, or to course a certain quantity of air, q, is related to the other factors as follows. The pressure required will increase linearly with increases in

(1) The coefficient of friction or the roughness of the airway (k)
(2) The length of the airway (l)
(3) The perimeter of the airway (o)
(4) The rubbing surface of the airway (s)

Similarly, the pressure required will decrease linearly with decreases in any of the above four factors.

For example, if the length of an airway increases by 10% in length (all other factors remaining unchanged), the pressure required will increase by 10%. Similarly, if the coefficient of friction [k] of an airway is decreased by, say 20% by lining the airway, the pressure required will decrease by 20%.

3. For passing the same quantity of air, according to the second equation above, if the area of the airway is increased, the pressure required will decrease approximately by the third power of the increase in the area of the airway. For example, if the area of an airway can be increased by 10%, the pressure required will decrease to $\left(\frac{1}{1.1}\right)^3 x\, 100\% = 75\%$ of that in the smaller airway. This is the main reason why one would like to have a mine entry as large (wide and high) as possible, limited only by safety considerations.

4. Consider two airways 1 and 2 with the following relationships for pressure and quantity flowing:

$$p_1 = \frac{k_1 l_1 o_1 (q_1)^2}{(a_1)^3}$$

$$p_2 = \frac{k_2 l_2 o_2 (q_2)^2}{(a_2)^3}$$

The general pressure ratio is as follows:

$$\frac{p_1}{p_2} = \frac{k_1}{k_2} \; x \frac{l_1}{l_2} x \frac{o_1}{o_2} x \frac{q_1^2}{q_2^2} x \frac{a_2^3}{a_1^3}$$

Example MV-12: Consider airway 1 which is in a large, cavernous underground limestone mine with the following values:

$$k_1 = 30 \; x \; 10^{-10} \frac{in.\, min.^2}{ft.^6}$$

$$l_1 = 1000 \; ft.$$

$$o_1 = 2 \; x \; (30 + 50) \; ft.$$

$$q_1 = 10{,}000 \frac{ft.^3}{min.}$$

$$a_1 = 1{,}500 \; ft.^2$$

Airway 2 is in an underground coal mine with the following values:

$$k_2 = 60 \; x \; 10^{-10} \frac{in.\, min.^2}{ft.^6}$$

$$l_2 = 2000 \; ft.$$

$$o_2 = 2 \; x \; (6 + 20) \; ft.$$

$$q_2 = 20{,}000 \frac{ft.^3}{min.}$$

$$a_2 = 120 \; ft.^2$$

Calculate the general pressure ratio and comment on the result.

Solution:

$$\frac{p_1}{p_2} = \frac{30}{60} x \frac{1000}{2000} x \frac{160}{52} x \frac{(10000)^2}{(20000)^2} x \frac{(120)^3}{(1500)^3} = 0.000098 = 0.0001$$

The pressure ratio says that the limestone mine requires only [1/10^4] of the pressure required in the coal mine. This example clearly illustrates the advantages of large area, smaller length and smaller k-values.

C. POWER RELATIONS IN MINE VENTILATION

All power relationship can be explained by studying the following formulas:

$$u = \frac{ksq^3}{a^3} = \frac{kloq^3}{a^3}$$

1. For a given airway where k, l, o and a do not change. Therefore, the power required changes with the quantity changes.

a) According to the above equation, power required increases to the third power of the quantity increase. Even for small increases in quantity flowing, this means a large increase in power. For example, all other factors remaining same, if the quantity is to be increased by 10%, the power needs to be increased by $(1.1)^3$ = 1.33 times or 33% more than the current power. If the quantity flowing in the airway is to be increased by 50%, the power will increase to $(1.5)^3 = 3.375$ times or 337.5% of the current power.

b) The power required to course a certain quantity of air, q, is related to the other factors, as follows. The power required will increase linearly with increases in the

(1) coefficient of friction or the roughness of airway (k),
(2) length of airway (l),
(3) perimeter of airway (o), and
(4) rubbing surface (s).

Similarly, the power required will decrease linearly with decreases in any of the above factors. If the length of an airway increases by 20%, the power required will increase by 20%. Similarly, if the roughness of the airway [coefficient of friction] decreases by 20%, the power required will also decrease by 20%.

2. With all other factors not changing in the above equation, it is clear that the power required will decrease approximately by the third power of increase in the area of an airway.

3. Consider two airways, 1 and 2, with the following relationships for the power and quantity flowing:

4.

$$u_1 = \frac{k_1 l_1 o_1 q_1^3}{a_1^3}$$

$$u_2 = \frac{k_2 l_2 o_2 q_2^3}{a_2^3}$$

The general power ratio is stated as follows:

$$\frac{u_1}{u_2} = \frac{k_1}{k_2} x \frac{l_1}{l_2} x \frac{o_1}{o_2} x \frac{q_1^3}{q_2^3} x \frac{a_2^3}{a_1^3}$$

Example MV-13: Consider two airways, 1 and 2, which have the same roughness factor (k), same length (l), and same perimeter (o). The quantity flowing through airway 1 is 20,000 cu. ft. per min. and through airway 2 is 30,000 cu. ft. per min. The areas of airway 1 and airway 2 are respectively 90 sq. ft. and 120 sq. ft. Calculate the ratio of the powers consumed in airways 1 and 2. Comment on the results.

$$\frac{u_1}{u_2} = \frac{k}{k} x \frac{l}{l} x \frac{o}{o} x \frac{20000^3}{30000^3} x \frac{120^3}{90^3} = 1.056$$

The power consumed in airway 1 is about 1.056 times of that consumed in airway 2. Though airway 2 has about 50% more air flowing, its 33% larger area, leads to a lower power consumption. In fact, the two power consumptions are nearly equal.

D. EQUIVALENT ORIFICE

1. DERIVATION

Remember from Section 1, P = Pressure in inches of water, q = quantity flowing in cfm, and $R = mine\ resistance\ in \frac{in.min.^2}{ft.}$, and the equations.

$$R = \frac{klo}{5.2a^3}$$

$$P = Rq^2$$

The last equation basically states that the quantity of air flowing and the pressure on the air are related by the resistance of a passage. This resistance will vary in nearly every airway or mine. The *equivalent orifice* is a term which expresses these relations in terms of the area of an orifice in a thin-plate through which the same air flow causes the same pressure loss.

When a fluid flows through a thin-plate orifice, the effective area is always less than the orifice area, due to the contraction of the fluid stream as it passes through. This contraction is called the contracted vein or vena contracta of the stream. The value for this effective area for the thin-plate orifices with knife edges where the fluid stream is air is 0.62 of the orifice area.

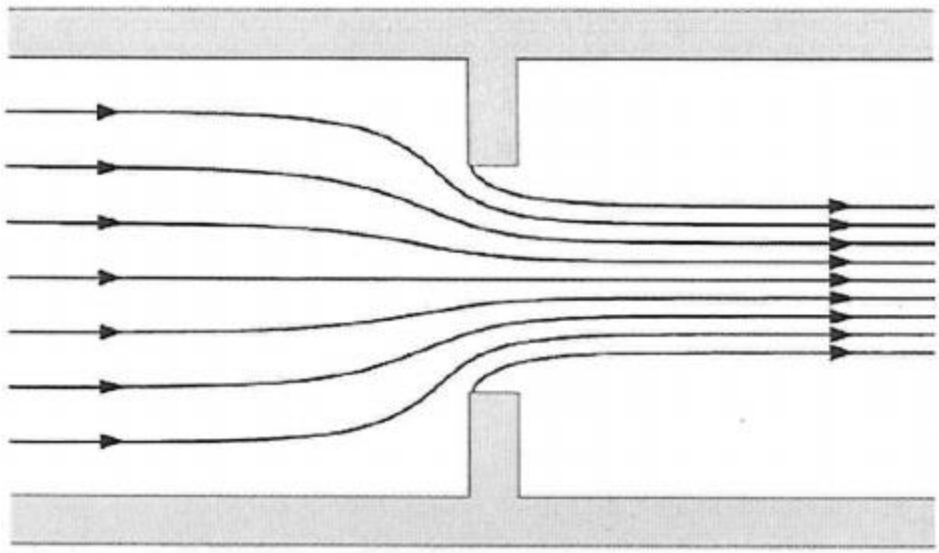


Figure MV-2: Vena Contracta effect on a sharp edged orifice

The Vena Contracta effect is illustrated in Figure MV-2. Note that the air converges toward the orifice opening to flow through it. Also note that the area of the airflow is smaller than the area of the opening in the orifice plate. Vena Contracta is that point in the stream where the diameter is the smallest. The ratio of the area at Vena Contracta to the area of the opening is called the coefficient of contraction. Depending on the opening type, it can vary from 0.6 to 0.8.

Recall that $q = av$;

Therefore $v = \frac{q}{a}$.

Consider the quantity of air flowing through the orifice as q (cu. ft. per min.) and the area of the orifice as A (sq. ft.). From the discussion above on the vena contracta, the relationship between the quantity (q) and the velocity of the air through the orifice is:

$$V = \frac{q}{0.62A}$$

Recall the discussion on the velocity pressure or kinematic pressure in Section 1, where the velocity pressure (in inches of water) due to a velocity of v (ft. per min.) is given by:

$$i = \left[\frac{v}{4000}\right]^2$$

Substituting for $v = [\frac{q}{0.62A}]$,

$$i = \left[\frac{v}{4000}\right]^2 = \left[\frac{\frac{q}{0.62A}}{4000}\right]^2 = \left[\frac{q}{2480A}\right]^2$$

Therefore,

$$A = \frac{q}{2480\sqrt{(i)}} = \frac{0.0004q}{\sqrt{i}}$$

The area "A" in the thin plate is called the equivalent orifice. The equivalent orifice concept has several useful applications.

2. APPLICATION OF EQUIVALENT ORIFICE

The application of this formula to mine ventilation is extremely simple since the values which are necessary are those which can be obtained with the least trouble. If a mine is passing 150,000 cu. ft. of air at a 4-inch water gauge, its equivalent orifice would be,

$$A = \frac{0.0004\ x\ 150000}{\sqrt{4}} = 30\ square\ feet$$

The value in knowing the orifice of a mine lies in the fact that the resistance of the mine to the passing of the ventilating current can be checked at various times with very little effort. If airways are clogged with falls, or if other factors add to the mine resistance, the effect of these factors will show on the equivalent orifice value. The lower the resistance, the higher will be the orifice value; likewise, a mine with high resistance will have a low equivalent orifice.

As will be illustrated in a later section, the equivalent orifice formula is useful to calculate the size of the opening of a box regulator for passage of a certain quantity of air, q, through the regulator for a pressure drop of "i" inches of water.

E. SPLITTING THE AIR CURRENT

1. REASON FOR SPLITTING

In small mines, and in mines which are just being developed, the ventilating air is conducted through all of the active and inactive working faces and is returned to the outside atmosphere without a break in the current; this is called <u>continuous</u> ventilation. As the mine becomes larger and the number of workers increase, it becomes necessary to divide the mine workings into sections for purposes of ventilation. The various state mining laws require that only a certain number of men shall be allowed to work in the same current of air, so a mine employing a greater number of men than is permitted on one current must provide with two or more separate currents of air.

There are a number of reasons which justify splitting the air current. Division of the mine into sections, each with its particular split of air, allows complete control of the air in any one section without affecting the others; thus, the air volume may be increased or decreased at will, although in actual practice this affects to a certain extent the air input to adjacent sections as well. Again, certain sections of a mine may be more gaseous than others and will demand a larger volume of air to effectively reduce the gas content. Another reason for splitting is that foul or gassy air from one section is taken directly to the returns without being conducted through adjacent sections. Also, an explosion in one section need not cause interruption of the ventilation in other sections, and the dangerous gases formed by the explosion are carried directly to the returns. Finally, splitting the air is an economical procedure because of the reduced power needed to force air through the mine.

2. SPLITTING TERMS

The division or splitting of the main intake air usually takes place at or near the foot of the downcast shaft or slope or near the mouth of the intake airways. Divisions of the main intake current are called primary splits. Secondary splits are divisions of any primary split. Tertiary splits are divisions of any secondary split.

3. EQUAL AND UNEQUAL SPLITS

When a mine has two or more equal splits, it means that the length and size of the airways forming each split are equal. The resistance of each split will also be the same; hence, the amount of air flowing will be the same for each split. Any split in which a regulator must be used to make the resistance of that split equal to resistance of another split cannot be termed an equal split.

Unequal splits are those in which the resistance and the amount of air flowing in the various splits are unequal due to a difference in length or size. In considering unequal splits, we will treat them as (a) natural division of the air current and (b) proportionate division of the air current.

4. NATURAL SPLITTING OF THE AIR CURRENT

Natural splitting of the air current means that the air is allowed to divide among the several splits without artificial control of the amount going to each split. The split having the least natural resistance to the flow of air will receive the larges proportioning of the total intake air; the split having the next lowest resistance after the first one mentioned will receive the next largest portion of the total intake air, etc. The split having the shortest length of airway will usually have the least resistance although this may not hold true in all cases since the size of the airway must also be considered.

Calculation of quantity of air in a split. In making calculations where natural division of the air current exists, it is assumed that the pressure existing at the mouth of all splits starting from one point will be the same. Assuming this to be true, we find that the quantities of air passing in several splits may be obtained by following this rule:

The quantity of air passing in any split is to the pressure potential of that split as the quantity of air passing in all of the splits is to the sum of the pressure potentials of all of the splits, and this ratio is equal to the square root of the pressure.

If the quantity flowing in split 1 is q_1 and the pressure drop across a split,p_1 we know that

$$p_1 = \frac{k_1 l_1 o_1 q_1^2}{a_1^3}$$

Therefore, $q_1^2 = \frac{p_1 a_1^3}{k_1 l_1 o_1}$

Similarly, pressure drop across split 2, p_2, for quantity, q_2, will be

$$p_2 = \frac{k_2 l_2 o_2 q_2^2}{a_2^3}$$

$$q_2^2 = \frac{p_2 a_2^3}{k_2 l_2 o_2}$$

In natural splitting, the pressure drops across the splits are equal i.e. $p_1 = p_2$. Therefore, the quantity, q_k through the split, k, is

$$q_k \propto \frac{a_k\sqrt{a_k}}{\sqrt{k_k l_k o_k}}$$

Where $a_k, k_k l_k \; and \; o_k$ are the area, coefficient of friction surface, roughness factor, length and perimeter of the airway split k, respectively.

The quantity $[\frac{a_k\sqrt{a_k}}{\sqrt{k_k l_k o_k}}]$ is called the pressure potential of split k and denoted by the symbol x_k. The quantity flowing through split k, q_k, and be calculated from the following rule:

$$q_k = (\frac{\left[\frac{(a_k\sqrt{a_k})}{\sqrt{k_k l_k o_k}}\right]}{\sum_{i=1}^{n}\left[\frac{a_i\sqrt{a_i}}{\sqrt{k_i l_i o_i}}\right]} \; x \; q \; = \left[\frac{x_k}{\sum_{i=1}^{n} x_i}\right] x \; q$$

Where q = total quantity flowing in all the splits
n = number of splits in which the total quantity is divided

An example will illustrate the use of the formulas.

Example MV-14: A mine is passing 60,000 cu. ft. per min. through 2 splits (split 1 and split 2). Split q is 6 x 8 feet and 5,000 feet long and split 2 is 5x 8feet and 10,000 feet long. The value of $k \; = \; 0.0000000217 \frac{lb-min^2}{ft^4}$ for both splits. Find the volume of air in each split.

Solution:

$$a_1 \; = \; 6x8 = 48 \; sq.ft.$$

$$l_1 = 5000\ ft$$
$$o_1 = 2(6+8) = 28\ ft$$
$$k_1 = 0.0000000217 \frac{lb - min^2}{ft^4}$$

$$a_2 = 5x8 = 40\ sq.ft.$$
$$l_2 = 10{,}000\ ft$$
$$o_2 = 2(5+8) = 26\ ft$$

$$k_2 = 0.0000000217 \frac{lb - min^2}{ft^4}$$

$$x_1 = \frac{48\sqrt{48}}{\sqrt{0.0000000217\ x\ 5000\ x\ 28}} = 6035$$

$$x_2 = \frac{40\sqrt{40}}{\sqrt{0.0000000217\ x\ 10000\ x\ 26}} = 3368$$

$$x_1 + x_2 = 9403$$

$$Q_1 = \frac{x_1}{x_1 + x_2}\ x\ 60000 = \frac{6035}{9403}\ x\ 60000 = 38{,}500$$

$$q_2 = \frac{x_2}{x_1 + x_2}\ x\ 60000 = \frac{3368}{9403}\ x\ 60000 = 21{,}500$$

Check: [The two quantities should add to 60,000 cfm]

$$q_1 + q_2 = 38500 + 21500 = 60000$$

Since the value of “k” is same for both the splits, we could have omitted the use of “k” in our calculating the formula for pressure potential.

Example MV-15: An air current of 80,000 cubic feet per minute is passing in a mine in the following three splits:

Split No. 1: 6 x 8 x 3000 feet long
Split No. 2: 4 x 16 x 2000 feet long
Split No. 3: 8 x 10 x 5000 feet long

[1] Find the natural division of the air, specifying the quantity flowing in each split. The value of “k,” the coefficient of friction, is 0.0000000217 $\frac{lb{-}min^2}{ft^4}$ for all of the splits.

[2] Calculate the pressure at the mouth of a split.

[3] Calculate the horsepower on the air at the mouth of a split

Solution: Since the value of k is same for all the splits, we will use the following formula for pressure potential:

$$x = a\sqrt{\frac{a}{s}} \qquad \text{or} \qquad a\sqrt{\frac{a}{l\ x\ o}}$$

Split No. 1: a=48; o=28; l=3000
Split No. 2: a=64; o=40; l=2000
Split No. 3: a=80; o=36; l=5000

$$x_1 = \frac{\left(48\sqrt{48}\right)}{\sqrt{3000\ x\ 28}} = 1.15$$

$$x_2 = \frac{\left(64\sqrt{64}\right)}{\sqrt{2000\ x\ 40}} = 1.81$$

$$x_3 = \frac{\left(80\sqrt{80}\right)}{\sqrt{5000\ x\ 36}} = 1.69$$

$$x_1 + x_2 + x_3 = 4.65$$

$$q_1 = \left[\frac{x_1}{x_1 + x_2 + x_3}\right]\ x\ q = \left[\frac{1.15}{4.65}\right]\ x\ 60000 = 19{,}785$$

$$q_2 = \left[\frac{x_2}{x_1 + x_2 + x_3}\right]\ x\ q = \left[\frac{1.81}{4.65}\right]\ x\ 60000 = 31{,}140$$

$$q_3 = \left[\frac{x_3}{x_1 + x_2 + x_3}\right]\ x\ q = \left[\frac{1.69}{4.65}\right]\ x\ 60000 = 29{,}075$$

Check: $q_1 + q_2 + q_3 = 80{,}000$

[2] Calculation of pressure at the mouth of a split.

As stated previously, the pressure at the mouth of several splits will be the same for each split. This pressure can be calculated from formula:

$$p = \frac{ksq^2}{a^3}$$

The following illustrates the use of the formula for split 1.

Split No. 1: $Area = 6\ x\ 8 = 48\ square\ feet;$

$$Perimeter = 2(6+8) = 28\ feet;$$
$$Length = 3000\ feet;$$
$$Quantity = 19{,}785\ cubic\ feet;$$

$$p = \frac{ksq^2}{a^3} = \frac{0.0000000217\ x\ 28\ x\ 3000\ x\ 19785^2}{48^3} = 6.45\ pounds\ per\ square\ foot$$

[3] Calculation of horsepower on the air at the mouth of a split. The horsepower on the air at the mouth of a split (or the foot of the downcast shaft if the splits begin at this point) is found by the formula,

$$h.p. = \frac{p\ x\ q}{33000}$$

In our case the pressure across all the splits will be the same, and the quantity is 80,000 cfm. The power on the air will be,

$$h.p. = \frac{6.45\ x\ 80000}{33{,}000} = 15.64\ horsepower$$

5. PROPORTIONATE DIVISION OF THE AIR CURRENT

Natural division of the air current among several splits is often the reverse of what is desired. The longest splits under natural division would get the least amount of air, whereas such splits with their greater number of working places and greater amount of gas liberation (if the mine is gassy) should receive the largest quantities of air. The short splits with their low resistance would, under natural division, receive a large share of the total volume of air and this amount would probably be excessive for the few places and small amount of gas liberated. It thus becomes necessary to control the flow of air where several splits exist and create proportionate division of the air; this means dividing the air into the proportions desired for each split. The manner in which this is done is to create additional resistance in the splits having low resistances or in some other manner to reduce the air flow in these splits and force air through the splits normally having greater resistances. This proportionate division is obtained by the use of regulators of either the door or box type.

Calculation of the Size of the Box Regulator

Box regulator size is calculated using the equivalent orifice formula that was developed earlier. Recall from earlier discussion, the formula for equivalent orifice:

$$A = \frac{q}{2480\sqrt{(i)}} = \frac{0.0004q}{\sqrt{i}}$$

This formula is useful to calculate the size of the opening of a box regulator for passage of a certain quantity of air, q, through a box regulator for a pressure drop of “i” inches of water.

Example MV-16: Calculate the size of the box regulator opening to course 17,500 cu. ft. per min. across the regulator for a pressure drop of 2.8 inches of water.

Solution:

$$A = \frac{0.0004q}{\sqrt{i}} = 0.0004\,x\,\frac{17500}{\sqrt{2.8}} = \frac{7.0}{1.67} = 4.18\, sq.ft.$$

Example MV-17: The quantity of air flowing through a box regulator is 18,000 cu. ft. per min. The area of the regulator opening is 6.6 sq. ft. Calculate the pressure drop in inches of water across the regulator.

Solution:

$$A = \frac{0.0004q}{\sqrt{i}}$$

$$\sqrt{\mathrm{i}} = \frac{0.0004q}{A}$$

$$i = \left[\frac{0.0004\mathrm{q}}{\mathrm{A}}\right]^2 = \left[\frac{0.0004\,x\,18000}{6.6}\right]^2 = 1.19\, inches$$

Example MV-18: With 9000 cubic feet of air passing through a box regulator each minute, a water gauge shows a difference in pressure on the two sides of the regulator of 0.75 inches. If the shutter opening is 14 inches high, how wide is the opening?

Solution: $A = \frac{0.0004q}{\sqrt{i}} = \frac{0.0004\,x\,9000}{\sqrt{0.75}} = 4.15\, square\ feet$

Now 14 inches represents 1.16 feet. Therefore, the opening width must be 4.15 ÷ 1.16 = 3.56 or 3 feet 6.8 inches.

Calculation of the Size of the Door Regulator

In the use of door opening to regulate the quantity of air through an entry, the height of the door will be approximately equal to the height of the mine opening and width of the opening is adjusted to ensure the desired quantity of flow. The formula for the calculation of the area of the door opening is in general the same as before except that in its derivation no allowance is made for the vena contracta effect. Essentially, the area of flow stream through the opening is the same as the area of the opening itself. Thus, the area of the opening of a door regulator is:

$$A = \frac{0.00025q}{\sqrt{i}}$$

Where $A = area\ of\ the\ door\ opening\ in\ sq.ft.$

$q = quantity\ flowing\ in\ cu. ft. per\ min.$
$i = pressure\ drop\ in\ inches\ of\ water$

Example MV-19: What must be the width of the opening of a regulator door in an entry 5 feet high if 40,000 cubic feet of air is passing each minute? The split w.g. is 1.25 inches and the free split w.g. is 1.75 inches.

Solution: The difference in pressure to be regulated by the door will be,

$$1.75 - 1.25 = 0.5\ inches\ w.g.$$

$$A = \frac{0.00025\ x\ 40{,}000}{\sqrt{0.5}} = 14.14\ square\ feet$$

Since this is the area of the opening and the door height is 5 feet, the width of the opening will be, 14.14÷5=2.83 feet or 2 feet, 10 inches.

6. PROPORTIONATE SPLIT CALCULATIONS INVOLVING THE BOX REGULATOR

Division of air through control of split resistances

To obtain the proper division of air among several splits, box regulators are placed in all of the splits save one and this is termed the open or free split. This placing of the box regulators increases the resistance in all of the regulated splits until the desired quantity is flowing in each one. It thus becomes necessary to find the resistance of each split under natural division of the air in order to determine which splits must be regulated.

First, the pressure required to circulate air in each split is found by using formula: $p = \frac{ksq^2}{a^3}$. An example will serve to show its use.

Example MV-20: A mine has four splits as shown below. Where should regulators be placed to give the division of air among the splits, and what will be the mine pressure? The coefficient of friction is $0.0000000217\ \frac{lb-min^2}{ft^4}$ for all the airways.

Split No. 1: 6 x 9 feet; 8,000 feet long; 40,000 cfm.
Split No. 2: 5 x 8 feet; 6,000 feet long; 40,000 cfm.
Split No. 3: 9 x 9 feet; 8,000 feet long; 10,000 cfm.
Split No. 4: 6 x 8 feet; 10,000 feet long; 30,000 cfm.

Also, calculate the regulator size opening for one of the regulated splits and the horsepower for the entire mine.

Solution:

Using the formula $p = \frac{ksq^2}{a^3}$, the pressure due to friction in each split would be found as follows.

$$p_1 = \frac{0.0000000217\ x\ 2\ x\ (6+9) x\ 8000\ x\ 40000^2}{(6x9)^3} = 52.92\ psf.$$

$$p_2 = \frac{0.0000000217\ x\ 2\ x\ (5+8) x\ 6000\ x\ 40000^2}{(5x8)^3} = 84.63\ psf.$$

$$p_3 = \frac{0.0000000217\ x\ 2\ x\ (9+9) x\ 8000\ x\ 10000^2}{(9x9)^3} = 1.176\ psf.$$

$$p_4 = \frac{0.0000000217\ x\ 2\ x\ (6+8) x\ 10000\ x\ 30000^2}{(6x8)^3} = 49.45\ psf.$$

The split with the greatest pressure, which is No. 2 split, will be the ***open split and will give the pressure for the entire mine.*** Regulators must be placed in splits Nos. 1, 3, and 4 to increase their respective pressures to that of the open split pressure.

The box regulator is placed at the end of the split and is adjusted so as to increase the resistance of that split until it equals the resistance of the mine or the open split. The resistance, therefore, which the box regulator introduces in each split, must equal the resistance of the open split less the natural resistance of the regulator split. In the above example the open split resistance is 84.63 pounds per square foot so the regulator No. 1 split must have a pressure loss of$[84.62 - 52.92] = 31.71\ psf.$

Area of box regulator opening in Split No. 1. The area of the opening in the box regulator is calculated using the equivalent orifice formula as in the examples before. Since the regulator pressure drop is in inches of water gage, it is necessary to convert 31.71 *psf* into inches of w.g. Remember that one inch w.g. = 5.2 *psf.* In the above example, the regulator pressure of split No. 1 would represent $\frac{31.71}{5.2} = 6.1$ inches w.g.

$$Area\ regulator\ opening = \frac{0.0004\ x\ 40000}{\sqrt{6.1}} = 6.47\ square\ feet$$

The areas of the regulator openings in splits 3 and 4 will be found in the same manner.

Calculation of horsepower on the air. The horsepower on the air for the entire mine is found by using formula: hp = [pq/33000]. In this case the quantity of air is the sum of the quantities circulating in all of the splits, while the pressure is that of the split with the greatest natural resistance. We see that,

$$q = 40000 + 40000 + 10000 + 30000 = 120{,}000\ cu.ft.$$

$$p = 84.63 \; pounds \; per \; square \; feet$$

Then, $$h.p. = \frac{84.63 \; x \; 120000}{33000} = 307.745 \; horsepower$$

If the hp. on any single split is desired, the pressure will be that of the mine, (since the regulator has brought each split pressure up to that of the open split) and the quantity will be that of the split to be calculated. Thus in split No. 4,

$$h.p. = \frac{84.63 \; x \; 33000}{4 \; x \; 33000} = 76.93 \; horsepower$$

Calculation of Regulator Size when Power of the Air in the Split or the Pressure in the Split Remains Same [after Regulation].

When the power of the air remains the same, it is to be noted that the pressure in the split will increase because of the decrease in quantity. Therefore, the regulator size has to take into account this pressure increase due to the reduced quantity as well as the reduced pressure due to reduced quantity. The following problem illustrates the calculations involved.

Example MV-21: An airway, 7 x 10 feet, is passing 35,000 cubic feet of air per minute when the water gauge is 0.75 inches. It is desired to reduce this quantity to 21,000 cubic feet per minute. If the <u>power on the air remains constant</u>, what must the area of the regulator?

Solution: First, we must determine what the water gauge will be under the new condition. Remember that power is proportional to [pressure x quantity]. Therefore, when quantity decreases, pressure increases for the same power.

35000 x 0.75 = 21000 x [new pressure]

New pressure = [35000/21000] x 0.75 = 1.25 *inches*

Second, the water gauge required for the reduced quantity flowing after regulation in the split will vary directly as the square of the quantity. The new water gage required can be calculated as follows:

$[H_{new}/0.75] = [21000/35000]^2$

$H_{new} = [0.75][21000/35000]^2 = 0.27$ *inches*

The final w.g. to be regulated will be found by taking the difference between New pressure and H_{new}

$$Final \; w.g. = 1.25 - 0.27 = 0.98 \; inches$$

$$A = \frac{0.0004\ x\ 21000}{\sqrt{0.98}} = \frac{8.4}{0.99} = 8.48\ square\ feet$$

It should be noted that this solution is proper only when the power on the air is held constant. An interesting comparison will be noticed in the next example when only the water gage in the split is held constant before and after regulation.

Example MV22: If, in the previous example, the new volume of air is passed under the same water gauge [0.75*inches*]. What should be the area of the regulator opening?

Solution: Under this condition, we must first solve for the w.g. required for the reduced quantity of air which will be passing in the airway. Then, since our w.g. is to remain at 0.75 inches, the pressure due to the regulator will be the difference between 0.75 inches and the regulated pressure.

The w.g. at the new quantity will vary as the square of the quantity. The new water gage required is calculated as follows:

$$[H_{new}/0.75] = [21000/35000]^2$$

$$H_{new} = [0.75][21000/35000]^2 = 0.27\ inches$$

Then the pressure which the regulator must exert will be,

$$0.75 - 0.27 = 0.48\ inches$$

The area of opening to pass 21,000 cubic feet of air per minute under a w.g. of 0.48 inches will be,

$$A = \frac{0.0004\ x\ 21000}{\sqrt{0.48}} = 12.12\ square\ feet$$

F. PROPORTIONATE SPLIT CALCULATIONS INVOLVING DOOR REGULATORS

1. POWER CONSUMPTION AND DIVISION OF AIR

The use of door regulators supposedly does not add to the resistance of splits and is, therefore, preferable to the use of box regulators. All calculations involving door regulators are made on the theory that their use permits each split to operate under its natural resistance. Actually, door regulators do add some resistance to the natural split resistance, just as in the case of box regulators, although it is probable that in this respect they are more efficient because of their type of construction.

Example MV-23: Consider a mine with four splits as follows. The coefficient of friction is $0.0000000217\ \frac{lb-min^2}{ft^4}$ for all the airways.

No. 1: 6 x 9 feet; 8,000 feet long; 40,000 cfm.
No. 2: 5 x 8 feet; 6,000 feet long; 40,000 cfm.
No. 3: 9 x 9 feet; 8,000 feet long; 10,000 cfm.
No. 4: 6 x 8 feet; 10,000 feet long; 30,000 cfm.

[1] Find the natural division of the air and the horsepower on each split of air.
[2] Calculate the hp. for each split for the proportionate splitting
[3] Calculate door regulators for the controlled splits.

Solution: Since all splits have the same "K" value, natural division of the air will be found by using formula:

$x = a\sqrt{\frac{a}{l\ x\ 0}}$, and then solving formula: $q = \frac{Q\ x\ X}{\sum X_p}$

Using this formula for each split we have,

$$x_1 = (6x9)\sqrt{\frac{(6x9)}{2(6+9)\ x\ 8000}} = 0.810$$

$$x_2 = (5x8)\sqrt{\frac{5x8}{2(5+8)\ x\ 6000}} = 0.640$$

$$x_3 = (9x9)\sqrt{\frac{9x9}{2(9+9)\ x\ 8000}} = 1.377$$

$$x_4 = (6x8)\sqrt{\frac{6x8}{2(6x8)\ x\ 10000}} = 0.624$$

$$\sum_p^x = 0.810 + 0.640 + 1.377 + 0.624 = 3.451$$

Using formula for the quantities in each split we have,

$$q_1 = \frac{120,000\ x\ 0.81}{3.451} = 28,170\ cfm.$$

$$q_2 = \frac{120{,}000 \; x \; 0.640}{3.451} = 22{,}265 \; cfm.$$

$$q_3 = \frac{120{,}000 \; x \; 1.377}{3.451} = 47{,}885 \; cfm.$$

$$q_4 = \frac{120{,}000 \; x \; 0.624}{3.451} = 21{,}680 \; cfm.$$

$$\sum Q(q_1, q_2, q_3, q_4) = 120{,}000 \; cfm.$$

In using door regulators, we revise the natural flow of air in the several splits as shown above, and the pressures in these splits will vary with this flow of air. The pressures will be found by formula: $p = \frac{ksq^2}{a^3}$. Using this formula,

$$p_1 = \frac{0.0000000217 \; x \; 2(6+9) x \; 8{,}000 \; x \; 40{,}000^2}{54^3} = 52.92 \; lbs. per \; sq. ft.$$

$$p_2 = (repeat, inserting \; proper \; values) = 84.63 \; lbs. per \; sq. ft.$$

$$p_3 = (repeat, inserting \; proper \; values) = 1.176 \; lbs \; per \; sq. ft.$$

$$p_4 = (repeat, inserting \; proper \; values) = 49.45 \; lbs. per \; sq. ft.$$

The horsepower on each split will be found by formula:

$$h.p. = \frac{pq}{33{,}000}$$

$$h.p._1 = \frac{52.92 \; x \; 40{,}000}{33{,}000} = 64.145 \; horsepower$$

$$h.p._2 = \frac{84.63 \; x \; 40{,}000}{33{,}000} = 102.582 \; horsepower$$

$$h.p._3 = \frac{1.176 \; x \; 10{,}000}{33{,}000} = 0.356 \; horsepower$$

$$h.p._4 = \frac{49.45 \; x \; 30{,}000}{33{,}000} = 44.955 \; horsepower$$

$$TOTAL = (h.p._1 + h.p._2 + h.p._3 + h.p._4) = 212.038 \; horsepower$$

In solving for power where box regulators were used under the same conditions as were used in this problem, we found that 307.745 hp. were consumed. The saving in power which resulted from the use of door regulators will be,

$$307.745 - 212.038 = 95.707 \text{ } horsepower$$

<u>Area of Regulator Opening</u>. The area of opening made by swinging a door regulator is calculated using the formula:

$$A = \frac{0.00025q}{\sqrt{i}}$$

In this example, the door regulator for Split No. 1 will have to lose [84.63-52.92] lbs. per sq. ft. pressure [psf]. Note that this difference, 31.71 psf, is equal to [31.71/5.2] inches of w.g., i.e. 6.1 inches of w.g. substituting these values in the above equation, we have,

$$A = \frac{0.00025 \; x \; 40000}{\sqrt{6.1}} = 4.05 \; sq. ft.$$

The areas of the door regulators for Split No. 3 and Split No. 4 will be found in a similar manner. Note that in the box regulator opening for Split No. 1 for the same problem was 6.47 sq. ft.

G. CALCULATIONS INVOLVING PRIMARY AND SECONDARY SPLITS OF AIR

Our work so far has dealt with primary splits only. In dealing with secondary splits we will use the same formulae as in solving primary split problems but we must treat the splits in a different manner.

Example MV-24: Let us assume a mine as having two primary splits (No. 1 and 2) and two secondary splits (No. 3 and 4) which are taken from No. 2 split. It is hardly possible that the air will flow through these splits in the desired quantities without regulation so either door or box regulators must be used. To illustrate our problem, we will assign the following values to the several splits:

No. 1: 4 x 5 feet; 800 feet long; 3,500 cubic feet air per minute
No. 2: 4 x 5 feet; 500 feet long; 6,500 cubic feet air per minute
No. 3: 4 x 5 feet; 400 feet long; 4,000 cubic feet air per minute
No. 4: 4 x 5 feet; 300 feet long; 2,500 cubic feet air per minute

Solution:

First we will find the pressure in each split, using the formula:

$$p = \frac{ksq^2}{a^3}$$

$$p_1 = 0.47848\ lbs. per\ sq. ft. (primary\ split)$$
$$p_2 = 1.03140\ lbs. per\ sq. ft. (primary\ split)$$
$$p_3 = 0.31248\ lbs. per\ sq. ft. (secondary\ split)$$
$$p_4 = 0.091546\ lbs. per\ sq. ft. (secondary\ split)$$

We must first determine which of the secondary splits will be the free split. Since No. 3 has the greater pressure loss, it will be the free split and No. 4 will be regulated until its pressure will equal that of No. 3 (supposing a box regulator is used).

The entire No. 2 split (including the secondary splits) will now have a pressure of 1.0314+0.31248 or 1.34388 lbs. per sq. ft. and the primary splits will have pressure as follows:

No. 1 = 0.47848 lbs. per sq. ft.
No. 2 = 1.34388 lbs. per sq. ft.

It will be recognized that No. 2 split will be the free split as far as these two are concerned, and No. 1 split will be the regulated split.

Regulator pressures would be as follows:

No. 4 regulator pressure = $p_3 - p_4 = 0.220934\ lbs. per\ sq. ft.$
No. 1 regulator pressure = $p_2 + p_3 - p_1 = 0.86540\ lbs. per\ sq. ft.$

The areas of regulator openings, the horsepower on the air for each split, and the various split resistances would be found in a manner similar to those used in solving primary split problems.

1. LIMIT IN SPLITTING OF AIR CURRENT

The velocity of the air and the pressure and power required to force air through a passage will decrease as the cross-sectional area increases. It is feasible, then, to take a current of air which has been sweeping 10,000 feet of airway and split it so that two currents of air each sweep an airway 5,000 feet in length. If the size of airway has not changed (the rubbing surface will be the same), it is possible (theoretically) to supply with the same amount of power twice the quantity of air formerly flowing and at the same velocity.

Theoretically, then, an unlimited number of splits is possible; practically, there is a limit to splitting. All of the air supplied to a mine must come through one opening (or at the most two or three openings, although in such cases each opening will have its own ventilating current), before it reaches the point of splitting. Sometimes the air must travel through a short airway before the spits are taken off. If the total volume of air for the mine is increased, the increase in velocity down the shaft or through the short connecting airway will be considerable and this velocity increases will cause a drop in pressure between the surface and the point where the air current is split. Each additional split will thus cause a reduction in the pressure at the mouth of the splits and the velocity of the air in the splits will drop due to this decreased pressure. When the air current at the working face drops below 60 feet a minute additional splitting will produce a hazard resulting from too low a velocity of air current at the face. It then becomes necessary to increase the fan speed, which will result in increased velocity but also in increased power consumption, if adequate ventilation is to be maintained.

TABLE OF CONTENTS

VENTILATION SECTION 3

SECTION 3

VENTILATING APPARATUS

In Section 3, we discuss a number of topics on ventilation equipment and systems that are associated the coursing of mine air. There are a large number of devices that are used for controlling the flow of air through a mine – doors, stoppings, air crossings and regulators. The ventilating pressure is generated either naturally or by mechanical means. The openings to the mine and the main intake and return airways must be properly located, sized, designed, constructed and maintained for ensuring good performance of the ventilation system. On several of these topics, there are mandatory standards in the Federal and Pennsylvania Bituminous Coal Mining Laws. The mine officials must be fully familiar with the requirements on design, construction, maintenance and inspection.

A. DEVICES FOR CONTROLLING VENTILATION

1. DOORS

Use

Ventilation doors are used in mines to deflect air currents and still allow the passage of men, equipment, or vehicles Figure MV-3]. Their use is confined to haulageways, either of the permanent or temporary type, or in a few isolated cases to passageways between intake and return airways where a door, rather than another type of air current deflector, is needed.

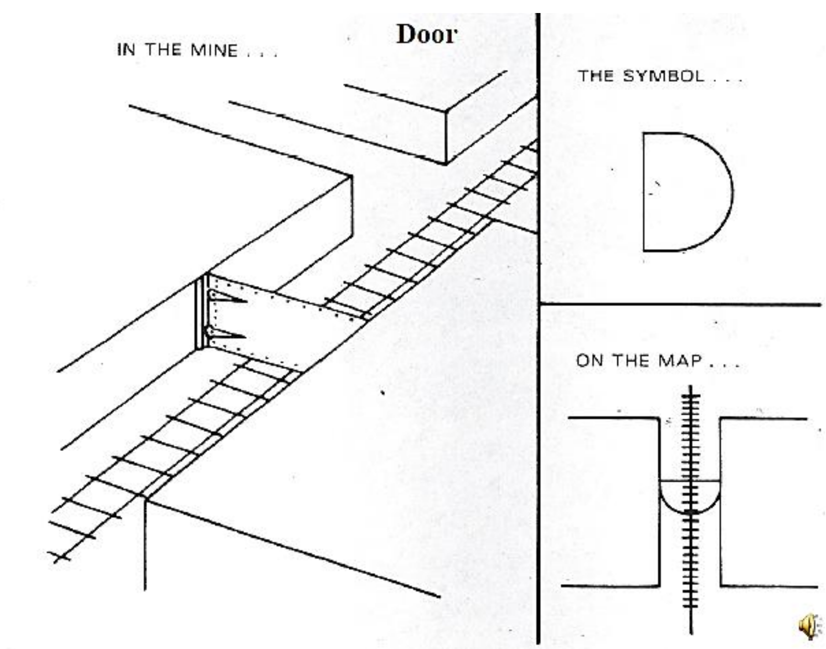


Figure MV-3: Door in the Mine, Symbol, Door on the Map [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

Good mining practice frowns on the indiscriminate use of doors so they are used mainly in development sections. As the mine continues to develop, the coursing of air is revised by establishing spits, thus eliminating the need for doors.

Rules which should never be broken in using a ventilation door are: Never allow the use of a latch which will hold the door open permanently; and never block open a door that is intended to remain closed. Many explosions have resulted from gas accumulations in working sections when the air was short-circuited by an open door, either latched or blocked open for a long period of time by motorman, or some other person who did not realize the hazard being created.

Airlocks

Where the opening of a door will short circuit the main ventilation current, doors should be built in pairs to form an air lock. Air lock doors should always be made to close in the same direction, that is, toward the low pressure side. An air lock incorporating three doors is shown in Figure MV-4. This kind of air lock is common in an airway connecting the intake and return airways near their entrance to the mine where the pressure difference is very high making it very difficult to open the doors if there were only two doors. Another important reason is that because of the high pressure difference, the leakage potential is also very high. Three substantially constructed doors substantially reduce these problems. Today, automated airlocks are available to ensure remote and safe operation of the doors.

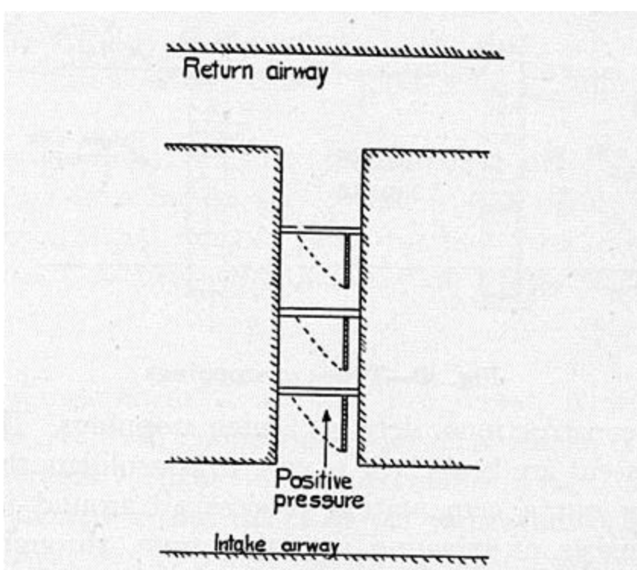


Figure MV-4: An Airlock between Main Intake and Main Return incorporating Three Doors.

[Source: http://www.theoildrum.com/node/6665]

Figure MV-5: A Man Door in a Coal Mine
[Source: http://www.zeromandoor.com]

Often, miners may have to travel from an intake airway to a return airway at some selected crosscut. In the stopping in this crosscut, a door, called man-door, is erected [Figure MV-5].

2. STOPPINGS

Use

Stoppings in a ventilation system are used to prevent intake air from short circuiting and to extend the predetermined path the air must follow to reach the active faces or pillar workings. Permanent stoppings are those which will remain for a long period of time; temporary stoppings are those which are placed as the workings advance, and which are later removed or replaced by permanent stoppings.

Construction

Ordinarily a stopping should be made as tight as possible because of the cost of air losses through stoppings which leak air.

Permanent stoppings [Figure MV-6] for use in crosscuts between main intake and return airways should be made of incombustible materials. Concrete block are frequently used for permanent stoppings; cement mortar is used, and central pilasters give additional strength where necessary. Additional tightness under high pressure differences can be obtained by using a mortar facing on one side.

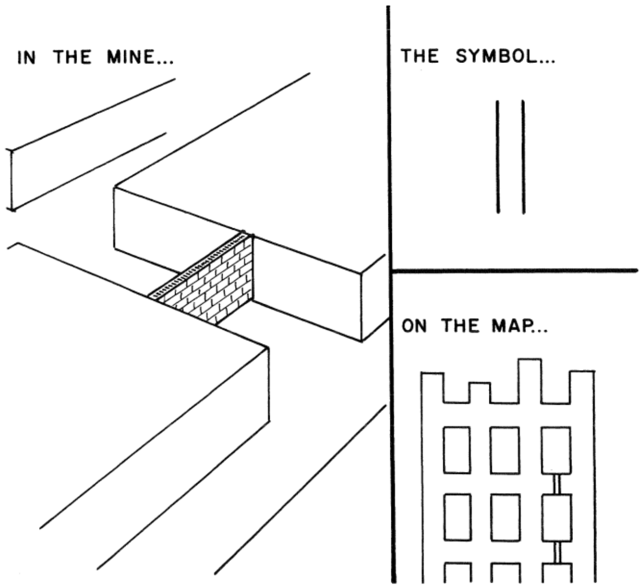


Figure MV-6: Permanent Stopping in a Mine, Symbol, and on a Mine Map [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

Temporary stoppings [Figure MV-7] in bituminous coal mines are generally of flameproof material fastened to a board framework supported by posts, although the use of corrugated galvanized steel or aluminum sheets offer the advantage of speed in construction and reuse of materials. These are used to close the next to the last open crosscuts in advancing headings or rooms. In heading work they are later replaced by permanent stoppings; in rooms, they remain until robbing of pillars makes their removal necessary.

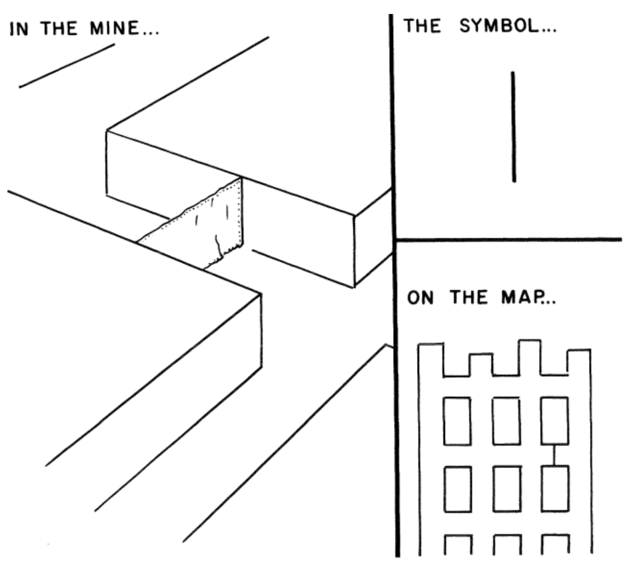


Figure MV-7: Temporary Stopping in a Mine, Symbol and on a Mine Map [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

Features Which Affect Construction

The length of time a stopping will be needed and the pressure difference it must withstand will determine the materials to be used and the care with which a stopping will be built. Stoppings which will be in place a long time should be substantially built, and leakage should be kept to a minimum; air lost in a main stopping means so much less air carried to the working sections and, if more air is needed, this means increasing the quantity produced by the fan and increasing the power cost for ventilation. Tightness can be obtained by hitching the stopping into the roof, ribs, and floor. An excellent practice is to gunite all permanent stoppings at which high-pressure differences exist, spraying the stopping face and the ribs, roof, and floor for several feet back from the wall.

The stopping location should be far enough back form the pillar corners to prevent leakage developing at some future data as a result of spalling at the pillar corners.

When a soft fireclay bottom exits at the stopping location, heaving of the bottom may cause damage. Sometimes removal of this bottom in all that is necessary, while the use of one or two soft wood ties built in at the top will act as a cushion for the stopping if the bottom heaves.

Box Checks [Figure MV-8] are stoppings with a hole in them to allow equipment such as a conveyor to pass through them while limiting the flow of air through the belt entry. They are generally placed in pairs in an entry, at the very beginning and at the very end, to limit both the intake and return air flowing through the belt entry [Figure MV-9].

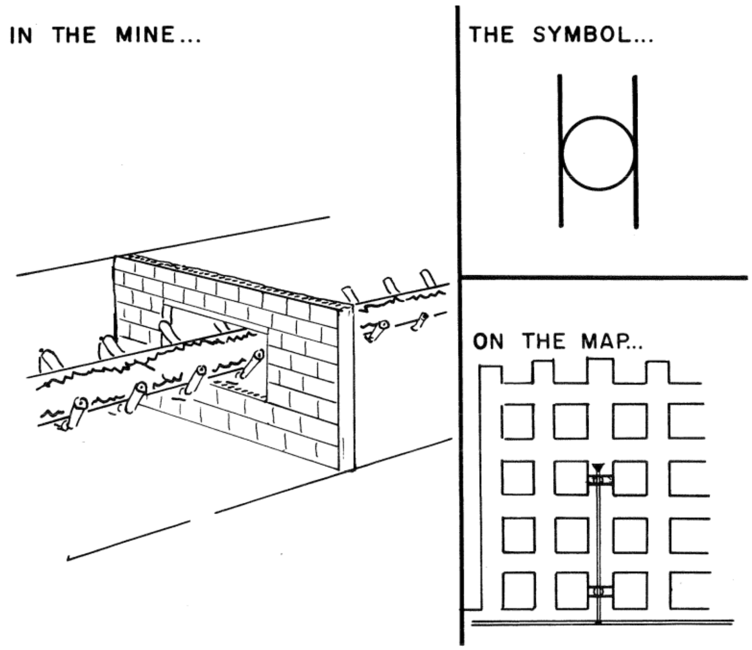


Figure MV-8: Box Check in a Mine, Symbol and on a Mine Map [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

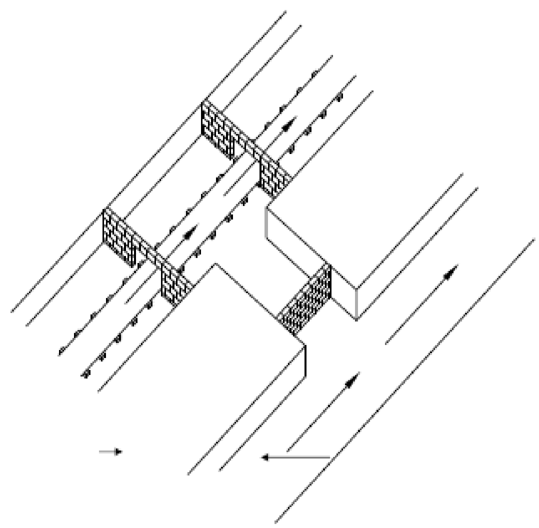


Figure MV-9: Box Checks showing Belt Entry Ventilated by Leakage Air

3. AIR CROSSINGS

<u>Use</u>

An air crossing or air bridge is a structure used to permit air currents to cross each other without mixing or intermingling. The return air may cross over the intake airway, in which case the stopping is overcast or it may pass under the intake airway, in which case it is call an undercast [Figure MV-10 and Figure MV-11]. Overcasts are the most widely used type of air crossing, while undercasts are used only in special conditions. Undercasts are liable to become filled with dust and dirt, or to collect water, so overcasts normally are preferred.

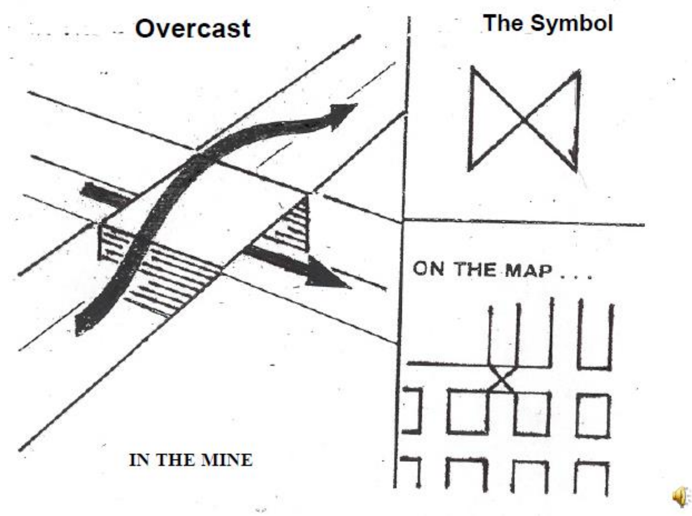


Figure MV-10: Schematic and Cross-Sectional Views of Air Crossing with Air Crossing on a Mine Map and the Symbol of Air Crossing – Overcast [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

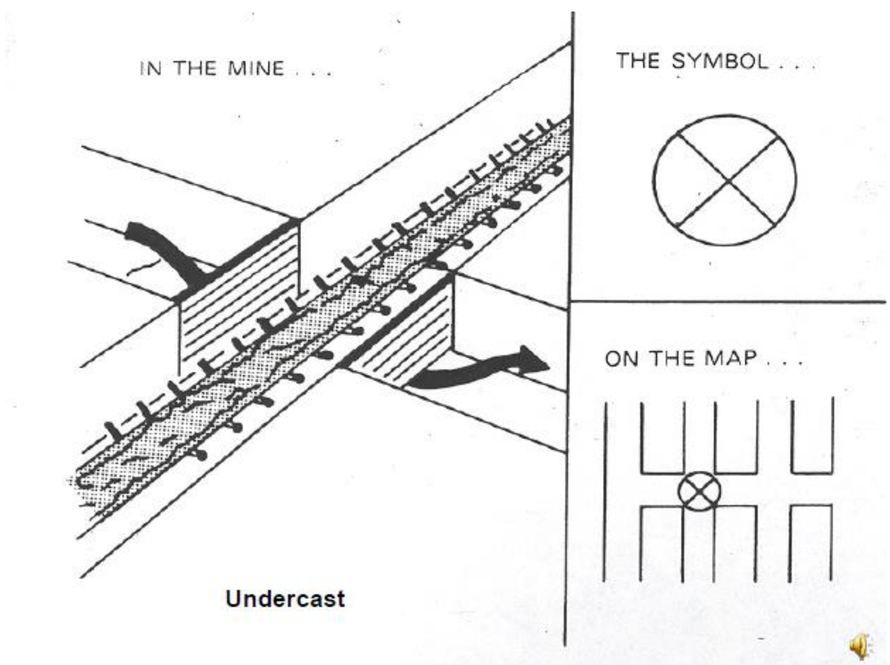


Figure MV-11: Schematic and Cross-Sectional Views of Air Crossing with Air Crossing on a Mine Map and the Symbol of Air Crossing - Undercast [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

4. CURTAINS AND DEFLECTOR BRATTICES

Curtains

A curtain, sometimes called a check curtain, check, or checker, is a means of diverting air near working faces so that a sufficient quantity will sweep the face and remove dust and methane or other dangerous gas issuing from the face [Figure MV-11]. It is constructed by hanging flameproof material. The curtain may be slit in the center or, when several pieces are used, short pieces may be hung to overlap in the center and provide a place where men or equipment may pass through.

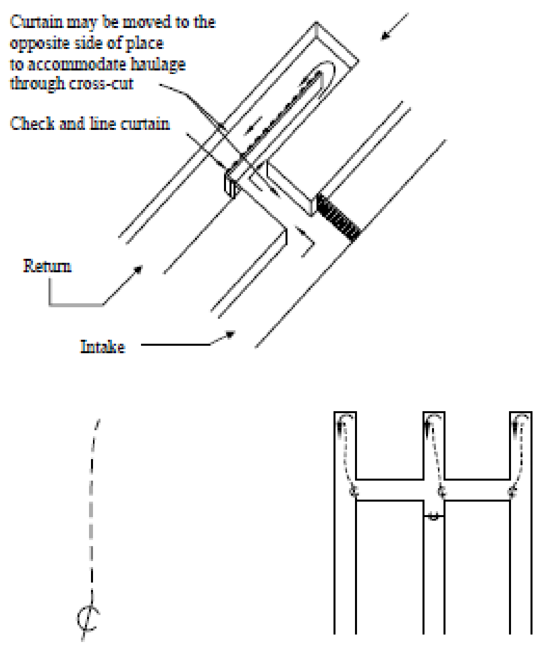


Figure MV-11: Check Curtain

5. REGULATORS

Use

As a mine increases in size it becomes necessary to divide or spit the main intake air current. When the resistance areas which the various split air currents must ventilate are approximately equal, the intake air will be divided into approximately equal quantities for each split, such a condition resulting in natural proportioning of the air. Mining, however, must be varied according to the difficulties encountered, and equal division of the air among several splits is rare. Developing sections may have much longer airways than sections where pillars are being robbed. In either case one section will offer less resistance than the other sections to the flow of air and, since air will always take the path with the least resistance, more air will flow through one section than through the others. The section with the greatest resistance to air flow, however, may require more air than it would normally get under this natural division because of the greater amount of liberated gas which must be removed. Regulators are usually placed in the splits having the low-resistance sections and force more air through the high-resistance section. A regulator, then, is a means of introducing artificial resistance to the flow of air in sections with low natural resistance so as to bring the total resistance of these sections up to or near that which exists naturally in the unregulated or "free" split.

Construction and Location

There are three types of regulator used in coal mines. These are (a) the box type, (b) the half-brattice type, and (c) the hinged door type.

The box regulator consists of an adjustable opening in a stopping [Figure MV-12]. The convenience of the opening for travel of men usually places its lower edge from 18 inches to 2 feet above the bottom. Its maximum height is about 30 inches, and its length depends on the amount of resistance which must be introduced but is rarely over 4 feet. A sliding shutter which move horizontally is used to regulate the amount of air which experience indicates must pass through the section.

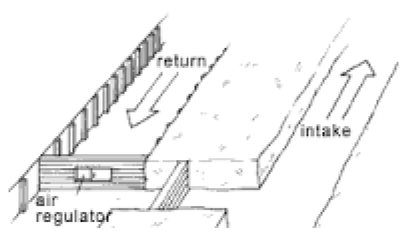


Figure MV-12: Box Regulator [Regulator Opening is adjustable]

The location of the box regulator is usually in the return airway from the split so as to not interfere with haulage [Figure MV-13]. The adjustable shutter permits great accuracy in the regulation of the amount of air which will be permitted to pass and, once set, need not be readjusted until a change in the natural resistance of the several spits require the opening or closing of the regulator until the proper resistance balance has been reached.

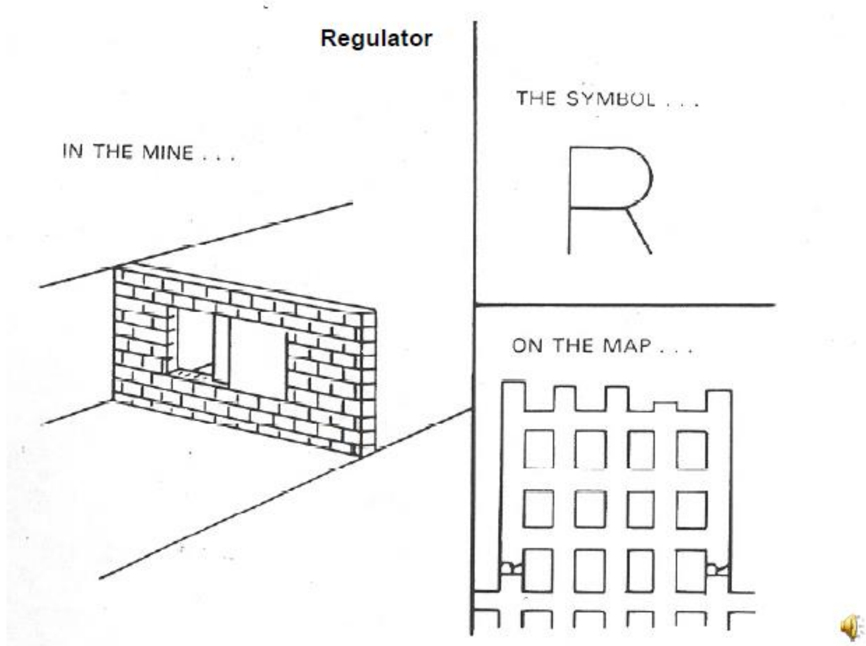


Figure MV-13: Schematic of a Box Regulator, Symbol and the Regulator in a Mine Map [http://miningquiz.com/Kentucky_Foreman/Foreman_Trng_Unit_7_Ventilating_A_Mine_Part_2_(Narrated)_115_slides.pdf]

The half-brattice regulator consists of a brattice extending out from one rib towards the opposite side and built to extend from floor to roof. The portion of the airway not covered by the brattice is the regulator opening. This type is constructed to have a fixed area of opening and does not lend itself to close adjustment of air flow under a variation of split resistances. It is usually placed in the return airway of a split for the same reason as the box regulator is placed in such a location. A similar type of regulator might well be used on the intake airway if constructed as half-door to allow the passage of equipment. This location would permit easy and constant inspection but would introduce a variation in air flow in all of the splits when the half-door would be open to allow men or equipment to pass since reducing the resistance of one split would affect the flow of air in all of the other splits.

The hinged door regulator is a door placed at the junction of an air split and the main intake air course so as to divert a porting of the main intake air current to the split. It is hung form the pint of the rib separating the second intake airway and the main intake airway so it may be swung either way to divert more or less air to the split. A locking device holds it in a fixed position which is determined by trial. This type of regulator introduces much less resistance to airflow than either the box or half brattice types and is preferable where conditions permit its use.

B. VENTILATION OPENINGS TO COAL MINES

The course of air through a coal mine requires two or more openings to the surface. If only two openings are used, one will be used for the intake of fresh, uncontaminated air, the other will be for the return to the atmosphere of oxygen-depleted, contaminated mine air. If more than two openings are used, several may be intakes and the balance will be returns.

1. TYPES OF OPENINGS

The openings to a seam of coal used for haulage or hoisting purposes are also used for ventilation purposes. Shafts, slopes, and drifts thus form the intake and returns to mines [Figure MV-14]. Frequently the openings in any particular mine will be similar; that is, in a mine with a flat seam which outcrops, the ventilation openings may all be drifts. However not all of these openings may be used for haulage, since the ease with which rooms or entries may be driven to the surface at the outcrop will permit the use of many such openings for ventilation purposes. Mines where the coal lies wholly beneath the surface are usually developed from slopes or shafts, and these same openings are used for ventilation purposes. Frequently, the extent of deep-lying flat seam workings may be such that additional shafts for ventilation purposes must be sunk several miles distant from the main hoisting shafts or slopes.

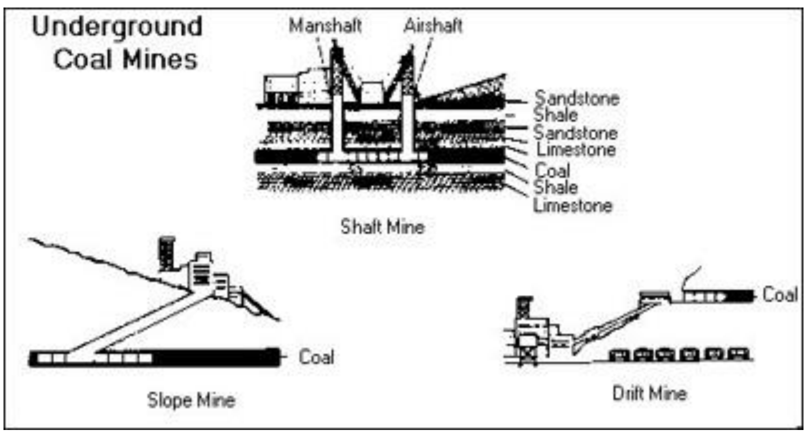


Figure MV-14: Types of Access to Underground Coal Seams
[Source: http://www.umwa.org/?q=content/types-underground-coal-mines]

2. SIZE AND CONSTRUCTION

Drift and slope openings driven in the coal are of a size which is determined by the thickness of the seam and the size of mine equipment used. The height is usually that of the seam, although in thin seams some roof or bottom rock may be removed to give sufficient headroom. The entrances to many drifts and slopes are concrete or masonry-built portals to support the loose rock or dirt above the entrance. Sometimes this concrete or masonry wall extends along the drift or slope for some distance, especially if the roof is of such a nature as to make such support necessary.

Shaft openings used for ventilation purposes may also be used for hoisting, either of coal, supplies or personal. The size of hoisting shafts is necessarily governed by the service for which the shaft is intended. In bituminous mines, mine shafts may have one, two or more compartments for hoisting coal, and the balance of the shaft area may be taken up with various pipes, electrical cables, or a stairway.

3. DISTANCE BETWEEN OPENINGS

Topography and mining conditions may require the location of intake and return openings near each other. The mining laws of the various states usually specify a certain minimum distance between these openings, such distance being dictated by experience. Pennsylvania's Bituminous Coal Mining Law requires minimum distances of 50 feet between drifts, 150 feet between slopes, and 200 feet between shafts. Preventing recirculation of mine air is in itself a good reason for keeping the intake and return openings widely separated.

4. AIR LEAKAGE

It is customary to consider the fugitive air problem of a mine as being confined to the doors, stoppings, or overcasts separating the intake and return air currents inside of the mine. There is a possibility that additional air leakage, and of a considerable quantity may exist at the intake or return openings.

5. RESISTANCE TO AIR FLOW

The velocities of the intake and return air currents at the entrance to a mine are greater than any other place unless a condition exists in the airways where the full air current must pass through a single airway with a constricted area at certain points. Keeping these velocities to a minimum so as to reduce ventilation power costs is good practice, but this can only be obtained through proper planning of these intake and return openings to fit into the ventilation scheme of the mine. Since the power required varies as the cube of the velocity, it will require nearly $3\frac{1}{2}$ times as much power to pass air down the shaft at 1500 feet per minute as at 1000 feet per minute as explained in Section 2. Velocity is only one of the factors which affect ventilation power consumption, for variation in resistance to air flow, size of cross-sectional area, and sudden turns in the ventilation opening are of sufficient importance as to require separate treatment.

In calculating the power or pressure required to pass air through an airway it has been customary to use a coefficient of friction of $0.0000000217\,\frac{lb.min^2}{(ft.)^8}$, although experimental results indicate that this figure may be 100 percent or more too high. A smooth lined airway, as well as bolted airways, may present only half as much resistance to air flow as the same airway moderately obstructed with timbers, while a heavily timbered airway may have five times as much resistance, and a rough-walled airway may have seven or eight times as much resistance. Intake and return airways, then, which are straight and have a smooth lining, will be the most economical type. Air shafts with concrete or smooth masonry linings and without hoisting compartments or stairways are the best type. Many modern hoisting compartments or stairways are the best type. Many modern hoisting shafts used as ventilation openings are so built as to have smooth concrete lining and a minimum of steel cross supports for the cage guides. Reducing shaft resistance, especially where high velocity intake and return air currents must pass down the shaft, is a paying proposition but will be obtained only through proper panning before the shaft is sunk. The same principles applied to drifts or slopes used as intakes and returns will produce similar benefits.

Resistance to air flow at intakes will be increased if the weather conditions are such as to freeze water in the shaft, drift, or slope opening and partially close to it.

Having an intake or return opening of ample size will reduce air flow resistance. As stated previously, such opening sizes are usually dictated by the other uses to which they will be put, except in the case of air shafts driven for the exclusive purpose. However, the opening at which the fan is located, either as a pressure or exhaust fan, must be a single opening, and ample size at this point will be necessary to obtain low resistance to air flow. Increasing the area by half will reduce the power consumption to less than a third of its former value, which is an economy factor worth consideration.

6. SHAFT-BOTTOM RESISTANCE

When air must make an abrupt turn, turbulence of the air is increased, the friction factor is increased, and the result is a ventilation power loss that may be considerable if the air velocity is high. This condition frequently exists at the bottom of ventilation shafts but is probably less frequent where drifts or slopes are used as ventilation openings. Figure MV3-15 illustrates this condition. In making the turn the air is crowded to the outer two-thirds of the airway and turbulence results as shown. It requires time for the air to expand to full airway width, and the "shock" loss which occurs during this period will vary according to the velocity of the air and the width of the airway. This shock loss is a drop in ventilating pressure similar to that which results when air flows through a certain length of airway, so the decrease in pressure is frequently given as equal to so many feet of straight airway of equal size.

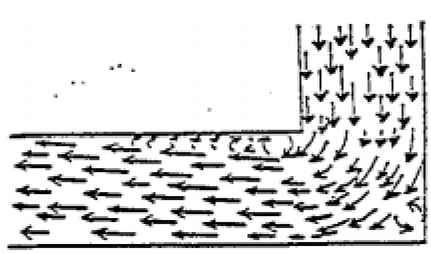


Figure MV3-15: Path of air flow at a corner

Proper planning of air shaft bottoms will do much to reduce power losses. Modern installations provide well-rounded corners where shafts and mine airways meet, usually concreted and quite smooth. Rounding off the upper corner of such a junction will reduce shock losses but will not reduce the turbulence to its lowest value. The bottom corner of the junction should also be rounded off as in Figure MV3-16. This will mean filling in the shaft bottom to provide a smooth, curved surface for the intake air.

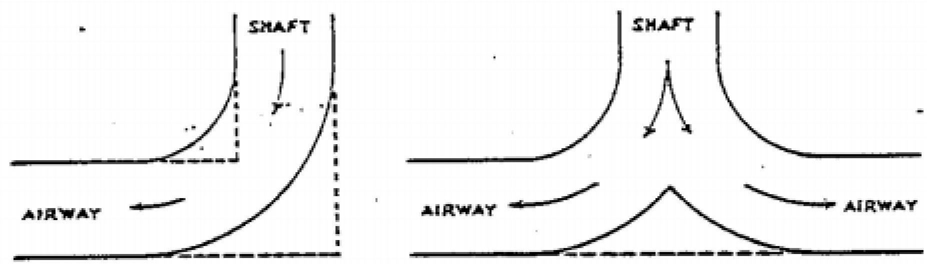


Figure MV3-16: Reducing resistance to air flow by use of rounded corners at shaft bottoms

C. INTAKE AND RETURN AIRWAYS

1. GENERAL CONSIDERATIONS

Coal mines in flat or slightly pitching seams are usually quite extensive in area as compared with mines in steeply pitching seams. It is not uncommon for haulage roads to stretch many miles across the tract of a flat seam bituminous coal mine, and this means that airways of considerable length are also necessary. Mines having large acreage are usually large producers and must necessarily employ large numbers of miners, which means that the ventilation current must have a large quantity to satisfy the requirements of the state mining laws. Flat seam coal miners, then, will need intake and return airways of a number and size which will be determined largely by the size of the coal property, its estimated production and, of course, whether the coal seam has a record in the vicinity as a gas producer.

2. MULTIPLE-ENTRY SYSTEM OF VENTILATION

As the size of the mine increase, the volume of air necessary to dilute, render harmless, and carry off noxious gases will require a large number of airways. It is quite common to find at least four or more entries being used as the main intake and return airways. Every mine shall have no fewer than five main entries connected to the openings or outlets to the surface.

Stoppings between the central entries, used as intakes, and the outside entries, used as returns, provide a separate division of the air for the workings on both sides of the mains.

Figure MV-17 illustrates one method of ventilation by a shaft.

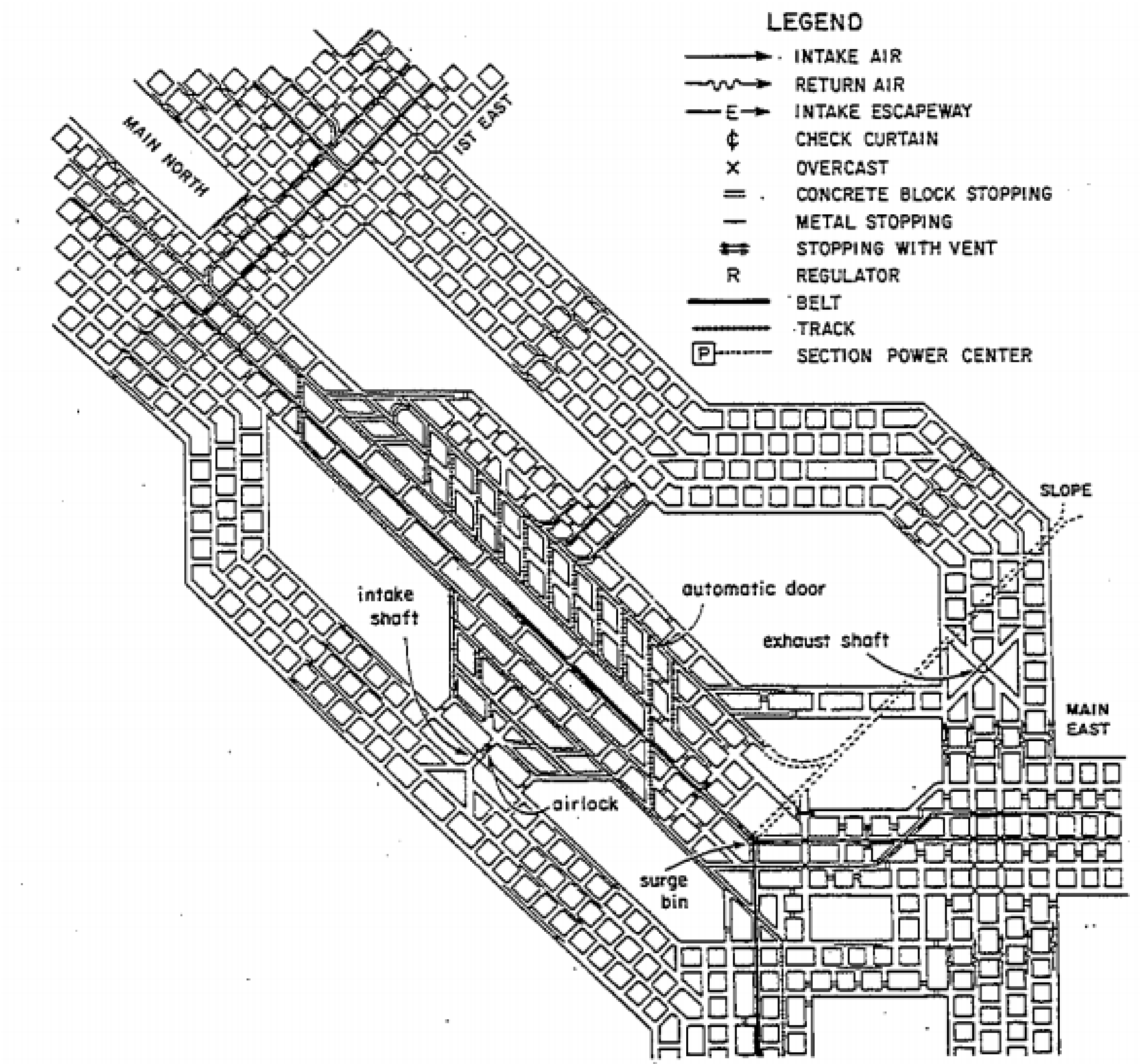


Figure MV-17: Use of shafts for intake and return openings, with multiple entry intake and return airways

3. "STREAMLINING" AIRWAYS

The section on "Ventilation Openings to Coal Mines" included a discussion on the reduction of resistance to air flow through the shafts, slopes, or drift openings. Any reduction in the resistance at these points will probably result in large ventilation power savings because all of the air which ventilates the mine must pass through these openings 24 hours a day while the mine is in operation. "Streamlining" the air passageways should not stop at these ventilation entrances and exits, for it is just as essential to reduce the resistance of the main intake and return airways and thus reduce the air velocity and the amount of power required. If no increase in the number of airways is possible, there are a number of ways in which the resistance of airways can be reduced and a reduction in power made possible.

Corners and Bends

The "crowding" effect occurs when air is compelled to make an abrupt turn at the bottom of a shaft used for air purposes. The shock loss which results from this confusion is comparable to the pressure drop when air is carried through an airway for a certain distance. This type of corner is not confined to shaft bottoms but exists at many intersections throughout any mine and, if the air passes this intersection at a high velocity, it constitutes a hidden and unnecessary power cost. If the corner could be formed into a bend, as at A, in figure MV-18 the shock loss would be considerably reduced, and this type of bend would give the least shock loss possible.

Some interesting comparisons between the type A bend and other types of bends are also illustrated in Figure MV-18. Rounding the inner corner, as at B, is not as effective as in A because turbulence still exists at the outer corner. If the outer curve crowds the inner corner, as in C, the shock loss still exceeds that of bend A. The illustration at E shows a Venturi bend, in which the square inner corner is built up so that the area between it and the outer curve is cut down to about two-thirds of the normal area and then is gradually brought up to full size. The results at E very closely approach those at A, but it is unlikely that attempts will be made to construct Venturi bends in mines because of the necessary for close measurement and the skill required. Type D illustrates what will probably occur when streamlining corners is attempted. Cutting off the inner corner gives results which are about the same as for type B.

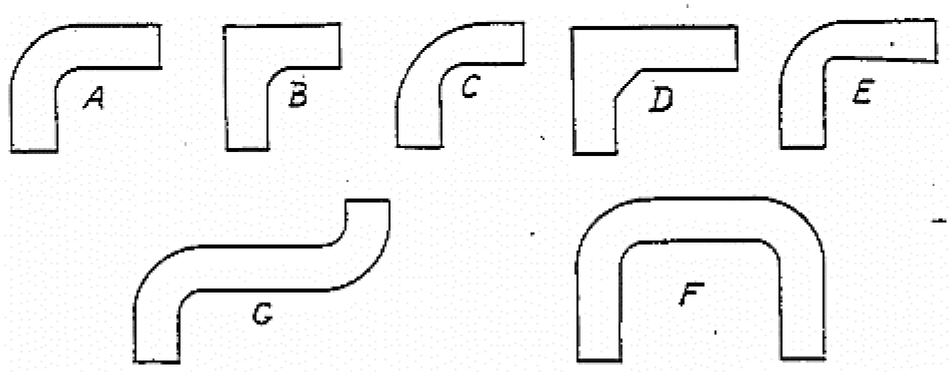


Figure MV-18: Types of corners that are typical in mine airways

Air which must pass around double bends, as at F, is subjected to less shock loss at the second bend than at the first. This is due to the air crowding the outer curve on the second bend. In the case of the reverse bend shown at G the shock loss at the second bend is even greater than at the first because the air crowds the inner curve.

Figure MV-19 illustrates a comparatively easy method of reducing shock loss where two airways meet. At A the turbulence resulting from violent mixing of the two air currents is illustrated. At B turbulence is reduced by beveling the outby corner of the entering airway. At C turbulence is still further reduced by driving the entering airway to make contact at an angle with the major airway and in the direction of air flow.

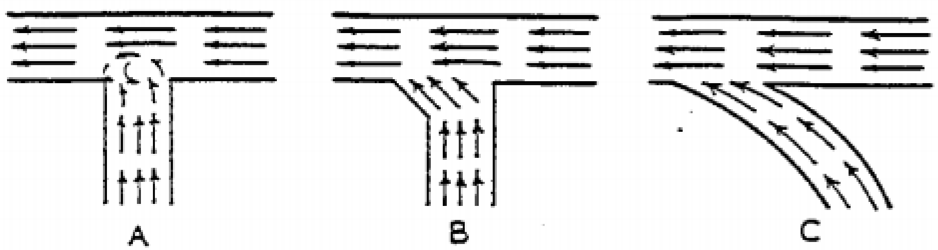


Figure MV-19: Shock loss and its reduction at airway intersections

4. TIMBER AND VENTILATION DEVICE CONSTRICTIONS

Timbering in haulageways and airways may be necessary where roof conditions are such as to require timber support. Timber sets such a contraction of the airway area which increases turbulence and increases the resistance of air flow. Tests have shown that timber sets on 5-foot centers increased the resistance to 6.5 times that of an equal length of untimbered airway. When the sets were placed on 10-foot centers the resistance was only 5 times as great. With crossbars hitched into the rib on 5-foot centers the resistance was only 3.5 times as great as for untimbered entry, and center props with short aps approximately the same resistance.

Doors, if constructed for service of considerable time, are usually supported by frames of wood with wing walls of wood or masonry. After the door has become unnecessary in the ventilation scheme of the mine it is usually removed but the frame and wing walls are frequently left standing. If this passageway carries a high velocity air current, the wing walls act as partial regulators and cause air turbulence, shock loss and an increase in the ventilation resistance. Stoppings which are partially torn down to permit the passage of air likewise add unnecessary resistance to the flow of air. Stoppings, door frames, and wing walls should be removed completely, not partially, if there is no more use for them.

Sloping the approaches to overcasts has long been recognized as good ventilation practice. However, this will not give effective streamlining to an overcast unless the top is shot down in such a way as to provide the same area of cross section over the approaches as well as over the air crossing.

Line brattices are intended to produce a strong current of air to sweep the face of a working place. As normally constructed, the air passage near the face is constricted by the placing of props and curtains across the room and at one side of the place. If the curtain is placed on the rib side of the props, a smooth lining results and there is less friction loss in the constricted zone.

Regulators contribute to the total shock loss and resistance which obstructions to the air current produce. Box regulators could hardly be streamlined or their regulating effects would be minimized. Door regulators offer certain benefits not obtained by box regulators and are a contribution to the streamlining of airways if conditions are such that they can be used.

5. TEMPORARY OBSTRUCTIONS

Obstruction of the air current by equipment produces a resistance which at times may become serious. When loaded trips are bucking the air current, the resistance to air flow will vary depending on the number, size, and spacing between cars in the trip. The end resistance of a single car represents about 75 percent of the total resistance. The total resistance of a string of cars equals the end resistance of the firs car (or locomotive) plus the frictional resistance along the sides and top of the rest of the cars. In a series of tests a single car offered about 8 times the unobstructed entry resistance, while seven cars coupled together offered only about 4 times the unobstructed entry resistance. When the seven cars were separated some distance, the total resistance mounted rapidly because each car offered end as well as side and top resistance. It will be recognized that in entries where the end area of the car is proportionately large as compared to the entry cross-sectional area, the resistance due to standing trips or to trips moving in the opposite direction to the air current will be excessively large.

There are many occasions when standing trips, left overnight or during an idle period, partially block an intake airway and add resistance to that normally offered by the mine. This most frequently occurs near shaft bottoms or drift mouths, and it could be avoided by making provision for the storage of such loaded trips where interference with the air current is at a minimum.

It is becoming more and more common to haul the coal out of mines by belt haulage. Yet, locomotive haulage is used for miner and materials transport. Therefore, impact of these on ventilation must be understood.

D. COURSING THE AIR THROUGH THE WORKING SECTIONS

1. THE USE OF AIR LOCKS IN DEVELOPMENT HEADINGS

In developing a system of entries, whether for main haulage or for room work, it may be necessary to use temporary door installations to deflect air currents and still allow the passage of men or equipment. Single door installations are hazardous because they allow leakage of air, and because they may be left open by an absent-minded or careless person thus short-circuiting the air current. Doors in development headings should always be built so that when one door is opened, another effectively checks the air current. This is called “locking” the air.

2. GOB BLEEDERS

In gassy mines the gob areas resulting from the working of pillars are a constant source of danger from the explosive gases which accumulate within them. In most gassy mines the collapse of the roof after pillar extraction is accompanied by either gas outbursts or quiet release of the gas held in the roof strata, and lack of ventilation within the gob area allows this gas to accumulate and sometimes overflow into the adjacent active workings. One method of preventing this overflow from creating too great an explosion hazard is to sweep the pillar line (that is, the edge of the pillar workings) with a strong current of air. However, at the best this method can only remove the gas along the edge of the gob area, while the accumulations of gas deep within the gob area are not affected.

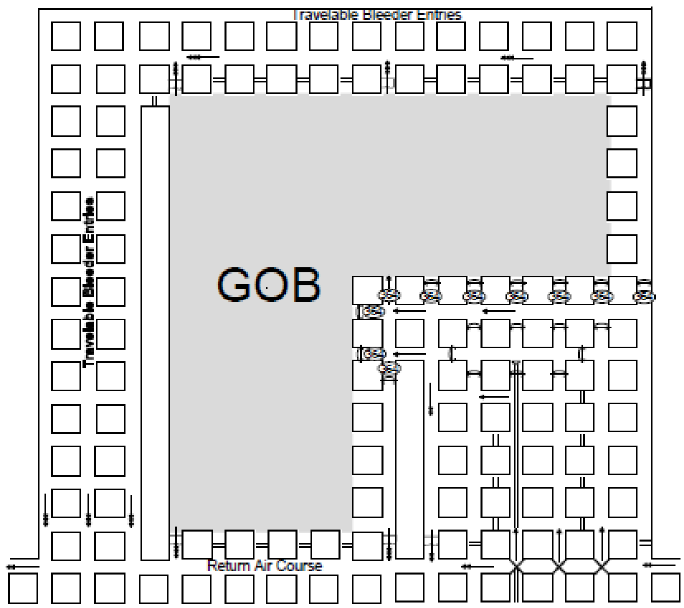


Figure MV20: Room and Pillar Section with Flow Through Bleeder Entries
[Source: http://arlweb.msha.gov/S&HINFO/TECHRPT/VENT/EXAM2.pdf]

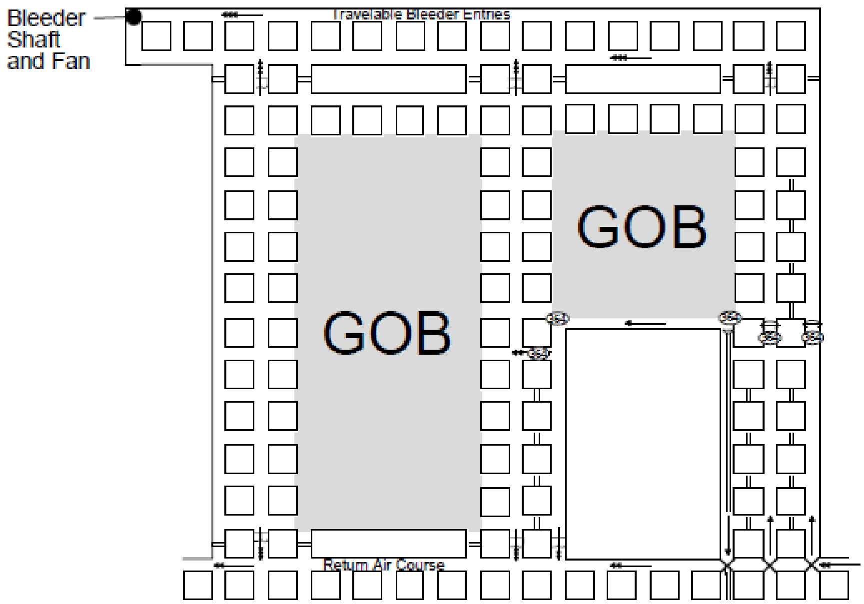


Figure MV21: Longwall Section with Bleeder Entries
[Source: http://arlweb.msha.gov/S&HINFO/TECHRPT/VENT/EXAM2.pdf]

Gob bleeders, used to overcome the tendency of gases to accumulate within gob areas, are openings from this are to adjacent return airways by means of which the gas bleeds off as it is released by the breaking of the roof strata [Figure MV-20]. This bleeding action is helped by the air current sweeping the pillar line, for a positive pressure is created by this air current which tends to sweep the gob gas towards the bleeders.

The physical condition within the gob area affects the efficiency of this bleeding action. If the rock which fills the caved area has a tendency to arch and form small openings throughout the area, then ventilating air may penetrate these openings and bleed away any gases which may accumulate. If the area is caved “tight,” then air will have trouble penetrating the area, but the likelihood of large accumulations of gas is also reduced.

In Figure MV-21 is shown the bleeder arrangement for a longwall system. The ventilation of the gob of the previous longwalls is achieved through coursing air from the gob of the active panel, through the return of the active panel and as well as directing intake air through the head gate.

It is becoming a common practice in underground coal mines to permanently seal old workings. With regard to seals, a large number of specifications are listed in 30 CFR 75.355 which are incorporated by reference in the Safety Laws of Pennsylvania for Underground Coal Mines.

Bleederless ventilation systems are being approved for use in mines, particularly where there may be such problems as spontaneous combustion in gobs. An example of a Bleederless ventilation system in a longwall in a western mine is shown in Figure MV-22.

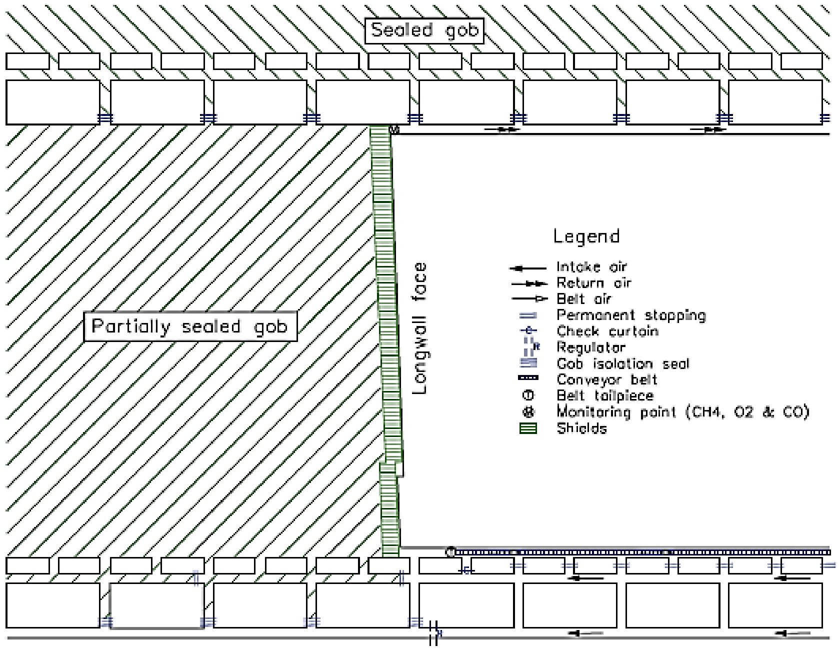


Figure MV-22: Bleederless Ventilation System
[Source: NIOSH, Information Circular 9508, 2008]

E. NATURAL VENTILATION

1. GENERAL DESCRIPTION

Natural ventilation means the flow of air, which is induced by unequal pressure that is the result of natural causes rather than having been caused by mechanical devices or other means of inducing the air to flow. The pressure which causes a natural flow of air is due to the difference in the temperatures of the air columns at the beginning and end of the natural flow system. The heavier of the two air columns (column with a lower temperature) sinks naturally to a lower level, thus forcing the air to flow towards the lighter of the two columns. These columns may exist in two shafts which are connected to mine workings, or in a shaft and slope, or in the dip and rise workings of a mine. The result is movement of the air from the point of higher pressure towards the point of lower pressure.

2. FACTORS WHICH AFFECT NATURAL VENTILATION

Natural ventilation is a condition which may be used to considerable advantage or which may be a disadvantage in mining practice. Natural ventilation, however, is not as easily controlled as mechanical ventilation, and mining today requires absolute control over the coursing of air through a mine.

The simplest form of natural ventilation is that shown in Figure MV-23, in which the principles of natural ventilation and the effect of the seasons are well illustrated. In Figure MV-23(a), the shaft connected with the mine workings is quite deep, and during the summertime the air in this shaft has a lower temperature than the outside air. Due to the greater density of the shaft air, it flows toward the drift opening and thus provides ventilation for the mine workings. If we imagine a column of air directly over the drift opening, this would be the upcast or hot column of air. The height of the upcast column in this case would be the same as the depth of the shaft.

In Figure MV-23(b), the conditions of our previous example are reversed, for now it is wintertime, the outside air is colder than the mine air, and the air in the shaft is the hot or upcast column while the imaginary column over the drift is the cold or downcast column.

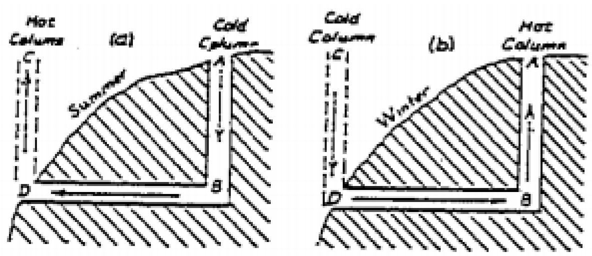


Figure MV-23(a) and (b): Natural ventilation and the effect of seasonal changes in temperature

Actually there is very little movement in the atmosphere high above the drift, but the air column which overbalances the weight of the hot column must extend to a height over the drift equal to the depth of the shaft. In this case, the fresh air flows into the drift and out of the mine at the top of the shaft.

Natural ventilation, then, is created by a difference in elevation of the two or more portals of a mine which are used as ventilation openings, and the flow of air depends on the difference in the temperatures between the outside and inside air. A change in the temperature relations will not only change the direction of flow but will also affect the quantity. Thus, the current of air supplied by natural ventilation will course through the mine workings in one direction in the summertime, and in the wintertime it will flow in the opposite direction and will vary with changing temperature difference. Also, at certain seasons the balance between the outside and inside air temperatures is such that the air will not flow in either direction, and there is no ventilation by natural means.

3. EFFECTS OF NATURAL VENTILATION ON FAN PERFORMANCE

If there is a difference in elevation between the portals used for intake and return air for a mine and if there are major differences in the temperatures of the air columns, there will be some natural ventilation taking place, and this natural ventilation pressure will either aid or oppose the mechanically induced ventilation depending on the differences in temperature between the atmosphere and the mine air.

As will be discussed in Sub-section F, the pressure which a fan develops depends on its construction and the speed with which it rotates. A fan can only develop a certain pressure at a certain speed. If the resistance of the mine is such that the volume of air which the fan is capable of passing at this pressure can flow through the workings, then the fan is considered as doing a capable job. If the resistance of the mine is increased, then the volume of air which will flow through the mine will be reduced and the pressure generated by the fan will increase. It will require speeding up of the fan to increase both the volume and the pressure that which will be required under this increased mine resistance. Likewise, the resistance offered to the flow of the air through the mine by the natural ventilation pressure bucking the fan pressure will be the same as increasing the resistance of the mine passageways to the flow of the air current, which means that the volume which will flow with natural ventilation against the fan will be decreased from the amount which would flow without the opposition of natural ventilation. On the other hand, when the natural ventilation is with the fan, it assists in getting a larger volume of air through the fan and through the mine with the fan exerting its normal pressure or rotating at its normal speed. As a rule, the water gauge of a fan remains the same or even increase slightly when the natural ventilation opposes the fan, depending on the type of fan used. With the natural ventilation current assisting the fan, the water gauge will drop slightly because part of its effective depression is consumed by drawing the excess volume of air through it.

In general, in the wintertime the natural ventilation will aid the fan because the inside air is warmer than the outside air. In the summertime the opposite condition will prevail. During those months when the inside and outside temperatures are approximately the same, the mine ventilation will be dependent entirely on the fan operation. Consideration must be given to natural ventilation and its effect on fan operation at those mines where the inlet and outlet openings have considerable difference in elevation, and also at deep mines. The ventilation which a fan will provide may be ample when the natural ventilation assists the fan, but the air delivered by the fan may not be adequate to properly ventilate the mine when natural ventilation opposes the fan.

F. MECHANICAL VENTILATION

When air flows in an airway, work is done, and work requires energy. The air horsepower required to course q cfm through an airway of resistance R is

$$AHP = \frac{Rq^3}{6350} = \frac{Hq}{6350}$$

Where H = head loss in the airway in inches of water. Air must be provided this amount of power continuously to cause it to flow without interruption. This power, under some conditions, may be provided by natural means, but it is more commonly provided by a mechanical device. In fact, according to the 1969 Coal Mine Health and Safety Act (1977), all mines are required to be ventilated by mechanical equipment. The most commonly used mechanical equipment is fans. The act further stipulates that the main fan should be installed on the surface.

There are two types of fans used in mines: the centrifugal and the axial-flow. In most modern installations, the preferred fan is the axial-flow type.

1. CENTRIFUGAL FANS

Centrifugal fans are similar in many respects to centrifugal pumps and centrifugal compressors. The principal distinction between pump and compressor is that the former handles liquids which are considered practically incompressible whereas the latter handles gases under conditions that result in their compression and appreciable change in their densities. Centrifugal fans also handle gases but they cause little change in density, and, therefore, compressibility effects are ignored in most calculations. A rudimentary centrifugal fan is shown in Figure MV-24 revolving in a clockwise direction. This is a radial-bladed fan—the blades emanate radially from the hub. Centrifugal fans can have blades that are curved opposite to or in the direction of rotation, and are known as backward- or forward-bladed fans, respectively. The action of the centrifugal fan, expressed in simple terms, is as follows: When the fan wheel (impeller), A, carrying the blades or vanes, B, revolves in its casting, the air is moved by the blades, but does not continue to rotate with them. Instead, it is thrown outward by centrifugal force to the casing, as shown by the arrow, BC, just as a stone flies off from the common sling. This displacement of air from the fan center to the circumference causes a reduced pressure at the fan center or inlet, I, toward which more air moves, to be thrown in turn to their circumference, thereby keeping up a continuous flow of air through the fan. If the casing has an opening, as at DD, to allow the air to escape, it will flow from the drift and be discharged into the atmosphere at the outlet.

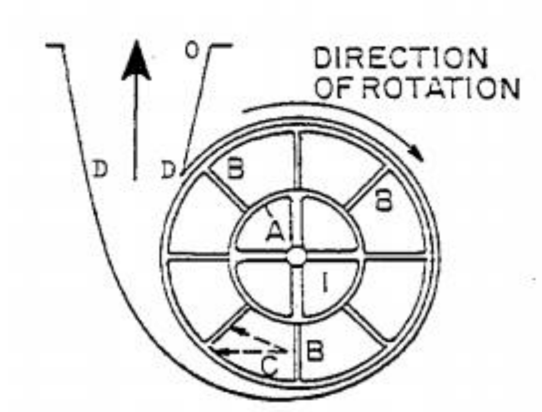


Figure MV-24: Centrifugal Fan

The Evasé of a Fan

In practice, all of the theoretical head developed by a centrifugal fan is not converted to static head. For the radial-bladed fan, only half of the theoretical head is actually available as the static head to generate the flow of air through the fan. The other half is in the form of kinetic energy in the air leaving the periphery of the fan because the air leaves the fan with a peripheral velocity of V and the corresponding velocity head is $\frac{V^2}{2g}$. If this energy is to be recovered as static head, then velocity of the air must be decreased. It is for this purpose that the fan is placed in a casting, and the air is then allowed to flow into a chimney (Figure MV-24) whose cross-section is gradually increased from the fan outlet (DD) to the point of discharge (0). This chimney is known as the evasé.

The evasé must be properly shaped; if the expansion is abrupt, i.e., the increase in area from the foot to the top is sudden, then eddy currents are set up, and the recovery is poor. Gentle slopes of the sides allow for efficient conversion on the kinetic energy to static energy.

Head Losses in the Fan

In reality however, energy is both incomplete and imperfect. Energy losses are incurred due to: air friction and shock, conversion, and recirculation (Roberts, 1960).

Air Friction and Shock Losses

As the air moves through the fan, air velocities are very high due to the narrow passages; at the same time, the air is also subjected to changes of direction. Therefore, a certain amount of the head developed is used in overcoming these frictional and shock loss conditions. This head loss appears as added heat in the air.

Conversion Losses

The theoretical head developed by a fan is in the forms of velocity and static heads. Some of the velocity head is converted to static head in the fan casting. Additional conversion takes place in the evasé. Losses during this conversion process cannot be avoided but can be reduced by the proper design of the fan casing and the evasé.

Recirculation Losses

Recirculation occurs when air leaks back from the periphery of the impeller to the center. It becomes a serious problem if the fan handles only a small fraction of the design volume whereby air tends to occupy only a small portion of the available space for flow. When the fan is operating at the designed capacity, this may not be a serious problem. Where there is recirculation, some of the pressure generated is used up in maintaining these eddies.

Fan Characteristics

The behavior of a fan under various practical conditions cannot be mathematically developed. The variables of interest in selecting a fan are pressure (total and static) it can develop, quantity of air it can move, its horsepower, and its efficiency. Horsepower and efficiency of a fan are defined with regard to the head and quantity of airflow it develops.

The head developed by the fan is the sum of two heads.

$$H_t = H_s + H_v$$

Where $H_t = total\ head\ generated\ by\ the\ fan$

$H_s\ =\ static\ head\ generated\ by\ the\ fan$

$H_v\ =\ velocity\ head\ of\ the\ fan\ at\ discharge$

Velocity head here is calculated from the velocity at discharge.

The characteristic curves of a fan include plots of the static head, the brake horsepower, and the static efficiency on the Y-axis, against the volume of air in the X-axis. The total head and the total efficiency may also be plotted. The curves are usually provided by the manufacturer for a given fan diameter, constant speed, and standard air density.

2. AXIAL-FLOW FANS

The action of the axial-flow fan is illustrated in Figure MV-25. It differs from the centrifugal fan in that the air passes axially along the fan instead of being discharged from its circumference. It consists essentially of one or more rotors, A, somewhat similar to airplane propellers. These rotors carry blades, B, and rotate at a high speed within a circular casing, C.

Air enters the casing at one end and is discharged at the other end. For high efficiency, and expanding discharge (or evasé, D) and stationary guide vanes, E, are required. The evasé recovers some of the velocity head as static head, and the stationary guide vanes take the spin out of the air as it enters or leaves the rotor.

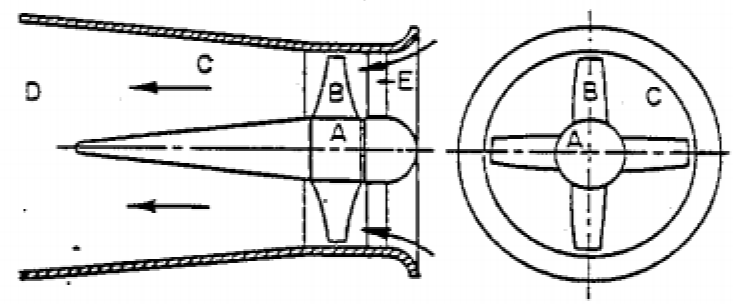


Figure MV-25: Axial Flow Fan

Although simple in construction and operation, an axial-flow fan calls for great skills in design and arrangement of the blades. The blades are airfoil-sectioned, and are mounted on a streamlined hub. The air reaches the rotating blades at a considerable velocity but it leaves them at approximately the same velocity, and the gain in pressure is essential static.

<u>Losses in an Axial-Flow Fan</u>

Since the flow in an axial-flow fan is direct with no changes in direction, shock losses are minimal. However, since the air velocities are higher than that in centrifugal fans, energy is lost in the turbulent boundary layer on the streamlined surfaces. When the fan handles small volumes of air or is near stall conditions, recirculation occurs through the rotor. Due to the narrow range of high efficiency, axial-flow fans require great care in operation.

<u>Characteristics of Axial-Flow Fans</u>

The characteristics of an axial flow fan is shown in Figure MV-26. At low volumes, the velocity of the approaching air is low and, therefore, the angle of attack is too high, leading to breakup of the streamline flow. As explained before, turbulence results, and the fan stalls.

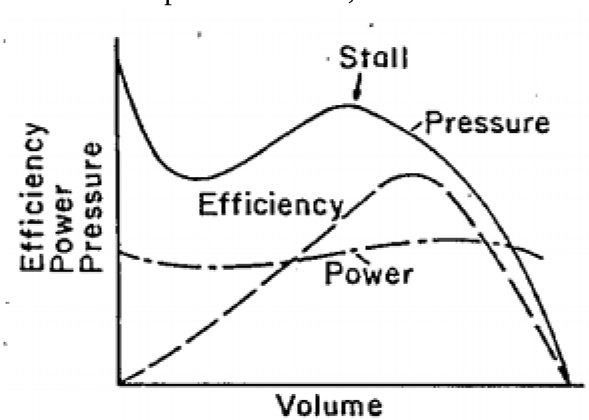


Figure MV-26: Axial-Flow Fan Characteristics

Most of the large axial-flow mine fans have adjustable blades which may be turned about on their radial axes to vary the angle of attack. In effect, this permits the angle of attack, and, hence, the pressure, to be maintained constant in the face of varying volumes caused by varying resistance. It is possible to turn the blades 20° either way before efficiency really suffers seriously. The use of adjustable blades permits an axial-flow fan to have a practical range of duty comparable to that of a centrifugal fan.

The pressure head generated by an axial-flow fan can be increased by increasing the number of rotors (or stages) in it. There is, however, a practical limit to the number of stages.

The characteristics for a 12A69 Jeffrey fan, running at 880 RPM and a density of 0.075 $\frac{lb}{(ft.)^3}$ are shown in Figure MV-27. This fan can be operated at several blade settings (e.g., 3F, 4E, and 5D). The most efficient operation (80% static efficiency) of the fan at 880 RPM occurs between the quantities of 110,000 cfm and 4' W.G. and 160,000 cfm and 6" W.G. Fan characteristic curves supplied by the fan manufacturers are usually limited to the recommended operating range of the fan as shown here and give no indication of the stall zone or information on operation in the low head range of the curve near free delivery (quantity at zero pressure). As a result, operation near the stall zone of a particular fan is hard to evaluate on the basis of information supplied by the manufacturer.

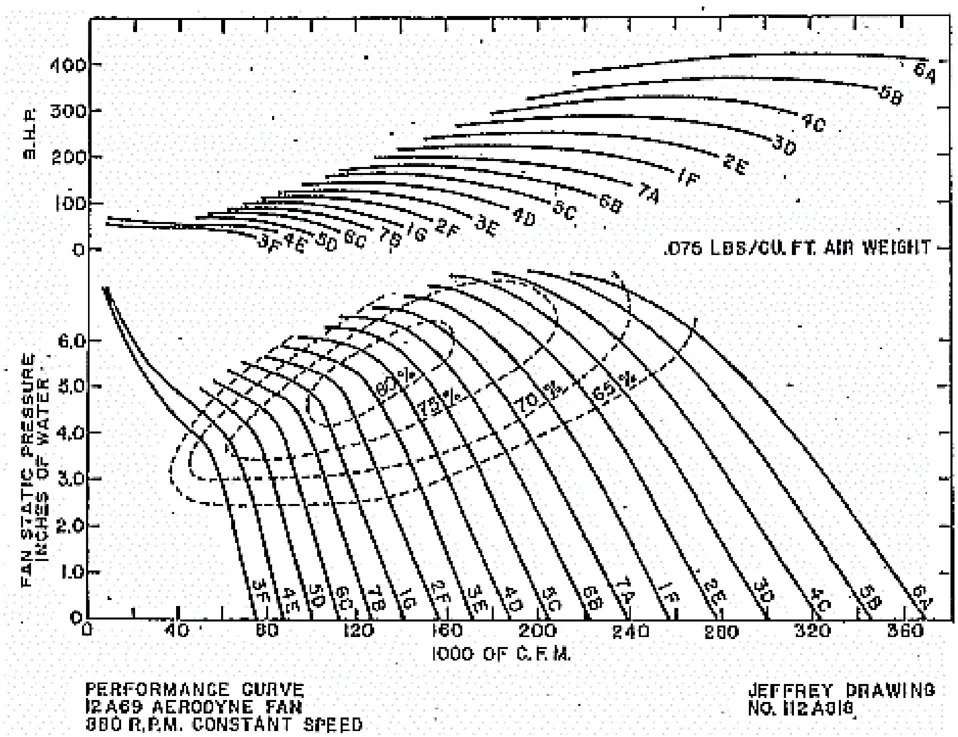


Figure MV-27: Axial Flow Fan Characteristics With Various Blade Settings

3. FAN AND MINE CHARACTERISTICS

When a fan is running at constant speed, it can produce only those combinations of head and quantity that appear on the head-quantity curve of the fan. The mine characteristic is the relationship between head and quantity that provides the heads required to force different quantities of air through the mine. When the fan and mine characteristics are plotted on the same graph, these intersect and their point of intersection is known as the operating point.

Consider the mine characteristic (B) shown in Figure MV-28, and the fan characteristic (A) shown superimposed on it. These two characteristics intersect at 0. From the proceeding discussion about the mine characteristics, it is known that, for a flow of 100,000 cfm, 5" W.G. will be needed. It is also known that at the point 0, the fan will produce a head of 5" W.G. and a flow of 100,000 cfm. The conclusion is that the fan with characteristic A will force through a mine with characteristic B, 100,000 cfm at 5" W.G.

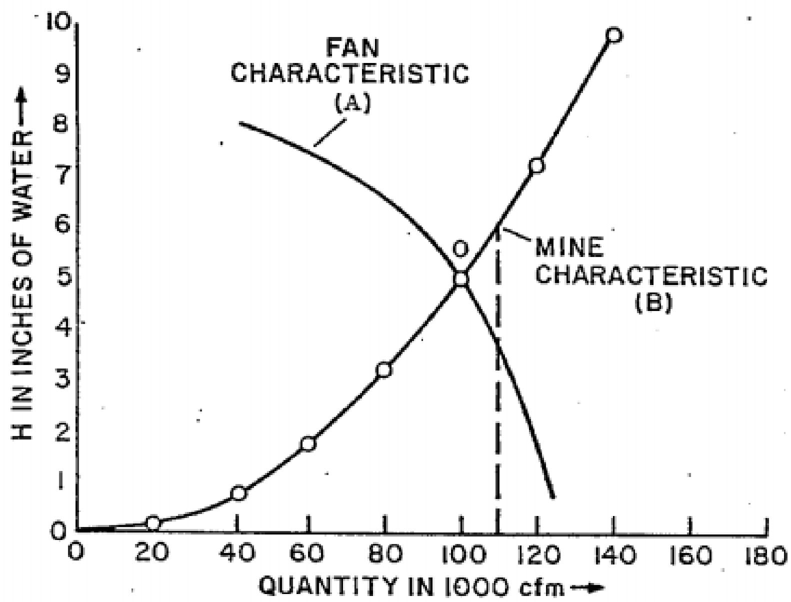


Figure MV-28: Mine and Fan Characteristics

To avoid low efficiencies and/or unstable performance, the intersection of the mine and fan characteristics curves should occur along the upper part of the fan characteristic to the right of the point where the fan will stall. If the measured operating point of a fan does not fall on the fan characteristic curve supplied by the manufacturer, and the fan performance is stable, the fan is operating on the descending, low-efficiency portion of the curve.

4. FAN APPLICATIONS

There are three distinct applications of fans in mine ventilation—main fans, booster fans, and auxiliary fans.

Main fans, as the name implies, are the principal fans used to produce the general ventilation air current in mines. They are permanent in nature and are large in capacity. According to the 1969 Coal Mine Health and Safety Act, the main fan must be located on the surface.

Booster fans are usually installed to ventilate parts of the mine where the main fans may, by themselves, be inadequate.

Auxiliary fans are usually of small size, portable, and temporarily installed to ventilate working places through which the general ventilating air must not otherwise pass (blind entries). It is a common practice to conduct the air from the fan to the working place by a duct.

Main Fan Installation

The main fans in coal mines must be placed on the surface. The primary advantages of locating a fan on the surface are:

(a) Accessibility—the fan is accessible in the event of underground disasters such as fires and flooding.
(b) Safety—the fan is less likely to be damaged in the event of underground disasters such as explosions and it is not subjected to ground movement or roof falls.
(c) Ease of installation—installation of a fan underground requires that the workings be advanced; also the size of the fan to be used underground is constrained by space limitations, and a complicated network of airways is necessary to channel the air into the fan drift.

When the main fan is installed on the surface, unless care is taken, there can be a considerable leakage of air, i.e., as much as 20% of the air quantity handled by the fan may be circulated through the fan back to surface rather than going underground.

Exhausting vs. Forcing Fans

The fan has been defined as an air pressure-generating source. The pressure it generates is always given relative to a fixed reference pressure, usually the atmospheric pressure. Depending on the location of the fan in the mine circuit, the static and total pressures generated by it relative to the atmosphere can be positive or negative.

For example, consider a mine ABC as shown in Figure MV-29, where A and C are the tops of the two shafts connecting the workings at B. It is desired to take intake air from A through B, and then return it from B to C. A fan may be placed at A or C to generate the necessary pressure to ensure flow from A to C through B.

When the fan is placed at A, i.e., at the top of the intake shaft, it is known as forcing fan. When it is placed at C, i.e., on the top of the return shaft, it is known as an exhausting fan.

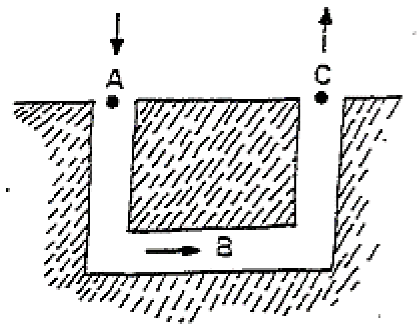


Figure MV-29: Simple Flow Circuit

Forcing System

In the forcing system, the fan inlet is connected to the atmosphere and the fan outlet to the mine intake shaft. The air increases in velocity and static head as it flows through the fan and eventually returns to the surface through the return shaft. Since air flows from a point of higher total pressure to one of lower total pressure, the absolute pressure measured at any point in the mine is higher than that of the atmosphere.

Exhausting System

In an exhausting system, the fan inlet is connected to the top of the return shaft, and the fan outlet is connected to the atmosphere usually through an evasé, a chimney. When the fan is operating the air on the inlet side is drawn into it, increases in velocity and static head and flows through the evasé into the atmosphere. The absolute pressure on the inlet side is lower than that of the atmosphere. As the fan continues operating, the air to it is continuously fed from the return shaft whose supply is drawn from the mine. The air lost form the mine to the return shaft is replenished from the surface through the intake shaft.

Booster System

The pressure gradient form a booster system is a combination of the gradients of the exhausting and forcing systems. 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. Note that the total pressure curve is always above the static pressure curve. However, the total and static pressures on the discharge side of the fan are greater than those on the inlet side. Because of this, unless the two sides of the fan are well isolated, air can recirculate from the discharge through the workings back to the fan inlet. In the United States booster fans are not allowed in coal mines.

On theoretical grounds, therefore, there is very little difference between the blowing and exhausting systems. Forcing fans handle clean air from the atmosphere which is denser than the hot return air. In a blowing system, the gases are held back in the gobs. While this is an advantage when the fan is operating, if the fan is stopped due to any reason, the workings may be flooded by gases from the gob.

On practical grounds, therefore, exhausting fans are preferred for coal mine ventilation. It is best not to locate the fan at the top of a shaft used for worker and material transport so that the surface air lock to the fan need not be disturbed at any time. Also, the main access and exit to the mine must be through the fresh intake air so that in case of emergency, these can be accessed from the intake side. More importantly, the workers can escape through the intake escapeway. To protect the main fan on the surface from the force of an explosion, it should be offset from the nearest side of the mine opening with explosion doors (or a weak wall) installed in the direct line of the possible explosion force (Figure MV-30, MV-31 and MV-32). The fan must be equipped with a pressure-recording gauge and an automatic signal stop. In the event of a fan stoppage, electrical power is disconnected from the mine and personnel are evacuated.

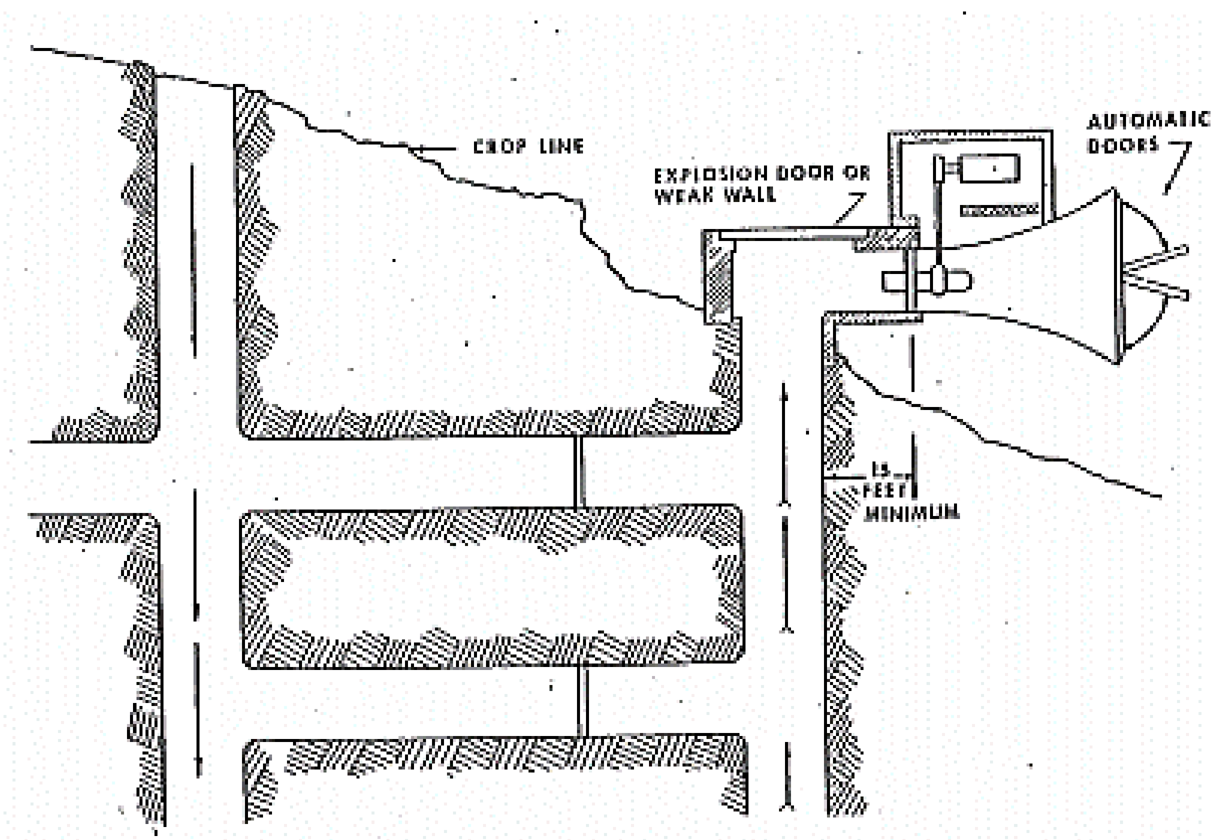


Figure MV-30: Diagram for Main Fan at Drift or Slope

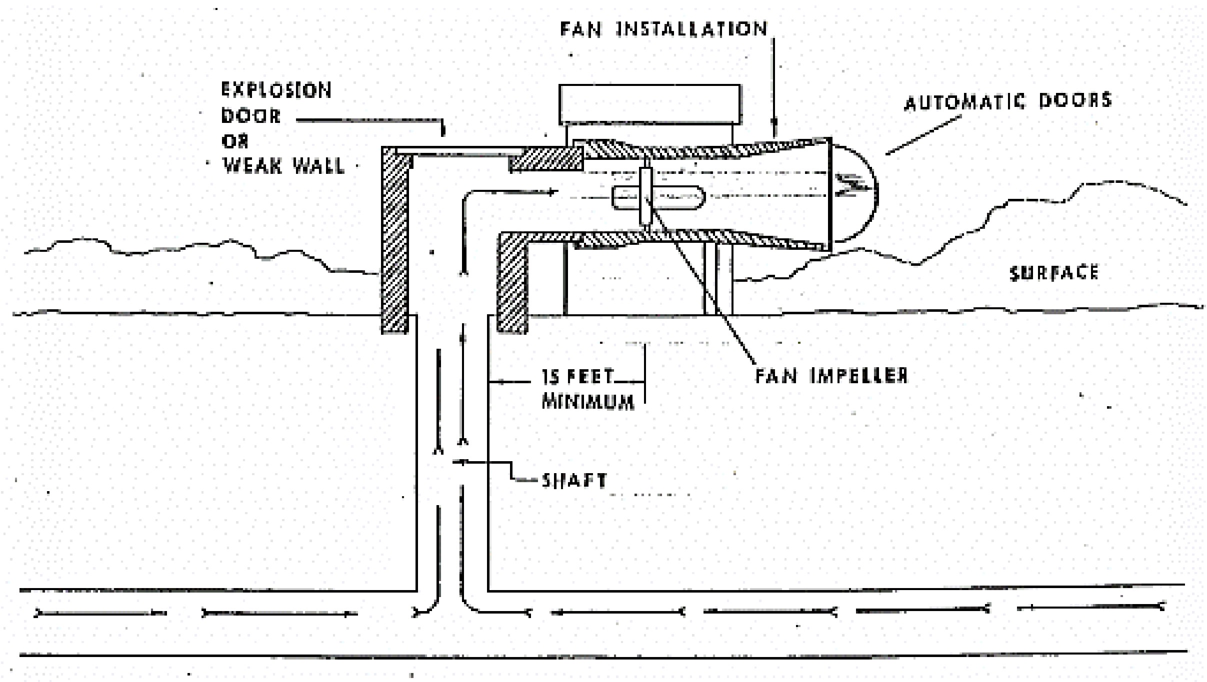


Figure MV-31: Diagram of Main Fan at Shaft

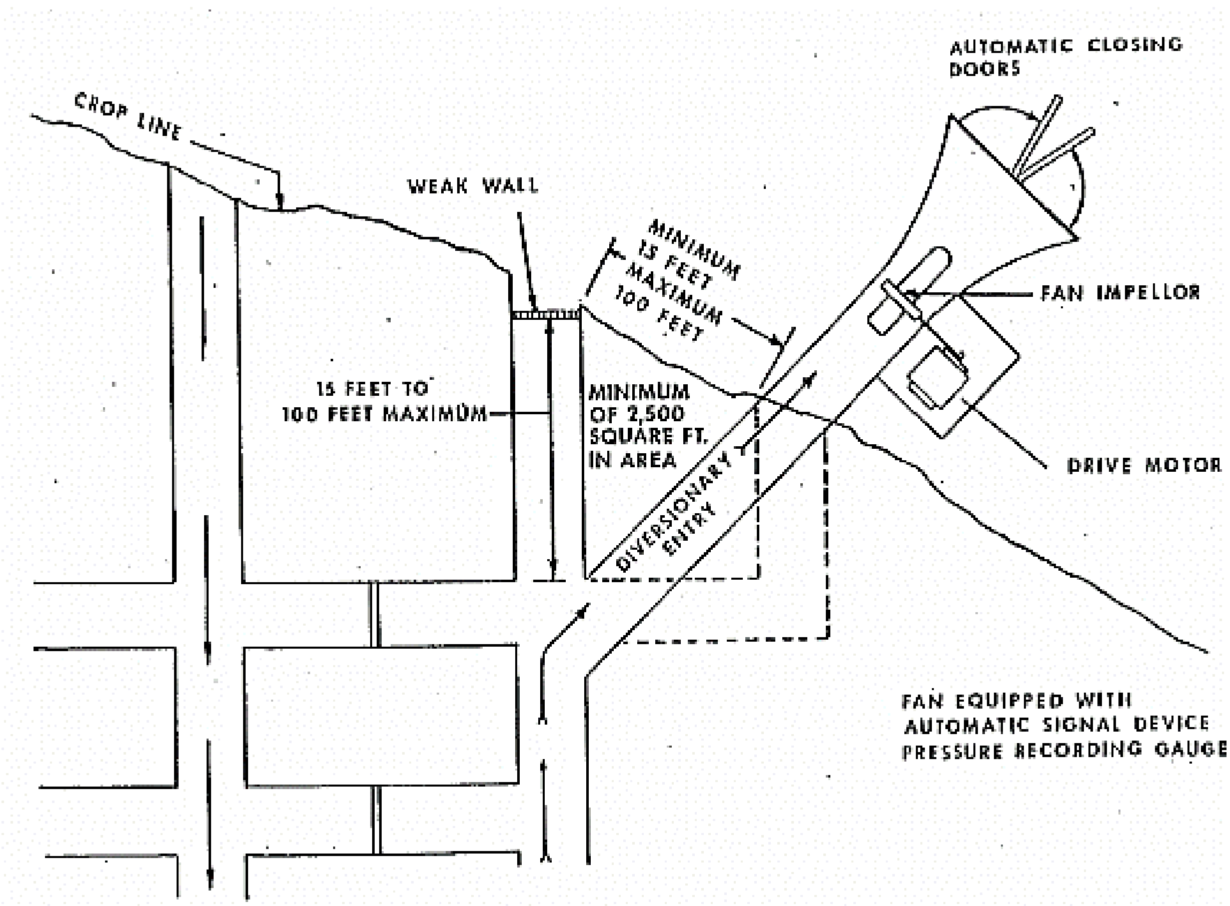


Figure MV-32: Diagram of Main Fan and Diversionary Entry

The fan structure as it will be seen in a mine, the symbol for a fan and the manner in which a fan is shown on a mine map are illustrated in the Figure MV-33.

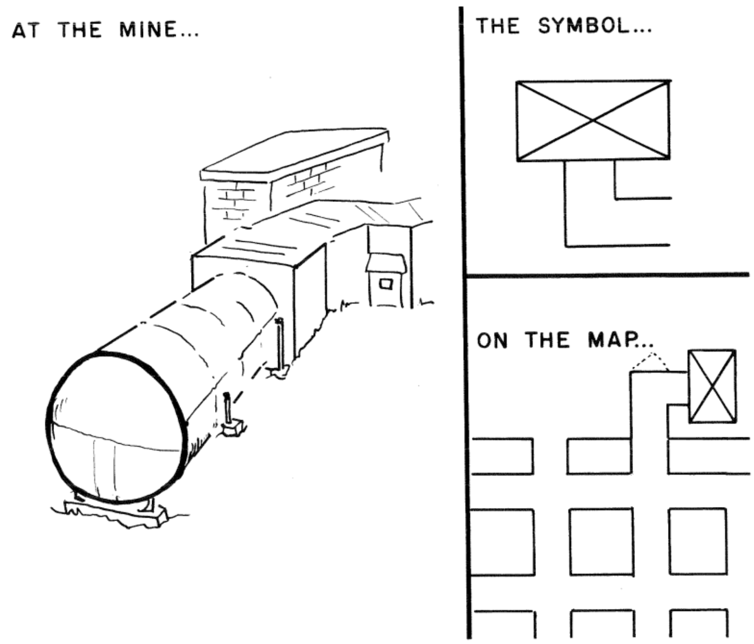


Figure MV-33: Fan House, Fan Symbol and Fan Location on a Mine Map

5. REVERSAL OF AIR CURRENT

In the event of a stoppage of the primary fan or one fan in a multi-fan system, the 1969 Health and Safety Act requires that provisions be taken to prevent adverse reversals of air current. In some countries, it is required by law to provide arrangements for the reversal of air current if such a need arises. With centrifugal fans, reversal of the air current requires a complicated arrangement of doors and ducts. With axial flow fans, reversing the direction of the rotor movement will ensure that air will flow in the opposite direction though the volume flowing is reduced by 50 to 60%.

G. FACE VENTILATION

Unless the air is properly distributed to the face, the mine ventilation system is not performing its primary function. While it has always been recognized that this last part of ventilation is the most important, it is also the most difficult to achieve.

There are basically two methods of ventilation the blind entries ahead of the last open crosscut: The use of line brattices or the installation of auxiliary fans. Each technique has its defenders as well as its outspoken critics.

1. LINE BRATTICES

The line brattice is essentially a space divider or temporary partition made of an impervious material that is installed and maintained very carefully and kept as close to the face as possible. Its purpose is to guide the airflow through the face area and last open crosscut and into the return. Brattices were formerly (and to some extent still are) made of untreated jute, but nylon reinforced plastics and similar materials are more commonly used in them today. While the more effective material is invariably higher in initial cost, it results in lower overall expense in that it allows for greater reuse and less air leakage.

The line brattice is installed so as to split the heading longitudinally and thus provides an inlet as well as a return from the face to the last open crosscut, Figure MV-34. Since the mining machine must have room to maneuver on one side of the brattice, it is not practical to split the entry evenly, so a wide side is provided for the machine. The air may be brought up the narrow side and, after it sweeps by the face, returned on the wide side, as in the blowing system shown in Figure MV-34, or it may be forced in the reverse direction as shown in the exhaust system shown in Figure MV-35. Since the blowing system produces a high velocity of air at the face, it achieves superior gas dilution, but the air, now contaminated with gas and dust, returns over the machine and its operator. As a result, this system is rarely used today. The more commonly employed exhaust system, with intake air coming in on the wide side of the brattice and returning on the narrow side, eliminates this problem because the fresh air passes over the machine operator before it reaches the face. However, since the air velocity provided at the face by the exhaust system is low, it does a less effective job of diluting the gas there. In fact, the corner of the face opposite the end of the brattice can easily gas up, so it is imperative that the end of the brattice be maintained no farther than 10 feet from the face.

Obviously, the line brattice must be higher than the thickness of the seam and must be installed tightly at the top with wedges on roof bolts, headers, or posts, or fastened directly to dowels driven into the roof for that purpose. The use of telescoping tubes (called pogo sticks) attached to grommets in a nylon-reinforced plastic brattice can be very effective. Where there are high ventilation quantities and exhausting canvas is used, brattice boards attached to pogos at bottom and centers have been necessary to prevent the canvas’s collapse against the rib. Experiments are being conducted in developing slide-bar brattice extenders that will make it possible to advance the line brattice relatively easily as the mining machine advances. Conventional line brattice construction interferes with machine operations and thus it is frequently kept too far back from the face to be really effective.

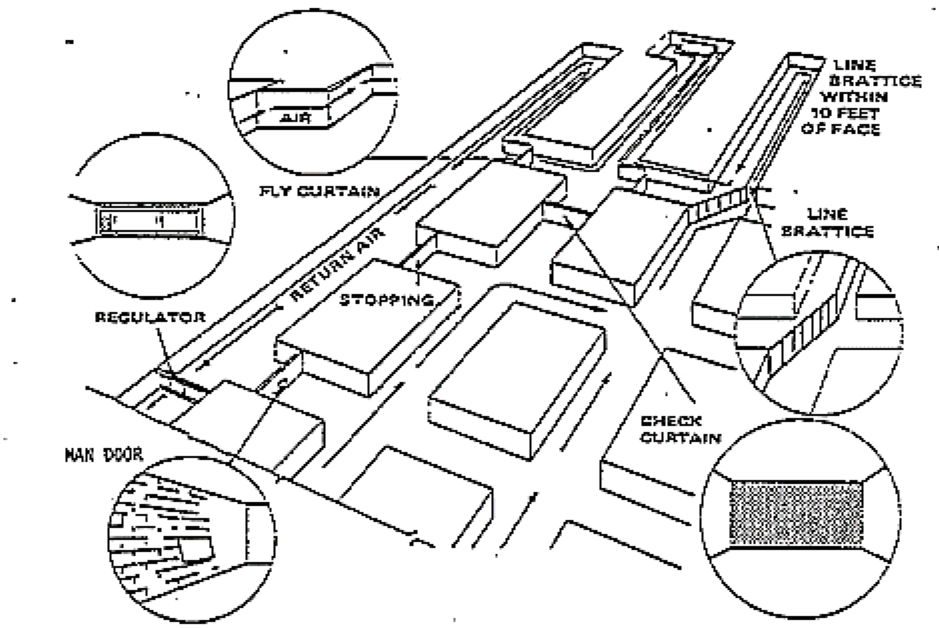


Figure MV-34: Example of the use of a blower line brattice

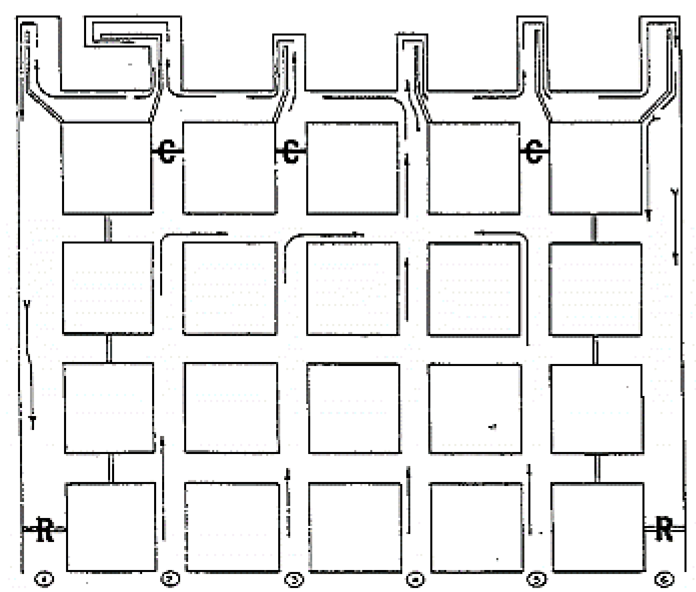


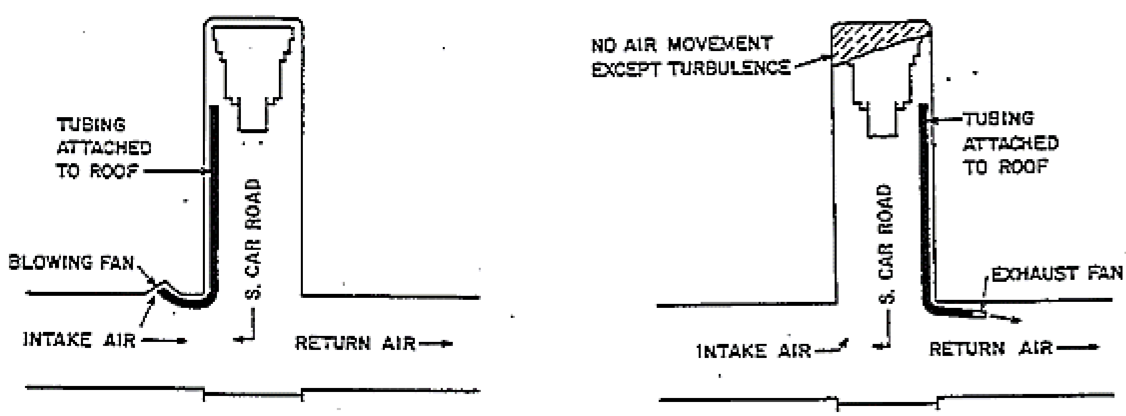
Figure MV-35: An exhausting line brattice system

2. AUXILIARY FANS

While auxiliary fans have not achieved the popularity they deserve in coal mines, they are not necessarily a panacea for face ventilation. As a concept, the idea of hanging tubing out of the way while providing an adequate flow of air to the face appears very attractive. However, when one begins to calculate the pressure and horsepower required by an auxiliary fan, one soon realizes that the tubing needs to be much larger than originally visualized. Also, it must be very carefully installed and maintained to minimize leakage and kept close to the face. However, it may be easier to keep the tubing closer to the face than it is with a line brattice.

Low-quality (5000 to 20000 cfm), high-head (2 to 15 in.), centrifugal or axial-flow fans, ranging in size from $2\frac{1}{2}$ to 26 HP are used for face ventilation purposes. Flexible tubing, 30 inches in diameter, made of jute or nylon, with rubberized coating, containing helical or spiral springs, and available in 25, 50, and 100 ft. lengths or smooth fiberglass tubing is utilized in auxiliary systems. Various arrangements of tubing and fans have been used, but the trend is to higher quantity, pressure, and horsepower with larger tubing.

A blower system can be installed as shown in Figure MV-36 (A). There is no problem in delivering more than 3000 cfm of air through 18 in. diameter tubing with reasonable pressure and horsepower. Even with the end of the tubing as far as 35 ft. from the face, adequate dilution can be achieved with methane liberations of as much as 20 cfm. The disadvantages of such a blower system are that dust and gas laden air passes over the machine operator, and a methane buildup can occur outby the machine. Because of the difficulty of controlling dust, this technique is seldom used today.



A. Blower Fan B. Exhaust Fan

Figure MV-36: Face ventilation with auxiliary blower (A) and exhaust fan (B)

The exhaust system shown in Figure MV-36 (B) eliminates the passage of contaminated air over the machine operator, and effectively removes the dust from the working environment. However, even when an airflow of 5000 cfm is maintained through an 18 in. tubing 100 ft. long and ending within 7 ft. of the face, it is not uncommon to have the far corner gas up. When the tubing is allowed to lag back from the face 15 to 20 ft., it is not uncommon for the entire face to gas up. By this time, the operator is between the end of the tubing and the face and is exposed to contaminated air, so this is a situation that must not be allowed to occur.

Gas control and dust control are, to a great degree, incompatible with respect to velocity requirements because the exhaust system is good for dust control and poor for gas control while the reverse is true for the blower system. The obvious answer is to utilize the combination system shown in Figure MV-37 in an attempt to take advantage of both the good diffusion characteristics of the blower and the good dust collection characteristics of the exhaust. However, it is still a poor compromise, since the blower may stir up too much dust, and a buildup of gas may occur at the back of the machine because the difference in airflow between the exhaust and blower capacities is the only portion of the total air that travels from the mouth of the place to the face. Of course, the inconvenience of installing and maintaining two vent-tube systems is another major disadvantage of the dual system.

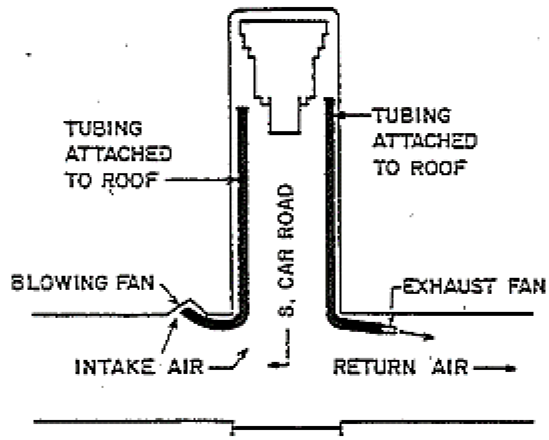


Figure MV-37: Combination Auxiliary Fans

The combination of a diffuser fan mounted on the mining machine and an exhaust fan as shown in Figure MV-38 achieves optimal results without the inconvenience of installing and maintaining two lines of vent tubing. A small, hydraulically driven, centrifugal fan powered by the miner’s hydraulic system is mounted directly on the machine. With as little as 650 cfm of air from 6 in. diameter tubing directed toward the face, it is possible to provide an air flow of 3300 fpm in the far corner of the face and effectively force contaminated air into the larger air volume circulated by the exhaust fan. By circulating 4000 cfm through a 100 ft. length of 14 in. tubing, methane liberations of up to 40 cfm have been safely handled. With this system, recirculation can occur during normal operation if the exhaust tubing is allowed to lag behind, and thus the end of the tubing should always be kept well ahead of the inlet to the diffuser fan. Also, if the exhaust fan stops for any reason, and the machine continues to operate, recirculation will occur around the diffuser fan with possibly a dangerous accumulation of gas.

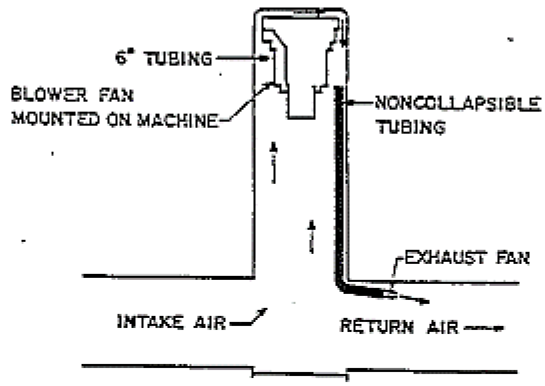


Figure MV-38: Machine-mounted diffuser fan

One more possible auxiliary fan system is shown in Figure MV-39. Here a water-injected dust collector is mounted in the throat of the mining machine. A 20 hp motor provides 3500 cfm of air through 100 ft. of 17 in. tubing that extends from the machine to the return air. Since the inlet of the tubing is located about 8 ft. from the face and in the center of the machine, poor ventilation of the corners is achieved by this arrangement, although it is considered highly successful in at least one mine.

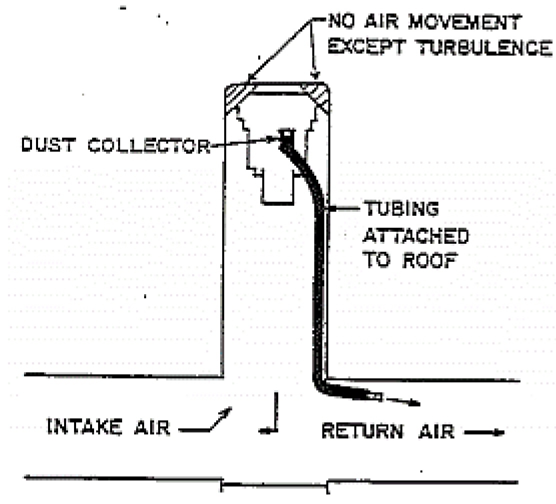


Figure MV-39: Machine-mounted dust collector and exhaust fan system

It would appear that by using three inlets, an additional one in each corner of the machine, and a larger fan, a more effective flow of air could be obtained. Also the location of the unit makes it vulnerable to damage by the passage of large lumps through the machine. This is a good example of the kind of difficulty encountered in modifying an existing machine and underscores the necessity of designing such systems into the machine. The problem of float dust in the return airways is minimized with this arrangement. Also, the ability to carry rock dust into the high-velocity discharge far down the returns before deposition is an added possible benefit with an exhausting fan system.

There are some basic problems with auxiliary ventilation that have limited its use. To prevent recirculation, the law requires that a quantity of air $2\frac{1}{2}$ times its capacity pass by the auxiliary fan. The other problem is that there is no ventilation to the face during a power failure or an idle shift so that under these conditions, it is imperative to install a line brattice immediately. But, if fan and tubing are properly matched, the tubing is aligned carefully, and the joints are kept airtight, an auxiliary system has a greater potential for maintaining a safer working environment at the face than a line brattice. While there is little reason for a leakage greater than 25% with the use of tubing, it is not uncommon to have losses of 95% in 100 ft. of line brattice where poor installation and maintenance prevail.

TABLE OF CONTENTS

VENTILATION SECTION 4

VENTILATION MEASUREMENTS

It is often necessary for a mine official to make measurements of the ventilation parameters in a mine. Among the most important of these parameters are: velocity, pressure across openings [differential pressures], the dry and wet bulb temperatures, the pressure developed by a fan, the gas [%] in the workings, and fan performance . From these and similar measurements, the mine official has to calculate several important parameters to determine the peformance of the ventilation system and to make changes to the ventilation system. Among the important factors that can be calculated from the basic measurements are such things as the quantity of air flowing, the amount of gas in the airstream, the performance of the fan, the relative humidity of the air, the size of the regulator to course a certain amount of air, and the amount of additional air needed to dilute the gas to the maximum allowable concentration. In this section, the following topics are covered: velocity measurement, pressure measurement, temperature measure, gas measurement, fan operation and monitoring, and ventilation maps and symbols. Some of the most important instruments and methods used for ventilation measurements are described along with important ventilation calculations.

A. VELOCITY MEASUREMENT

Velocity of the air in a mine airway is a fundamental property. There are several instruments that are used to measure the velocity of air in an underground mine opening. These instruments are based on certain principles and therefore only suitable in a particular velocity range. Some of the most commonly used instruments and their range are listed below:

Instrument	Velocity Range (fpm)
Smoke tube	20-120
Anemometer	150-2,000 (also 2000-10,000)
Velometer	30-3,000
Kata Thermometer	100-1,500
Pitot tube	750-10,000

Often, it is necessary to have a stop watch to note the time over which a velocity measurement is being made.The width and height of the mine opening or entry are measured with a suitable tape measure.

Once the velocity [v, fpm] at a cross-section has been determined, and the height [h, ft] and width [w, ft] have been measured, the area of the cross-section and the quantity of air flowing can be calculated:

Area, sq. ft. = width x height = wh = a

Quantity, cu. ft. per min. = velocity x area = va

1. ANEMOMETER

The anemometer is the most common instrument used for measuring velocity of the air in a mine entry. It is a simple instrument with which, with some training and experience, good velocity readings can be easily obtained in the range of velocities that are found in underground airways.

Figure MV-40: Anemometer [Davis Instruments Mfg. Co.]

As shown in Figure MV-40, the instrument itself is a vaned propeller attached to a shaft which is geared to record the lineal distance travelled by the air in the time for which a reading is taken. A clutch on the top of the instrument allows engaging and disengaging the gear, respectively, at the start and end of a velocity measurement. The instrument shown [Davis] has a main dial and two small dials. The main dial has a scale from 0 to 100, further divided into tens and ones, the dial to the right has a scale from 0 to 1000, further divided into units of 100's and that to the left, a scale from 0 to 10,000, divided into units of 1000. Thus the instrument is useful to measure lineal distance travelled from less than 100 ft. to up to 10,000 ft. Instruments with three to five smaller dials are available to measure higher velocities. The units in some anemometers are in meters.

For example, assume that at the start of a measurement in an underground airway, the above instrument dials were set to "zero" and the instrument was held in the airstream for a time period of three [3] minutes. Suppose the following are the positions of the

indicators on the dial at the end of three minutes: the main dial reads 57, the right dial, between 400 and 500 and the dial to the left, between 2000 and 3000. The lineal distance travelled is 2457 ft. Therefore, the measured average velocity of the air is [2457 ft. /3 minutes] = 819 ft. per minute. It is common for the manufacturer to provide a calibration chart with the anemometer to correct the measured reading to a true reading. The actual method used for making a velocity measurement and for applying the correction is described later in this section.

Shown in Figure MV-41 is a Taylor anemometer with its own carrying case with a calibration sheet. When an anemometer has been in use for a period of time or is suspected to have been damaged, it is a good practice to send it to the manufacturer for getting it repaired and a new calibration sheet.

Figure MV-41: Taylor Anemometer with Case and Calibration Sheet

2. MAKING A VELOCITY MEASUREMENT WITH AN ANEMOMETER [MSHA]

The standard method for performing a velocity measurement using a rotating vane anemometer involves a "**traverse**" of the cross-section to be measured. In a traverse, the anemometer is swept across the cross-section of the air course, or portion of the air course, in a controlled, steady motion. It is important to traverse the cross-section because air travels faster in the center of the air course than along the roof, walls, or floor. A single measurement at a fixed position in the air course would not be representative of the average velocity across the entire cross-section.

Figure MV-42 illustrates various traversing practices, depending on the dimensions of the air course cross-section to be measured. In small air courses, such as in a coal mine, the entire air course can be traversed. In larger air courses, it is preferable to divide the air course in half and traverse each half separately (split traverse). When performing split traverses, the velocities determined for both halves are averaged to determine the average air velocity for the air course as a whole. If the air course has a very large cross-sectional area and a high roof, divide the air course into four equal quarters with the upper edge of the lower quarters being at mid-wall height.

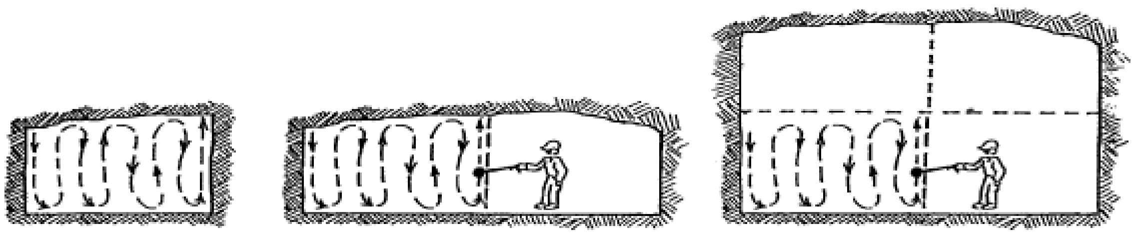


Figure MV-42: Traversing for Velocity Measurement

The anemometer can be attached to an extension handle when making a traverse. This allows the instrument to be held some distance away from the body to minimize air turbulence effects on the instrument reading. It is also helpful to use an extension handle when measurements must be made at locations with a high roof. If an extension handle is not available, the instrument should be held at arm's length away from the body when traversing. When traversing, it is necessary to ensure that the anemometer is always perpendicular to and downstream of the air flow. The traverse motion must be steady, and all parts of the area of the cross-section must be covered equally, including corners, walls, roof, and floor.

A suitable anemometer and a timer are required to make a measurement. If a stopwatch is not available, a watch that measures in seconds is acceptable. It is easier to perform a measurement with two people - one to perform the traverse and one to operate the stopwatch and provide verbal cues - but it can be accomplished by one person. Normally, the timing period for a traverse is one minute (to make subsequent calculations easier), but any convenient time period is acceptable. Measuring air velocity with a rotating vane anemometer involves the following basics:

(1) "Zero" the anemometer dial with the appropriate levers;
(2) Position the instrument at a corner (e.g., wall/floor or wall/roof) where the velocity is slowest;
(3) Allow the anemometer's vanes to reach full speed (a few seconds), then simultaneously start the stopwatch and release the dial movement to begin measuring the traverse;
(4) Simultaneously stop the stopwatch and the dial at the end of the traverse;
(5) Record the anemometer dial reading and the elapsed time from the stopwatch;
(6) If either traverse was not fully completed, do not use that measurement. Repeat until two good traverses are completed that agree to within 10% of the lower of the two readings. Again, this method is easiest if the time period is 1 minute. The resulting two readings would then be averaged, and that value recorded;
(7) If split traverses were performed, repeat the above steps for the other half of the air course.

The dial reading on a rotating vane anemometer reads in feet or meters. Air velocity is obtained by dividing the anemometer reading by the time measured on the stopwatch. If the traverse is completed in exactly 1 minute, the dial reading is equal to the air velocity in feet per minute (fpm). For example, if the final anemometer value is 655 feet, and the traverse was completed in exactly 1 minute, the velocity would be:

$$\frac{655\ feet}{1\ minute} = 655\ feet/min = 655\ fpm$$

If the same traverse was completed in 1 minute, 10 seconds, the time would first need to be converted to minutes, as follows:

$$1\ minute, 10\ seconds = 70\ seconds$$

$$\frac{70\ seconds}{60\ \left(\frac{seconds}{minute}\right)} = 1.17\ minutes$$

$$\frac{655\ feet}{1.17\ minutes} = 559.8\ \left(\frac{feet}{min}\right) = 560\ fpm$$

The next step in determining velocity is the velocity correction. Every anemometer is calibrated and provided with a velocity correction table [See Table MV-3]. Measured velocities must be corrected using the correction factors provided on such tables. Listed correction factors are designated either "+" or "-." Factors that are designated with a "+" are added to the measured velocity. Factors that are designated with a "-" are subtracted from the measured velocity. For example, using the data in Table MV-3, a measured velocity reading of 200 would be corrected to 215 ft. If the measured velocity is not on the table, the correction factor must be "interpolated," or estimated. For example, if the measured velocity is 340 fpm, the correction would be about -2. If the measured velocity is 1655 fpm, the correction would be about -104. If a split traverse measurement were performed, the two corrected velocities would be averaged to determine the overall corrected average air velocity for the entire air course cross-section.

Serial No. 123456 Calibration Date: 01-28-2015			
Est. Vel.	**Correction**	**Est. Vel.**	**Correction**
50	+65	500	-15
100	+42	600	-22
150	+29	750	-36
200	+15	1000	-68
250	+8	1250	-79
300	+2	1500	-90
350	-3	1750	-113
400	-8	2000	-131
When sign is : + Add - Subtract			

Table MV-3: Calibration Chart for Anemometer

Note that the correction factors table provided here is only an example and is not to be used for correcting actual velocity measurements from every anemometer. Every anemometer is individually calibrated and provided with its own applicable correction factor table.

Example MV-25: In the anemometer in Figure MV-43, there are three dials in addition to the main dial. The main dial has a range from 0-100 meters, the dial to the right, from 0 to 1000 meters, the dial at the bottom from, 0 to 10,000 meters, and to the left, from 0 to 100,000 meters. What is the reading on the instrument? Assume that no calibration chart is available and the reading took 157 seconds, calculate the velocity in meters/second.

Figure MV-43: Anemometer Reading for Example MV-25

Solution: Note that the left dial is at zero, the bottom dial is between 2 and 3, indicating that the reading is between 2000 and 3000 meters, the right dial is between 2 and 3, indicating that the reading is between 200 and 300. The main dial is reading 51. Therefore, the reading is 2,251 meters.

The velocity in meters per second = [2251 meters/157 seconds] = 14.34 meters/second. Anemometers with digital read out capabilities, such as the one shown in Figure MV-44 are particularly useful in underground environment. However, they must be permissible for use in underground coal mines.

Figure MV-44: A Digital Anemometer

3. SMOKE TUBE METHOD

The smoke tube method is commonly used for determining the presence of moving air, the direction of airflow and the approximate velocity in low velocity airflows. The method is based on the mechanical effect of air on the smoke. The smoke is generated using an aspirator and a smoke generating tube as shown in Figure MV-45. To make a measurement, it is necessary to have two persons, a tape measure and a stop watch.

Figure MV-45: Smoke Tube and Aspirator [MSHA]

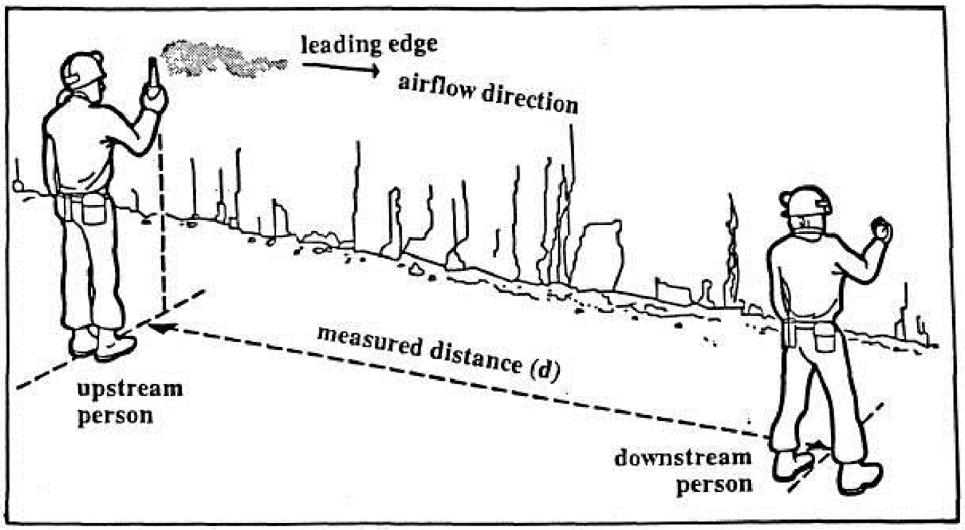


Figure MV-46: Smoke Tube Method for Velocity Measurement {MSHA]

The method of making a measurement consists of the following steps [Figure MV-46]:

[1] Mark a station upstream as the smoke release station.
[2] Mark a station downstream as the station at which the time will be recorded for the smoke to travel from the upstream station to downstream station.
[3] Measure the distance [d, ft.] between the upstream and downstream station.
[4] When the upstream person releases the smoke, the downstream person starts the stopwatch and monitors the travel of the smoke from the upstream station to downstream station.
[5] Stop the stopwatch when the leading edge of smoke reaches the downstream station.
[6] Note the time it has taken for the smoke to travel from upstream to downstream station. Let us say, it is "t" seconds.
[7] Calculate the velocity knowing the distance, d, and the time, t.

As an example, if the distance, d, is 15 ft., and the observed time, t, was 12 seconds, then the velocity of the air is:

$$\left[\frac{15\,ft.}{12\,seconds}\right] = 1.25\,ft.per\ second = 1.25\,x\,60 = 75\,ft.per\ min.$$

The average velocity in the airway will be less than this as this is the velocity of the leading edge. In practice, the average velocity will be about 90% of the calculated velocity.

$$Average\ velocity = 75\,x\,0.9 = 67.5\,ft.per\ min.$$

Steps 4, 5, 6 and 7 will be repeated to get several readings so that a reliable average velocity can be obtained.

To get accurate readings, it is necessary to have a regular cross-section of the airway, the distance between the two stations should be so chosen that the smoke cloud does not break up but stays somewhat intact, and the observer at the downstream station can see properly both the release of the smoke and the arrival of the leading edge of the smoke.

If the airway had a cross-section, 20 ft. wide and 5.5 ft. high, the area is 110 sq. ft. and the quantity of air flowing in the entry is:

$$67.5\ x\ 110\ =\ 7425\ cu.ft.per\ min.$$

4. VELOMETER

The Velometer (see Figure MV-47)is a direct-reading instrument that provides an instantaneous velocity measurement, but since it is somewhat cumbersome to traverse a cross-section with a Velometer, it is more commonly used to spot-check the velocity at a fixed point such as at the duct inlet or with special attachments , the duct interior.

Figure MV-47: Velometer with probes

B. PRESSURE MEASUREMENT

It is common in mining to measure both absolute pressures and differential pressures. An example of the absolute pressure that is measured is the atmospheric pressure. Differential pressures of interest are those that exist across stoppings or doors or between any two points in the mine.

1. ATMOSPHERIC PRESSURE MEASUREMENT

The most common instrument used to measure atmospheric pressure is a mercury barometer. The most common type of mercury barometer is known as cistern barometer which consists of a cistern and a tube filled with mercury [Figure MV-48]. A calibrated scale is attached next to the tube. As the atmospheric pressure increases or decreases, the height of the mercury level in the tube changes. At sea level and at a temperature of 85.4°F, the height of the mercury column over the mercury in the cistern is 29.92 in. This is equal to an air pressure of 14.70 lbs. per square inch [psi].

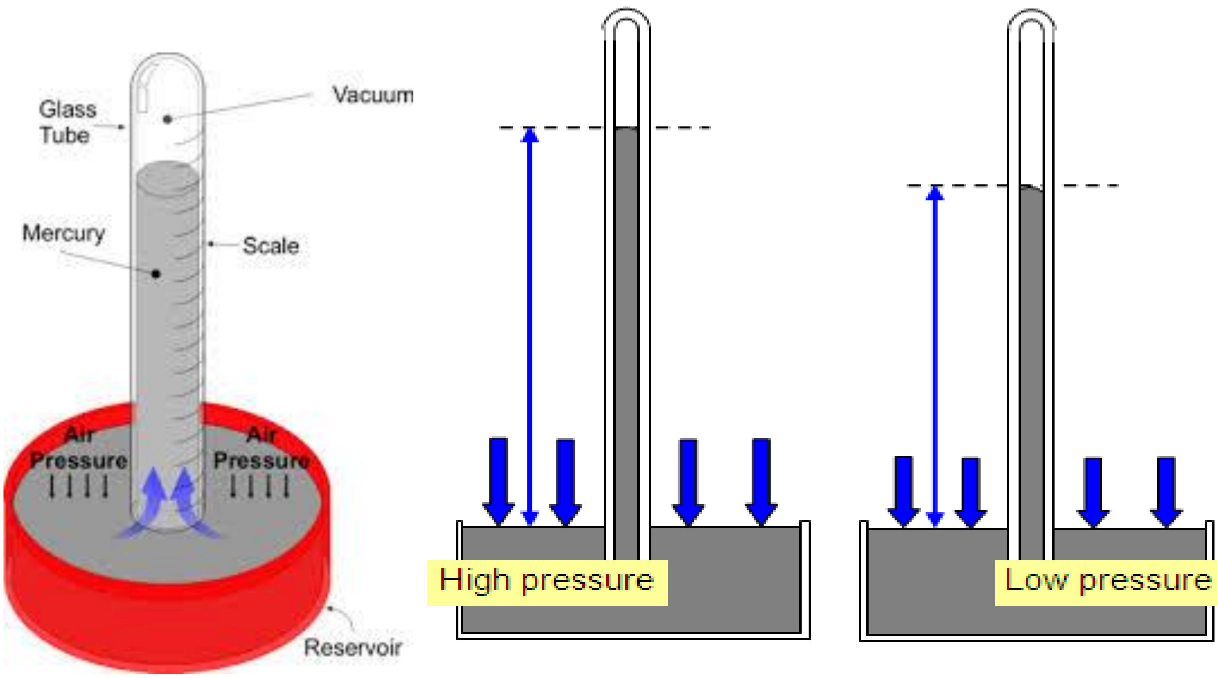


Figure MV-48: Cistern Type Mercury Barometer – Principle of Operation

The mercury barometer is an accurate but a rather delicate instrument to carry around and it takes a long time to stabilize after being positioned in a location to take a reading. Therefore, the instrument is permanently wall-mounted at a convenient location in the mine office or fan-room to keep track of the atmospheric pressure and the pressure trend.

An aneroid barometer is a widely used instrument for atmospheric pressure measurements due to its ruggedness and portability. It consists of an airtight, flexible diaphragm from which most of the air has been evacuated so that the pressure inside the diaphragm is less than atmospheric. The collapse and motion of the diaphragm is resisted by a spring element and when changes in atmospheric pressure tend to move the diaphragm, the movement is magnified by a system of levers that are connected to a pointer and a scale that is calibrated in terms of atmospheric pressure. Since atmospheric pressure is related to the elevation or altitude of a point, aneroid barometers can be can calibrated to measure the altitude of a location above a fixed datum. Such aneroid barometers are called altimeters and are widely used in underground ventilation pressure measurements. Shown in Figure MV-49 are the photographic views of the Wallace & Tiernan altimeter and the dial face. The altimeter pressure reading is in units of "ft. of air" and can be converted to traditional ventilation units of lbs. per sq. ft. or inches of water knowing the density of air.

In mine ventilation, it is customary to use the term standard air to refer to air that is at sea-level conditions, 70 degrees Fahrenheit, and dry. The density of standard air is 0.075 lb. per cu. ft. [lb./cft].

The relationship between the density of a fluid [w, lb./cu.ft.] and the pressures expressed in height of the fluid [h, ft.] and in lbs. per sq. ft. [p, psf] is as follows:

$$P = w \; x \; h$$

Recollect relationships between the various units of pressure such as lbs. per sq. ft. [psf], lbs. per sq. inch [psi], inches of water, inches of mercury, and ft. of air.

$$1 \; psf = 0.0069444 \; psi$$
$$1 \; in. of \; w.g. = 5.2 \; psf$$
$$1 \; in. of \; mercury = 70.73 \; psf = 0.491154 \; psi$$

Example MV-26: Calculate the pressure in inches of water gage [in. of w.g.] that is equal to a pressure of 20 ft. of standard air, given that the density of standard air is 0.075 lb./cft.

Solution: First calculate the pressure in psf that will be equal to the pressure of a 20 ft. column of standard air and then use the relationship between psf and w.g.

$$P \; in \; psf = 0.075 \; x \; 20 = 1.5$$
$$P \; in \; w.g. = \left[\frac{1.5}{5.2}\right] = 0.288$$

Example MV-27: What is the height of a column of standard air [density = 0.075 lb./cft] that will be equal to 1 in. of w.g.?

Solution: Note that 1 in. of w.g. = 5.2 psf. Therefore, the height of the standard air column [h, ft.] can be calculated as follows:

$$P \; in \; psf = 5.2 = 0.075 \; x \; h$$
$$h = \left[\frac{5.2}{0.075}\right] = 69.33 \; ft.$$

It is common to state that a 70 ft. height column of standard air has a pressure at the base of 1 in. w.g.

Figure MV-49: Wallace & Tiernan Altimeter [left] and dial face [right].

Example MV-28: What is the pressure in psf equal to 1 in. of mercury at zero degree Celsius?

Solution: Note that mercury [at zero degrees Celsius] is 13.6 times as heavy as water. As temperature increases, mercury becomes less heavy. For example, at 25 degree Celsius, it is 13.545 times heavier than water. At 50 degree Celsius, it is 13.472 heavier than water].

$$Specific\ gravity\ of\ water\ =\ 62.4\ lb/cft$$

$$Specific\ gravity\ of\ mercury\ =\ 13.6\ x\ 62.4\ =\ 848.64\frac{lb}{cft}$$

$$Pressure\ in\ psf\ equal\ to\ 1\ ft.of\ mercury\ =\ 848.64$$

$$Pressure\ in\ psf\ equal\ to\ 1\ in\ of\ mercury\ =\ \left[\frac{848.64}{12}\right]\ =\ 70.72$$

Example MV-29: What is the height of a column of standard air [h, ft.] that has the same pressure as 1 in of mercury? Note that 1 in. of mercury = 70.72 psf and the density of standard air = 0.075 lb./cft.

Solution: $$I\ in.of\ mercury\ =\ 70.72\ psf\ =\ 0.075\ x\ h$$

$$h\ =\ \left[\frac{70.72}{0.075}\right]\ =\ 942.93\ =\ 943\ ft.$$

Example MV-30: Dry air at a temperature of 32 degrees Fahrenheit and a barometric pressure of 29.92 inches mercury has a density of 0.0807 lb./cft. What is the height of an air column equivalent to one inch of mercury column?

Solution: $$1\ in.of\ mercury\ =\ 70.72\ psf\ =\ 0.0807\ x\ h$$

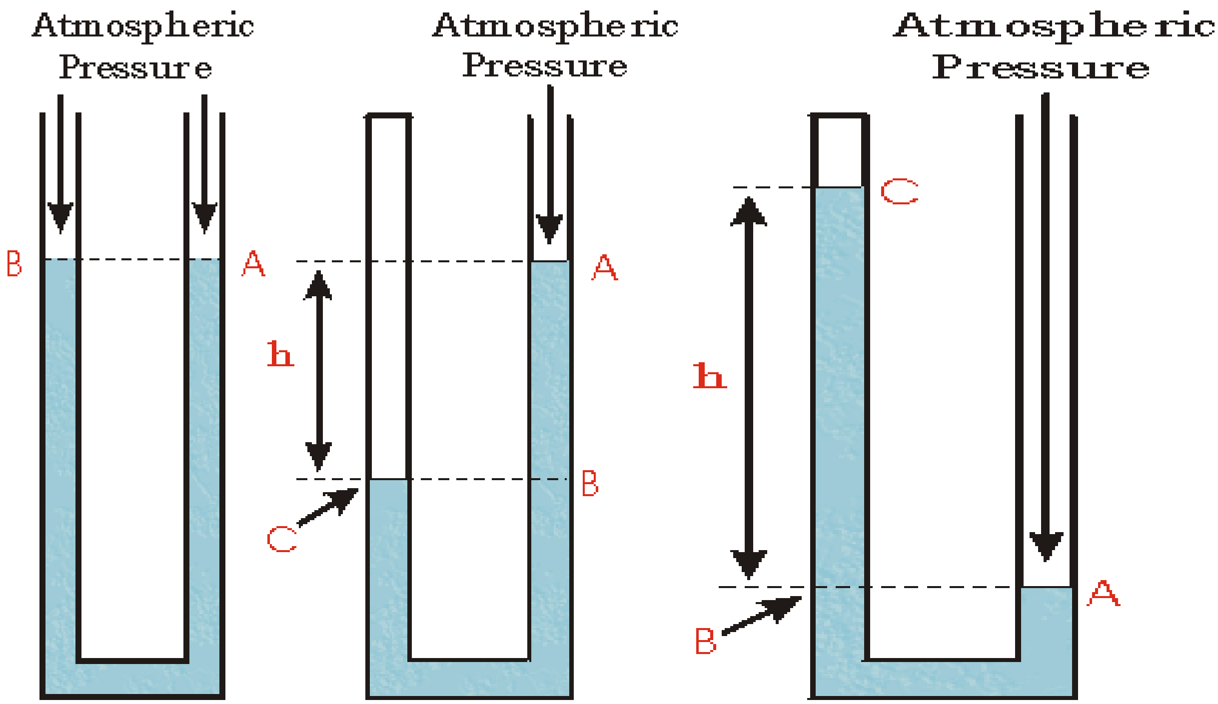
$$h\ =\ \left[\frac{70.72}{0.0807}\right]\ =\ 876.33\ =\ 876\ ft.$$

2. MANOMETER

A manometer is an instrument that can be used to measure absolute pressure [such as a barometer] or differential pressure. Though it is one of the earliest pressure measuring instruments, it is still widely used for pressure measurements due to its simplicity of construction and ease of use. There are many types of manometers in the market today, ranging from the simple U-tube manometers to digital ones with computer interfaces. The common manometer consists of a U-shaped tube filled with water or other similar fluid as shown in Figure MV-50. A graduated scale [say in inches] located in-between the two limbs can be read and will be the measure of the pressure differences in inches of the fluid in the manometer. If the fluid is water, the pressure measured is in inches of water. If it is mercury, then it is in inches of mercury. In figure MV-50[a], the ends of the U-tube are open to the atmosphere and the liquid heights in both the limbs of the U-tube are at the same level. For differential pressure measurements, each limb of the manometer is connected to a different pressure source. When one limb of the manometer is open to the atmosphere and the other limb is connected to a pressure source, the differential pressure measured is the gage pressure, which may be positive [above atmospheric] or negative [below atmospheric].

Consider the case where the differential pressure between the atmosphere and a pressurized point [e.g. a point in the fan drift] is to be measured. In Figure MV-50[b], the left limb is connected to a port on the discharge drift of a forcing fan [and therefore, has a higher pressure than atmosphere]. The pressure in the discharge drift pushes the fluid up the right limb. The difference in the liquid levels, denoted by ‘h’, is the differential pressure between the pressure source and the atmosphere. It is the amount by which the pressure of the source is higher than the atmospheric pressure. If Figure MV-50[c], the left limb is connected to a point which has a lower pressure than the atmosphere, such as say a port in the fan drift an exhaust fan. Here, the atmosphere pushes down on the liquid in the right limb. As before, the difference in the liquid levels in the limb, denoted by h, is the pressure by which the pressure in the fan drift is lower than the atmosphere.

Manometer can be used a barometer if one limb is open to the atmosphere and the other limb is connected to a vacuum. The differential pressure measured is the absolute pressure of the atmosphere.



[a] [b] [c]
Figure MV-50: Manometer

Manometers that use liquid are available with other forms of arrangement [e.g. Well Type Manometers, Inclined Tube Manometers] for use under varying conditions of service such as very high pressure or extremely low pressure measurements. U-tube manometers have several disadvantages for routine underground mine measurements. They are large and bulky and lack portability. They need leveling, and condensation inside the tubes may present problems. These manometers are more suited for permanent installation in the fan house or for laboratory use rather than for field applications.

3. MAGNEHELIC MANOMETERS

The Magnehelic manometer [Figure MV-51] is more like an aneroid barometer and consists of a diaphragm held by a spring. A magnet is attached to one side of the diaphragm and adjacent to the magnet is a helix that is free to turn to maintain a set distance between itself and the magnet. Any pressure difference across the diaphragm displaces the magnet and rotates the helix. The position of the helix is indicated by a pointer on a dial graduated in pressure units [psi, psf, inches of water, inches of mercury, etc.]. The ports of the Magnehelic manometer are labeled “high” and “low.” The pressure source with higher pressure is connected to the “high” port and the other to the “low” port and the reading on the dial is the differential pressure between the two sources. Magnehelic manometers are available in a wide range of scales and being a relatively small instrument, one can pack several Magnehelics with different ranges to cover the scope of pressures that may be encountered in a mine. For accurate measurements, it is necessary to ensure that the instrument has been calibrated. Further, prior to any measurement, the procedures must always include [1] zeroing the instrument, [2] orienting it correctly [either vertical or horizontal] as required, and [3] connecting the pressure sources correctly.

Figure MV-51: Magnehelic Manometer [Range 0 – 2 inches of water]

Digital manometers [Figure MV-52] are micro-processor based instruments that are suitable for field use with capabilities for transferring measured data. Shown in the figure below is a photograph of a handheld digital manometer. For underground coal mining use, it is necessary to ensure that the digital manometer is permissible.

Figure MV-52: Digital Manometer

4. PITOT TUBE

The Pitot-static tube, commonly called the Pitot tube, is a widely used device for measurement of high velocity in mine applications. The device, as shown in Figure MV-53 in a wind tunnel, consists of two concentric tubes bent in an L shape. The inner tube is open at ends, one end identified as the Pitot port and the other end open which can be connected to the high pressure limb of a manometer. The outer tube is closed at the Pitot port end but perforated with small openings a short distance back from the Pitot port. It is open at the other end which can be connected to the low pressure limb of the manometer.

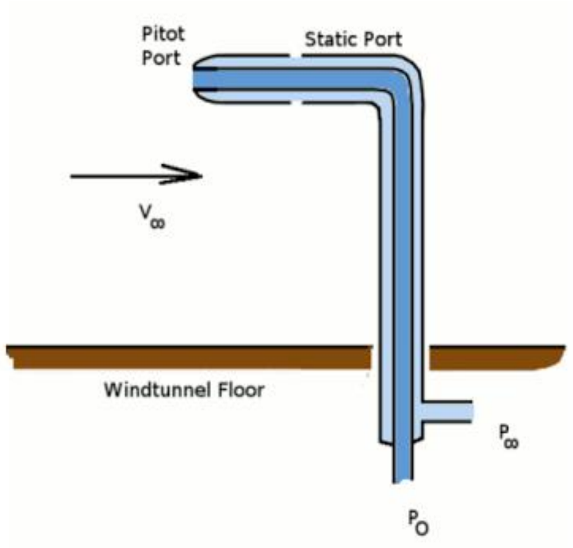


Figure MV-53: The Static Pitot Tub

Pitot tubes have many applications. One end of the tube can be open to the atmosphere and the other end to the pressure source. Shown in Figure MV-54 below is the Pitot tube and the manometer connections to measure the velocity pressure [VP] of the flowing air at a point in the airway [e.g. fan drift]. The Pitot port measures the total pressure [TP], the static port, the static pressure [SP] and the difference is the velocity pressure. Therefore, the manometer reading will represent the velocity pressure of the flowing air at the opening of the Pitot tube. If the manometer reading is in inches of water, as discussed in Section 1, the velocity can be calculated from the formula:

$$P_v = 5.2\ \left[\frac{v}{4000}\right]^2$$

$$i_v = \left[\frac{v}{4000}\right]^2$$

$$v\ =\ 4000\ x\ \sqrt{i}$$

where P_V is the velocity pressure in lbs. per sq. ft.
V is the velocity in ft. per min.
i_V is the velocity pressure in inches of water gauge.

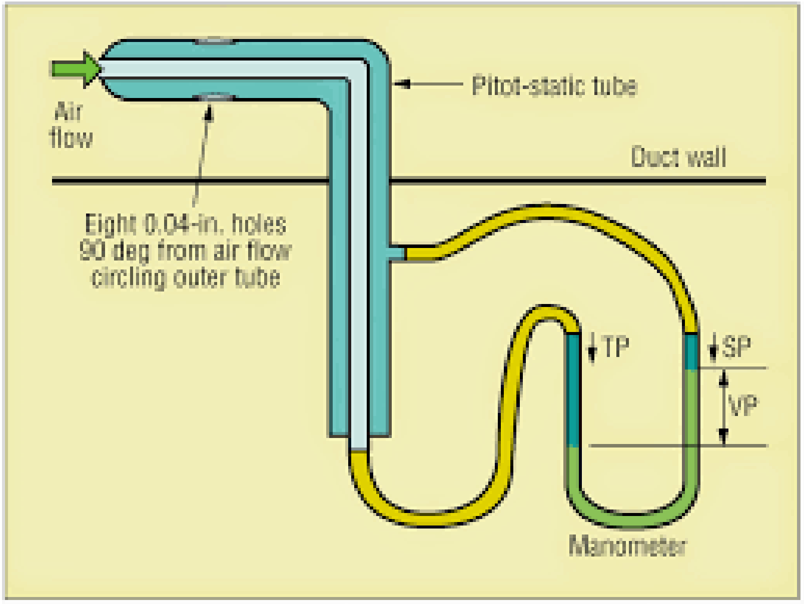


Figure MV-54: Pitot tube –Manometer Connection for Velocity Pressure Measurement.

Example MV-31: The following pressure observations have been made in a fan drift during a ventilation survey with a Pitot tube whose inlet is located in the center of the drift.

Total Pressure = 6 inches of water

Static Pressure = 5.25 inches of water

[a] What is the velocity of the air in the center of the drift?

[b] What is the average velocity of the air in the drift?

[c] If the fan drift is circular in cross-section and is 8 ft. in diameter, what is the quantity of air flowing in the fan drift?

Solution:

[a] Velocity of the air at the center of the drift

Velocity Pressure = Total Pressure – Static Pressure = 6 – 5.25 = 0.75 inches water

$$Velocity = 4000\ x\ \sqrt{0.75} = 4000\ x\ 0.866 = 3464.10 = 3470\,\frac{ft.}{min}.$$

[b] Average Velocity of the air in the drift

Remember from earlier discussion that the average velocity of the air in a cross-section is usually 90% of the velocity at the center of the cross-section.

Average velocity in drift = 0.9 x center velocity = 0.9 x 3470 = 3123 ft./min.

[c] Quantity flowing in the drift

Remember that the quantity = velocity x area. The area of a circular airway equals $\frac{\pi}{4}$ times diameter squared.

$$Area = \left[\frac{3.14}{4}\right] x[8]^2 = 200.96\ sq.ft.$$

$$Quantity = v\ x\ a = 3123\ x\ 200.96 = 627{,}598\ cu.ft./min.$$

$$Quantity = 627{,}600\ cfm$$

Table 1 – Relative Humidity, %
Difference Between Readings of Wet & Dry Bulbs in °F

Dry Bulb °F	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	Dry Bulb °F
40	92	83	75	68	60	52	45	37	29	22	15	7	0																		40
41	92	84	76	69	61	54	46	39	31	24	17	10	3																		41
42	92	85	77	69	62	55	47	40	33	26	19	12	5																		42
43	92	85	77	70	63	55	48	42	35	28	21	14	8																		43
44	93	85	78	71	63	56	49	43	36	30	23	16	10																		44
45	93	86	78	71	64	57	51	44	38	31	25	18	12	6																	45
46	93	86	79	72	65	58	52	45	39	32	26	20	14	8	2																46
47	93	86	79	72	66	59	53	46	40	34	28	22	16	10	5																47
48	93	86	79	73	66	60	54	47	41	35	29	23	18	12	7	1															48
49	93	86	80	73	67	61	54	48	42	36	31	25	19	14	9	3															49
50	93	87	80	74	67	61	55	49	43	38	32	27	21	16	10	5	0														50
51	94	87	81	75	68	62	56	50	45	39	34	28	23	17	12	7	2														51
52	94	87	81	75	69	63	57	51	46	40	35	29	24	19	14	9	4														52
53	94	87	81	75	69	63	58	52	47	41	36	31	26	20	16	10	6	1													53
54	94	88	82	76	70	64	59	53	48	42	37	32	27	22	17	12	8	3													54
55	94	88	82	76	70	65	59	54	49	43	38	33	28	23	19	14	9	5	0												55
56	94	88	82	76	71	65	60	55	50	44	39	34	30	25	20	16	11	7	2												56
57	94	88	82	77	71	66	61	55	50	45	40	35	31	26	22	17	13	8	4	0											57
58	94	88	83	77	72	66	61	56	51	46	41	37	32	27	23	18	14	10	6	1											58
59	94	89	83	78	72	67	62	57	52	47	42	38	33	29	24	20	16	11	7	3											59
60	94	89	83	78	73	68	63	58	53	48	43	39	34	30	26	21	17	13	9	5	1										60
61	94	89	84	78	73	68	63	58	54	49	44	40	35	31	27	22	18	14	10	7	3										61
62	94	89	84	79	74	69	64	59	54	50	45	41	36	32	28	24	20	16	12	8	4	1									62
63	95	89	84	79	74	69	64	60	55	50	46	42	37	33	29	25	21	17	13	10	6	2									63
64	95	90	84	79	74	70	65	60	56	51	47	43	38	34	30	26	22	18	15	11	7	4	0								61
65	95	90	85	80	75	70	66	61	56	52	48	44	39	35	31	27	24	20	16	12	9	5	2								65
66	95	90	85	80	75	71	66	61	57	53	48	44	40	36	32	29	25	21	17	14	10	7	3	0							66
67	95	90	85	80	75	71	66	62	58	53	49	45	41	37	33	30	26	22	19	15	12	8	5	2							67
68	95	90	85	80	76	71	67	62	58	54	50	46	42	38	34	31	27	23	20	16	13	10	6	3							68
69	95	90	85	81	76	72	67	63	59	55	51	47	43	39	35	32	28	24	21	18	14	11	8	5	1						69
70	95	90	86	81	77	72	68	64	59	55	51	48	44	40	36	33	29	25	22	19	15	12	9	6	3						70
71	95	90	86	81	77	72	68	64	60	56	52	48	45	41	37	33	30	27	23	20	17	13	10	7	4	1					71
72	95	91	86	82	77	73	69	65	61	57	53	49	45	42	38	34	31	28	24	21	18	15	12	9	6	3					72
73	95	91	86	82	78	73	69	65	61	57	53	50	46	42	39	35	32	29	25	22	19	16	13	10	7	4	1				73
74	95	91	86	82	78	74	69	65	61	58	54	50	47	43	39	36	33	29	26	23	20	17	14	11	8	5	3				74
75	96	91	86	82	78	74	70	66	62	58	54	51	47	44	40	37	34	30	27	24	21	18	15	12	9	7	4	1			75
76	96	91	87	82	78	74	70	66	62	59	55	51	48	44	41	38	34	31	28	25	22	19	16	13	11	8	5	3			76
77	96	91	87	83	79	74	71	67	63	59	56	52	48	45	42	39	35	32	29	26	23	20	17	14	12	9	6	4			77
78	96	91	87	83	79	75	71	67	63	60	56	53	49	46	43	39	36	33	30	27	24	21	18	16	13	10	8	5			78
79	96	91	87	83	79	75	71	68	64	60	57	53	50	46	43	40	37	34	31	28	25	22	19	17	14	11	9	6			79
80	96	91	87	83	79	75	72	68	64	61	57	54	50	47	44	41	38	35	32	29	26	23	20	18	15	12	10	7	5	3	80
82	96	92	88	84	80	76	72	69	65	61	58	55	51	48	45	42	39	36	33	30	28	25	22	20	17	14	12	10	7	5	82
84	96	92	88	84	80	76	73	69	66	62	59	56	52	49	46	43	40	37	35	32	29	26	24	21	19	16	14	12	9	7	84
86	96	92	88	84	81	77	73	70	66	63	60	57	53	50	47	44	42	39	36	33	31	28	26	23	21	18	16	14	11	9	86
88	96	92	88	85	81	77	74	70	67	64	61	57	54	51	48	46	43	40	37	35	32	30	27	25	22	20	18	15	13	11	88
90	96	92	89	85	81	78	74	71	68	65	61	58	55	52	49	47	44	41	39	36	34	31	29	26	24	22	19	17	15	13	90
92	96	92	89	85	82	78	75	72	68	65	62	59	56	53	50	48	45	42	40	37	35	32	30	28	25	23	21	19	17	15	92
94	96	93	89	85	82	79	75	72	69	66	63	60	57	54	51	49	46	43	41	38	36	33	31	29	27	24	22	20	18	16	94
96	96	93	89	86	82	79	76	73	69	66	63	61	58	55	52	50	47	44	42	39	37	35	32	30	28	26	24	22	20	18	96
98	96	93	89	86	83	79	76	73	70	67	64	61	58	56	53	50	48	45	43	40	38	36	34	32	29	27	25	23	21	19	98
100	96	93	89	86	83	80	77	73	70	68	65	62	59	56	54	51	49	46	44	41	39	37	35	33	30	28	26	24	22	21	100
102	96	93	90	86	83	80	77	74	71	68	65	62	60	57	55	52	49	47	45	42	40	38	36	34	32	30	28	26	24	22	102
104	97	93	90	87	83	80	77	74	71	69	66	63	60	58	55	53	50	48	46	43	41	39	37	35	33	31	29	27	25	23	104
106	97	93	90	87	84	81	78	75	72	69	66	64	61	58	56	53	51	49	46	44	42	40	38	36	34	32	30	28	26	24	106
108	97	93	90	87	84	81	78	75	72	70	67	64	62	59	57	54	52	49	47	45	43	41	39	37	35	33	31	29	27	25	108
110	97	93	90	87	84	81	78	75	73	70	67	65	62	60	57	55	52	50	48	46	44	42	40	38	36	34	32	30	28	26	110
112	97	94	90	87	84	81	79	76	73	70	68	65	63	60	58	55	53	51	49	47	44	42	40	38	36	35	33	31	29	27	112
114	97	94	91	88	85	82	79	76	74	71	68	66	63	61	58	56	54	52	49	47	45	43	41	39	37	35	34	32	30	28	114
116	97	94	91	88	85	82	79	76	74	71	69	66	64	61	59	57	54	52	50	48	46	44	42	40	38	36	34	33	31	29	116
118	97	94	91	88	85	82	79	77	74	72	69	67	64	62	59	57	55	53	51	49	47	45	43	41	39	37	35	34	32	30	118
120	97	94	91	88	85	82	80	77	74	72	69	67	65	62	60	58	55	53	51	49	47	45	43	41	40	38	36	34	33	31	120

C. TEMPERATURE MEASUREMENT

There are many reasons for the measuring the temperature of mine air. Temperature affects the density of air, humidity and cooling power of air. The discussion here is only related to the determination of humidity of the mine air.

We all know that when the temperature is high, it causes a lot of discomfort. When the temperature is high and the air is also very humid, it causes even more discomfort. The reason for this is that when work is performed by a person, the body generates heat and this heat has to be dissipated to the surrounding atmosphere to keep the body temperature normal. The mechanism by which the body regulates the body temperature is "sweating." When the sweat evaporates from the body, cooling of the body is ensured. If there is no appreciable air movement or when the air is very hot or when the air contains too much moisture, the performance of this regulatory mechanism is seriously affected. It can lead to sluggishness as well as more serious problems as heat stroke. A common measure of the amount of moisture in the air is called the relative humidity. Relative humidity is the amount of water vapor in the air compared to the amount of water vapor there could possibly be at that temperature, expressed as a percent.

The most common temperature measurements in mines are the dry-bulb and wet-bulb temperatures. The dry-bulb temperature of the mine air [T_d] is the temperature that is measured by a conventional mercury thermometer when placed in the mine air. The wet-bulb temperature of the mine air [T_w] is measured using a thermometer that has its bulb wrapped in cloth that is kept wet with water. The evaporation of the water from the wet cloth cools the bulb and the temperature drops. When equilibrium conditions are reached [heat coming from the mine air = the heat lost from the bulb], the correct wet-bulb temperature is reached.

The difference between the two temperatures [$T_d - T_w$] is a measure of the cooling power of the air. The smaller the difference between the two temperatures, the higher is the relative humidity of the mine air. When the two temperatures are the same, the relative humidity of the air is 100%, and the air has no cooling power.

Dry and wet-bulb temperatures are measured simultaneously by a number of instruments called hygrometers or psychrometers. Stationary hygrometers are useful for applications such as in fan-installations. For routine measurements in mines, sling psychrometers are common.

Stationary Hygrometer

As shown in Figure MV-55, two identical thermometers are mounted side-by-side on a rigid frame. A water reservoir, filled with distilled or tap water, keeps the wet-bulb sleeve wet at all times. The dry and wet bulb temperatures can be read on the Celsius or Fahrenheit scales.

Table MV-4: Relative Humidity Chart
http://www.iowadot.gov/erl/current/im/content/382.htm

Figure MV-55: Taylor Precision 5329 Indoor and Outdoor Thermometer with Hygrometer.

1. SLING PSYCHROMETER

A sling psychrometer is a portable instrument for quick and easy relative humidity measurements in mines. Shown in Figure MV-56 is a Bacharach Pocket-Size sling psychrometer. The compact design of the instrument allows the thermometers to be stored into the tube. During use, the thermometers are pulled from the body, and the wick of the wet-bulb thermometer is thoroughly wetted. The tube is used as handle to whirl the thermometers for about 90 seconds. The wet-bulb thermometer is read first and then the dry bulb thermometer. Setting the wet-bulb temperature against the dry bulb temperature on the slide rule type calculator on the tube, relative humidity can be read off.

Figure MV-56: Bacharach Pocket-Size Sling Psychrometer

2. DIGITAL HYGROMETER

Modern digital hygrometers use changes in the electrical capacitance or resistance to measure the humidity differences. They are generally easy to use and display dry and wet bulb temperatures and relative humidity. They also incorporate capabilities for data storage and download to computers. The readings on the face of a digital psychrometer are shown in Figure MV-57 are: relative humidity – 13%, dry-bulb or ambient temperature – 78.5 degrees F, and wet-bulb temperature – 56.2 degrees F. While in normal use, the sensor element on the top will be used; if the conditions inside a tube or duct are to be monitored, then the probe can be used.

Figure MV-57: Extech Digital psychrometer Kit Model RH 305

3. RELATIVE HUMIDITY DETERMINATION FROM DRY AND WET-BULB TEMPERATURE MEASUREMENTS

Table MV-4 provides a simple way of calculating the relative humidity given the *dry* and *wet* bulb temperature measurements. The first column of the table is the dry bulb temperatures and the entries in the first row are the differences between the dry-bulb and wet-bulb temperatures [T_d - T_w]. The entries in the table itself are the values of the relative humidities. Assume that the dry bulb temperature was 80 degrees F and the wet-bulb temperature was 65 degrees F. The difference between the two temperatures is 15 degrees. As [T_d - T_w] is 15, we will enter the column marked 15 and go down that column till we intersect the row marked 80 degrees. The reading at the intersection of 15 and 80 is 44, meaning the relative humidity is 44%.

Example MV-32: The dry-bulb and wet-bulb temperatures in the intake airway in a coal mine are, respectively, 75 and 50 degrees F. What is the relative humidity?

Solution:

1. The difference between dry bulb and wet bulb temperatures is [75-50] = 25 degrees F.
2. Enter the Table at the column marked 25 and go down that column till you intersect row marked 75.
3. Reading at the intersection of 25 and 75 is 9
4. The relative humidity of the intake air is 9%.
5.

Example MV-33: The dry-bulb and wet-bulb temperatures in the return airway in a coal mine are, respectively, 90 degrees and 88 degrees. What is the relative humidity?

Solution:

Following the same steps as before, you must be able to determine that the relative humidity in the return airway is 92%.

D. GAS MEASUREMENTS

There are several gases that are encountered in an underground coal mine. Oxygen, Nitrogen and Carbon Dioxide are normally present in both normal air and mine air. Mine air, in addition, may contain Methane, Carbon Monoxide, Hydrogen Sulfide, Oxides of Nitrogen, Sulfur Dioxide, etc. Methane is an explosive gas whereas hydrogen sulfide and carbon monoxide are toxic gases. Carbon Dioxide and Nitrogen, while neither combustible nor toxic, does not support life.

According to the health and safety laws and regulations, the concentrations of these gases must be kept below their permissible exposure limits [PEL] for miners to work. In the case of oxygen, the minimum amount of oxygen in the mine air in the workings is specified as 19.5%. Some PELs are actually Time Weighted Averages [TWA] such that they are exposures averaged over 8-hours that are deemed safe for an 8-hour working period. In addition, there are short term exposure limits [STEL] which are concentrations that are deemed safe for a 15-minute exposure. For some gases, up to four 15-minute exposures may occur in an 8-hour working period. Concentrations above which some immediate adverse health effect will be produced are referred to as Immediately Dangerous to Life and Health [IDLH] and must never be exceeded in the working place [See Table MV-5].

Mine Gas Chart

Name of Gas	Symbol	Common Name	Spec. Gravity	Source or Cause Where Found	Effect On Life	How Detected	Explosive	Combustible	Characteristics	Remarks
Carbon Dioxide	CO_2	Black Damp	1.52	Respiration of men And animals; fire	May displace Oxygen	Multi-Gas Detector	NO	NO	colorless, odorless, tasteless in low conc.	TLV 0.05% for 8 hours
Carbon Monoxide	CO	White Damp	0.967	From incomplete Combustion, blasting, & diesel exhaust	Poisonous	CO Detector	YES	YES	Colorless, odorless, tasteless, poisonous	TLV 0.005%; 50 ppm – 8 hours
Nitrogen	N_2	Nitrogen	0.967	Normal constituent of air	Will not Support life	Chemical analysis	NO	NO	Colorless, odorless, tasteless	78% of air
Hydrogen	H_2	Hydrogen	0.069	Battery charging stations; incomplete combustion in mine fires	no harmful effects with O_2 present	Methanometer or special lab testing	YES	YES	Colorless, odorless, tasteless	Explosive range: 4.1 – 74%
Oxides of Nitrogen	N_2O N_2O_2 NO_X etc.	Oxides of Nitrogen	1.59	Blasting fumes, diesel exhaust	Toxic, causes edema of the lungs	Multi-Gas Detector	NO	NO	Red-brown in high conc., tasteless, poisonous	TLV 0.0005%, 5 ppm anytime
Methane	CH_4	Fire Damp	0.555	Sewage, decaying plants or animals; coal seams	No harmful effects with O_2 present	Methanometer Or Flame Safety Lamp	YES	YES	Colorless, odorless, tasteless	Explosive range: 5 – 15%
Hydrogen Sulfide	H_2S	Stink Damp	1.19	Decomposition of some sulfur ores; stagnant water ponds	Sense of smell deadened; paralyze resp. system	Rotten egg odor or multi gas detector	YES	YES	Poisonous, colorless, smell of rotten eggs, irritating	Expl. range 4.3-45%, TLV 0.001%, 10 ppm
Oxygen	O_2	Oxygen	1.105	Atmosphere	Essential to Life	O2 detector or Flame Safety Lamp	NO	NO	Colorless, odorless, tasteless	Supports combustion

Table MV-5: Mine Gas Chart
http://miningquiz.com/2013_Coal_Contests/2013_Gillette_WY/MineGasChart.pdf

Depending on gases present in a mine, the mine official will be required to make the necessary measurements of the concentrations of these gases in mine workings and adjust ventilation to bring these gases to below their maximum allowed concentrations. There are also mandated methane checks that must be performed to ensure the safe operation of electrical-powered equipment or diesel-powered equipment.

A large number of gas measurement equipment is available for the various gases, each based on some physical, chemical or other principle to measure the concentration of a specific gas. There are also detectors that can be used to detect and measure the concentrations of several gases by installing several sensors specific to the specific gases to be detected. For example, the Solaris Multi-gas Detector [available from MSA International and shown in Figure MV-58] which is permissible for use in underground coal mines can be equipped to detect [1] combustible gases like methane, [2] oxygen-deficient and oxygen-rich atmospheres, and [3] toxic gases such as carbon monoxide, hydrogen sulfide, nitrogen dioxide, etc. The Detector has several alarm set points such as Low, High, Time-Weighted Average [TWA] and Short Term Exposure Limit [STEL], and alarm signals [alarm sounds, lights flash] to indicate that a set point has been reached. The detector is supplied with a rechargeable Lithium Ion battery or three AA alkaline batteries with a nominal run time of about 12 hours.

Figure mv-58: Solaris Multigas Detector

Example MV-34: A particular mine gas has a STEL [short term exposure limit of 50 parts per million {ppm}]. In a continuous 15-minute period, a miner was exposed to 65 ppm for 10 minutes and 35 ppm for the next 5 minutes. Did the miner's exposure exceed the STEL?

Solution: First we have to calculate the miner's weighted exposure in 15 minutes.
Miner's total exposure in 15 minutes = (65 x 10) + (35 x 5) = 650 + 175 = 825 ppm
Miner's weighted exposure = [825/15] = 55 ppm
Since 55 ppm > 50 ppm, the miner has been exposed to higher than the STEL.

Example MV-35: The Coal mine dust respirable dust standard is an 8-hour TWA. The present standard is
1.5 mg/m^3. Consider that a coal miner had the following exposures during a particular shift.

Activity	Duration, hours	Dust Level, mg/m^3
Travel to Face	0.5	0.1
Work in Face	3.0	1.0
Work Outby/Supplies	2.0	0.5
Work in Return	1.0	1.75
Work in Face	1.0	1.0
Travel to Outside	0.5	0.1

Calculate the 8-hour exposure of the miner.

Solution: As in Example 1, we will multiply the dust level with the exposure duration time and add to get the total exposure in mg/m^3 for the 8 hours.

1. O.5x0.1 + 3.0x1.0 + 2.0x0.5 + 1.0x1.75 + 1.0x1.0 + 0.5x0.1 = 6.85 [mg/m^3][hours]
2. Miner's total exposure in 8 hours = 6.85 mg
3. Miners weighted exposure = [6.85/8] = 0.86 mg/m^3

1. DILUTION VENTILATION

It is often necessary to calculate the quantity of air required to dilute explosive or toxic gases to below their maximum allowable concentrations in the mine. The following discussions are illustrative of problems in this area.

Q_I = Amount of intake air entering a working place (no methane gas in the intake air), $\frac{cu.ft.}{min.}$

Q_G = Amount of methane entering a working place from the face, roof and ribs, $\frac{cu.ft.}{min.}$

%G = Concentration of methane gas in the working place, %

The exact formula relating the above three is:

$$\%G = \left(\frac{Q_G}{QG + Q_I}\right) 100$$

An approximate formula relating the above three, particularly when Q_G is very small compared to Q_I is:

$$\%G = \left(\frac{Q_G}{Q_I}\right) 100$$

Example MV-36: Calculate the amount of intake air required to keep the concentration of methane gas to or below 0.8% when the amount of methane gas entering the working place is 135 [cu. ft./min.]. First use the approximate formula and then the exact formula.

Solution: First using the approximate formula, we get

$$0.8 = \left(\frac{135}{Q_I}\right) x\ 100$$

$$Q_I = \frac{135\ x\ 100}{0.8} = 16{,}875$$

$$Q_I = 16{,}900\,\frac{cu.ft.}{min.}$$

Using the exact formula, $0.8 = \left[\frac{135}{135 + Q_I}\right] x\ 100$

$$QI = 16{,}875 - 135 = 16{,}740\ cu.ft.per\ min.$$

As can be seen, the approximate formula gives a higher quantity of intake air and is preferable to use in most calculations. The approximate formula will also give a slightly higher concentration of gas in the working places.

Example MV-37: Given that quantity of intake air entering a working place is 20,000 $\frac{cu.ft.}{min.}$ and the amount of methane gas entering this working place is 275 $\frac{cu.ft.}{min.}$. Calculate the concentration of methane in the working place.

Solution:

$$\%G = \frac{Q_G}{Q_I}\ x\ 100 = \frac{275}{20000}\ x\ 100 = 1.375\%$$

It is left to the reader to verify that by using the exact formula, a lower concentration of gas will be obtained in the workings.

Example MV-39: In example MV-38, calculate the additional amount of intake air needed to bring the methane concentration in the working place to 0.85%.

Solution: First calculate the amount of air needed to bring the concentration down to 0.85%.

$$0.85 = \left(\frac{275}{Q_I}\right)\ x\ 100$$

$$Q_I = \left(\frac{275}{0.85}\right)\ x\ 100 = 32353$$

$$Q_I = 32400\ \frac{cu.ft.}{min.}$$

We already have 20000 cfm. Therefore, the additional amount needed is the difference between what we need and what we need.

$$Additional\ amount\ of\ air = 32400 - 20000 = 12{,}400\ \frac{cu.ft.}{min.}$$

Suppose the quantity of air flowing in an airway is Q_I with concentration of gas, % G_I. If we want the concentration of gas in the airway to be $\%G_2$, the quantity of air to be added (Q_{ADD}) can be calculated as follows:

$$Q_{ADD} = \frac{(Q_I x\ \%\ G_I)}{\%G_2} - Q_I$$

Example MV-39: Given that the quantity of air in a return entry is 60,000 $\frac{cu.ft.}{min.}$ and the methane concentration is 2.5%, calculate the additional quantity of air to be supplied to this return to bring the concentration down to 1.5%.

Solution:

$$Q_{ADD} = \frac{(60{,}000\ x\ 2.5)}{1.5} - 60{,}000 = 40{,}000\ \frac{cu.ft.}{min.}$$

Example MV-40: Due to increase in methane flow, the methane concentration in a working place where 20,000 $\frac{cu.ft.}{min.}$ of intake air is flowing is now predicted to be 1.375%. Calculate the additional amount of intake air needed to bring the concentration down to 0.85%.

$$Q_{ADD} = \frac{20{,}000\ x\ 1.375}{0.85} - 20{,}000$$

$$= 12{,}353\ \frac{cu.ft.}{min.} = 12{,}400\ \frac{cu.ft.}{min.}$$

If the percent of methane gas in the fan drifts is $\%G_F$ and the quantity of air flowing in the fan drift is $Q_F \frac{cu.ft.}{min.}$, then the quantity of gas (Q_{G-24hr}) coming out of the fan in a 24-hour period

$$Q_{G-24hr} = \left(\frac{\%G_F}{100}\right)(Q_F)(60)(24)\ cu.ft.$$

where the multipliers 60 and 24 on the right hand side stand respectively for 60 minutes in an hour and 24 hours in a day.

Example MV-41: The methanometer [an instrument that measures methane gas] in the fan drift shows an average concentration of 0.55% and the quantity measurement in the fan exhaust, an average of 675,000$\frac{cu.ft.}{min.}$, calculate the quantity of gas exhausted by the fan.

$$Q_{G-24hr} = \left(\frac{0.55}{100}\right)(675{,}000)(60)(24) = 5{,}346{,}000\ cu.ft. = 5{,}346{,}000\ cu.ft.$$

E. FAN OPERATION AND MONITORING

The Pennsylvania Mining Law [Section 237(d)] states that "all main fans shall be provided with pressure-recording gauges or water gauges." Further, Section 237 (e) states that a record of the charts shall be kept for one year. An example of a commercially available fan chart is shown in Figure MV-59. Note that in the chart, around the outer circumference, the following are marked: the days of the week, and the hours of the days of the days, from midnight to noon and noon to midnight. The radial arcs from the center of the chart to the periphery have the numbers 0 to 10, representing the water gage of the fan. There are other manufacturers of fan charts incorporating essentially the same information. The charts would continuously record the pressure of the fan from the time the chart is installed to the time the chart is replaced with another chart.

Variations in fan pressure recordings must be investigated as to their causes and explained with other observed phenomena. Variations in fan pressure may occur due to several reasons. Some of these include changes in atmospheric conditions, changes in the mine ventilation system, major roof falls in airways, major blow out of stoppings between intakes and returns, major inundations of gas, and methane and dust explosions.

The quantity of air handled by the fan is not available from the chart. This data must be collected separately and noted on the mine ventilation record.

1. EXAMPLES OF FAN CHART RECORDINGS

Shown in Figure MV-60 and Figure MV-61 are two fan charts from the report of investigations into an underground mine explosion in Dutch Creek No. 1 mine in Colorado in 1981. Both the charts are Bristol Fan Recorder Charts. Note that the radial arcs has numbers 0 to 10 going from the middle of the arc to the circumference whereas the numbers 0 to -10 going from the middle of the arc to the center of the arc. In an exhausting system, the pressure in the fan drift will be negative as compared to atmosphere and the fan chart records this negative pressure.

The mine was ventilated by two fans, both operating in the exhaust mode. The first fan, called Fan No. 1, was a Jeffrey 8HU84 Aerodyne fan, connected to a 500 hp. motor, operating in blade position no. 5 and running at 1200 revolutions per minute [rpm]. Prior to the explosion the fan was exhausting about 423,750 cfm of air into the mine at a pressure of -7.5 inches water gage. The second fan, called Fan No.2 was a Jeffrey 8H60 Aerodyne fan, connected to a 500 hp. motor, operating in blade position no. 5 and running at 1200 rpm. Prior to the explosion, the fan was exhausting 104,800 cfm at -5.3 inches water gage. The total quantity of air handled by the fans is 528,550 cfm. The methane removed by the fans was 1,612,000 cu. ft. per day. In addition, the mine had a methane drainage system, the methane from which was used for mine heating purposes. The explosion occurred on April 15, 1981 at about 4:00 p.m.

<u>OBSERVATIONS FROM CHART FOR FAN NO. 1 [FIGURE MV-60]</u>

Figure MV-60 is the chart for Fan No. 1. On the chart is the handwritten note that the chart was installed on April 13, 1981 at 7:30 a.m. It was a Tuesday. The fan chart shows that the recorder stopped recording the fan pressure on Wednesday at about 9 p.m. and did not resume recording again till Thursday, around noon time. The fan pressure before the interruption of recording was -7.5 inches water gage, and after the interruption, the fan chart started recording again on Thursday [April 16] around -6.9 inches water gage. The fan continued to operate at this water gage for the rest of the time. The difference in pressure developed by the fan after the explosion was 0.6 inches.

<u>OBSERVATIONS FROM CHART FOR FAN NO. 2 [FIGURE MV-62]</u>

Figure MV-62 is the chart for Fan No. 2. On the chart is the handwritten note that the chart was installed on April 14, 1981 at 11:00 a.m. It was a Wednesday. The chart shows that the recorder was recording a fan pressure of -5.3 inches from 11:00 a.m. to about 4:00 p.m. when the pressure changed and the recording leveled off at -4.5 inches. The explosion occurred at 4:00 p.m. on Wednesday. The difference in pressure developed by the fan was 0.8 inches.

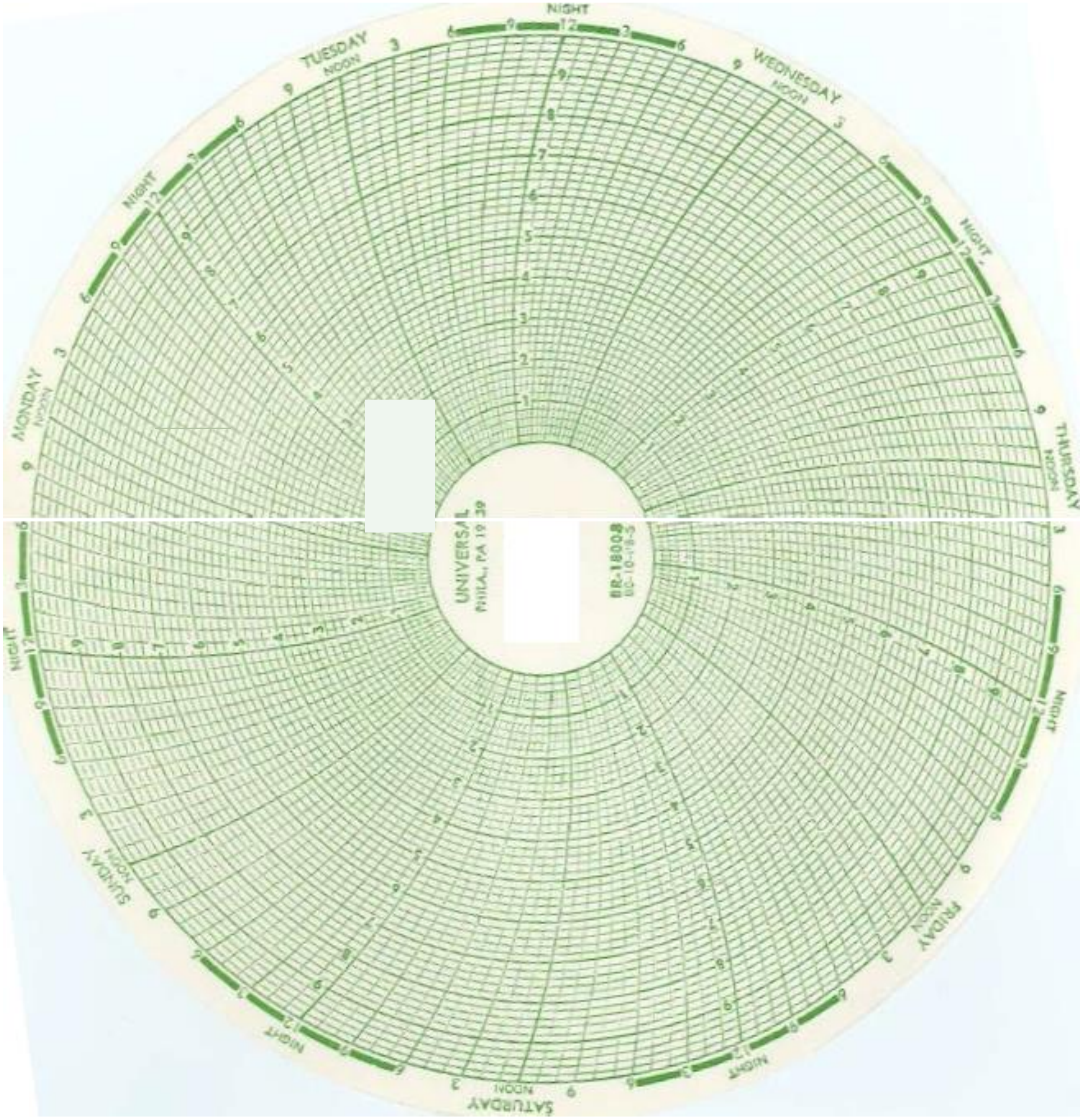


Figure MV-59: Universal Fan Pressure Chart.

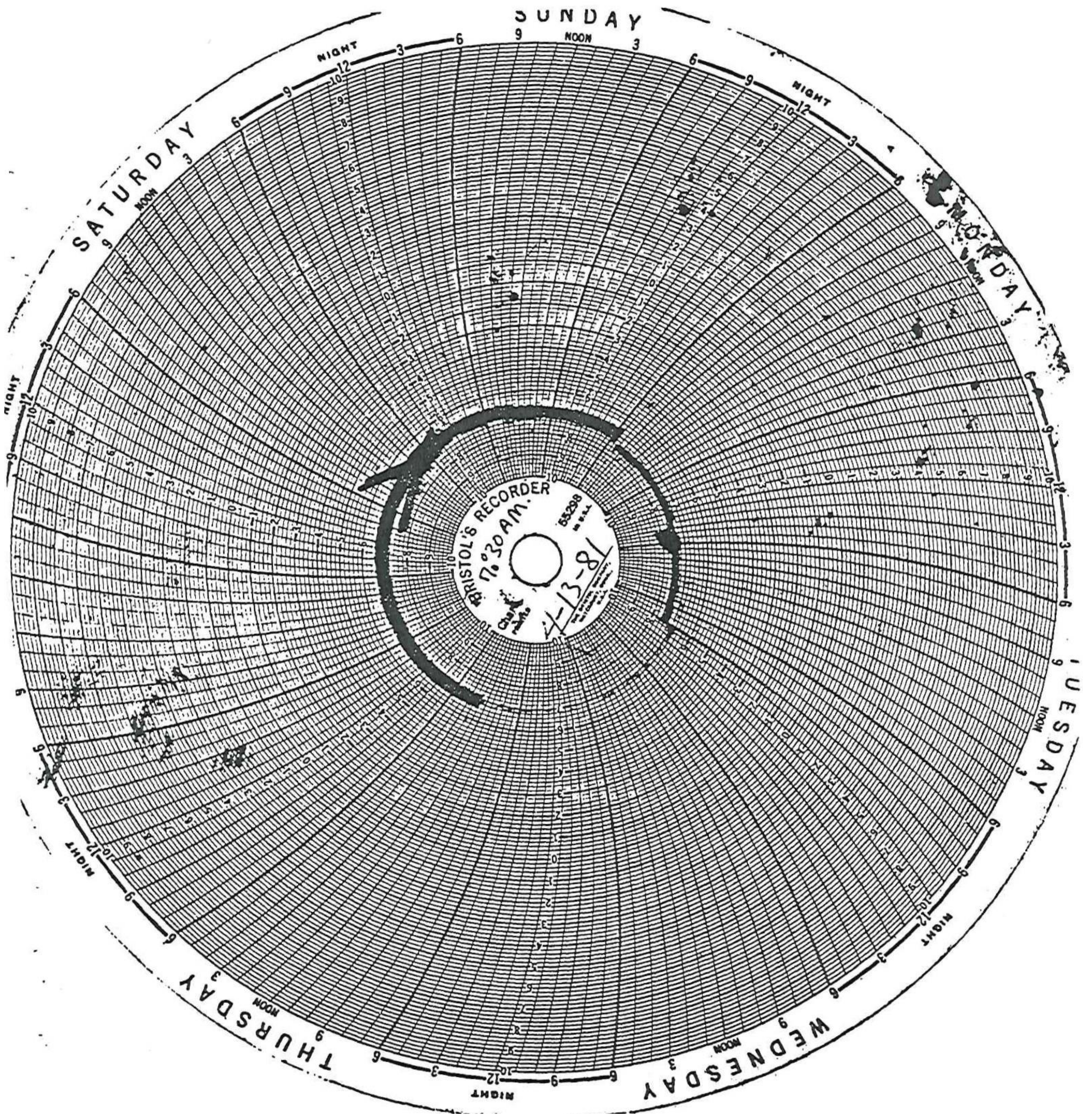


Figure MV-60: Fan Chart for Fan No. 1. This fan chart was installed on Tuesday at 7:30 a.m.

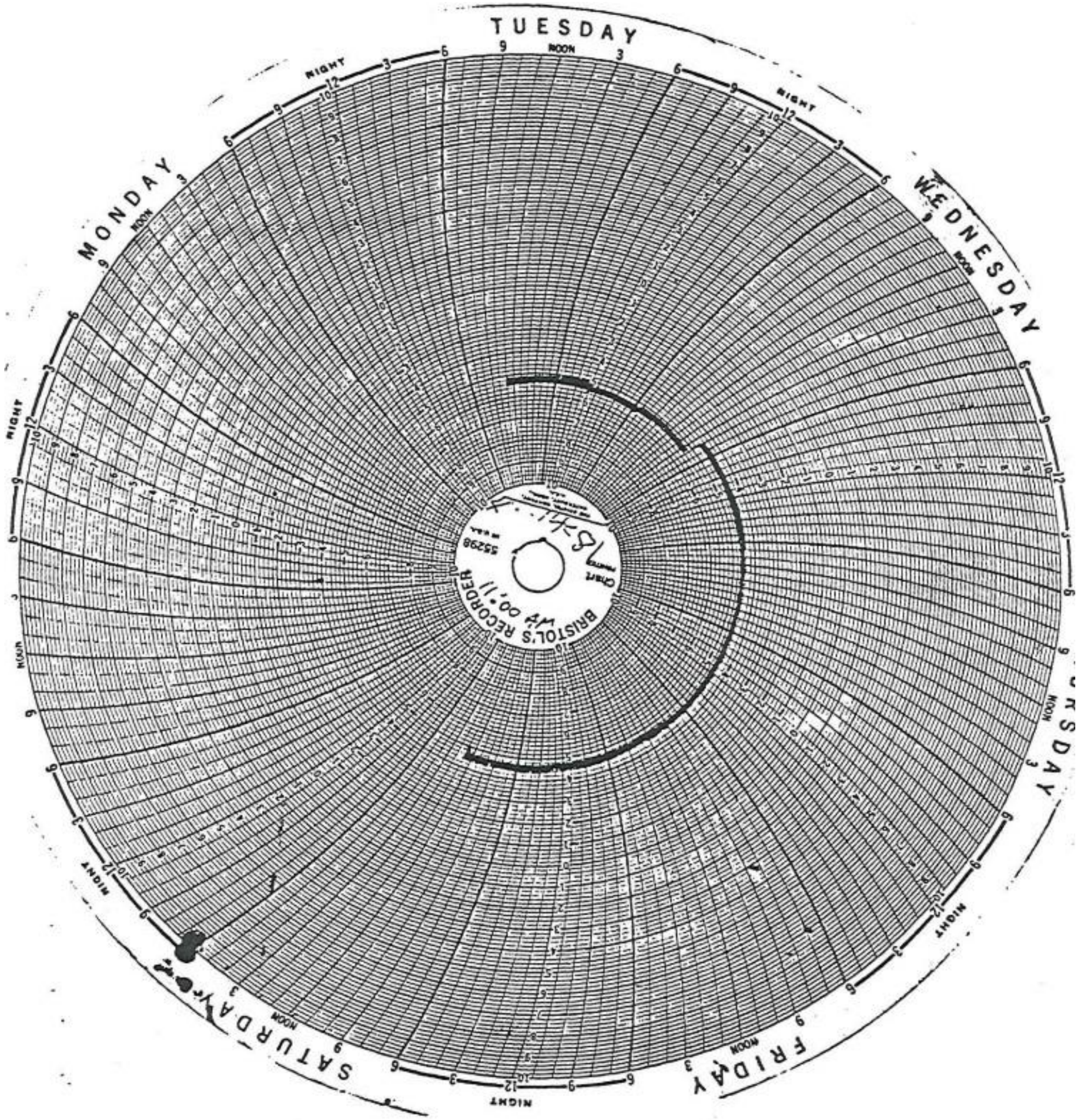


Figure MV-61: Fan Chart for Fan No. 2. This fan chart was installed on Tuesday, April 14 at 11:00 a.m.

F. MINE VENTILATION MAPS AND SYMBOLS

Ability to read mine ventilation maps and identify all the symbols in a mine ventilation map is important functions of mine officials. In fact, mine officials must be aware of the several aspects of mapping and surveying requirements in mining as mine officials have several statutory responsibilities with regard to the maintaining these up-to-date during operation and at the time of closure. Maintaining entries and the pillars on centers and dimensions in underground is as important as maintaining these maps with regard to the progress of the workings.

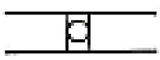
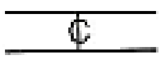
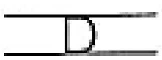
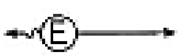
1. BOREHOLES

A borehole is a hole connecting the surface with the underground workings of a mine. The hole may be as small as 2 inches in diameter for a diamond drill hole or as large as 30 feet for an airshaft [The labor commission of the State of Utah].

2. MINE VENTILATION MAP SYMBOLS

The symbols that are listed here are typical, but not standard, symbols for use in mine ventilation maps. Several states and agencies provide a list of symbols for use and some are easy to identify because of their commonality in use. Table MV-6 is from the book "Mine Ventilation and Air Conditioning" published by John Wiley and Sons [Hartman, H.L., et al., 1997]. Color-coded arrows, such as blue for intake air, red for return air, green for escapeways, and yellow for belt air, are used to help identifying airflows. The type of arrows may also differ to denote intake and return airstreams. Therefore, it is necessary to familiarize oneself with the list of symbols that is associated with a particular map.

Table MV-7
Mine Ventilation Symbols

Symbol	Description
	Airflow (intake)
	Airflow (return)
	Airlock; a double-door system to allow equipment to pass through without disrupting the ventilation circuit
	Auxiliary fan and vent pipe or tubing (flow direction may be indicated by an arrow)
	Brattice (also called a *line brattice*); a curtain of plastic or plastic-covered fabric hung from the roof to direct air to or from a working face
	Box check; a stopping with a hole in it to allow a conveyor or other equipment to pass through while limiting the airflow quantity
	Check curtain; a barrier of plastic or plastic-covered fabric hung across an opening from the roof to block the flow of air
	Door
	Escapeway with direction of escape in the direction of airflow
	Escapeway with the direction of escape in the direction opposite to the airflow direction

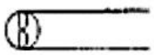
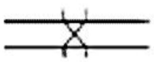
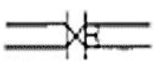
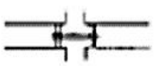
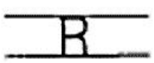
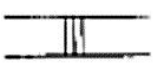
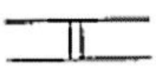
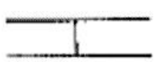
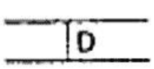
	Fan (flow direction may be indicated by an arrow)
	Fire door (normally open)
	Main fan (the dotted lines show the location of the weak wall)
	Overcast or air crossing; an area where roof material is taken to allow one airflow to pass over another without mixing (the parallel lines indicate the airway that goes straight through the overcast); may also be constructed as an undercast or sidecast crossing
	Overcast with a built-in regulator
	Pipe overcast; a method of using pipes to pass a small quantity of return air through an intake airflow without mixing the two airflows; generally used for taking belt air directly to the return in a coal mine
	Regulator
	Seal
	Self-contained self-rescuer cache location
	Shaft with a downcast flow of air (alternately, this symbol may represent an undercast)
	Shaft with an upcast flow of air (note that this symbol could also represent a gas well or a borehole location on some mine maps)
	Stopping (permanent); an impermeable stopping made of masonry, steel, or other flame-resistant material to block the flow of air through an opening
	Stopping (temporary); a quickly erected and movable stopping normally made of brattice material to temporarily block the flow of air through an opening
	Stopping with small door to allow the passage of personnel

TABLE OF CONTENTS

MINE GASES SECTION 1

MINE GASES

A. INTRODUCTION

Air is the transparent medium surrounding the earth in which plants, animals, and human beings live and breathe. Air is a mixture of several gases which, though ordinarily invisible, can be weighed, compressed to a liquid, or frozen to a solid.

Pure dry air at sea level contains by volume the following gases: oxygen (O_2), 20.94 percent; nitrogen (N_2), 78.09 percent; argon (AR),

0.94 percent and carbon dioxide (CO_2), 0.03 percent. Traces of other gases such as hydrogen, helium, etc., are also present.

Mine air may be contaminated by the presence of other gases such as carbon monoxide, sulfur dioxide, hydrogen sulfide, methane, oxides of nitrogen and excess carbon dioxide. Mixtures of gases found in mine environments are called damps.

<u>Black Damp</u> Carbon dioxide and nitrogen in an oxygen deficient atmosphere which will cause suffocation.

<u>White Damp</u> Carbon monoxide (CO). Very poisonous.

<u>Stink Damp</u> Hydrogen Sulfide (H_2S). It is colorless, has a sweetish taste, and a very distinctive odor of rotten eggs. As little as .01% in air being capable of detection by this means. The gas is even more poisonous than Carbon Monoxide.

<u>Fire Damp</u> Methane (CH_4).

<u>After Damp</u> Gaseous products and smoke produced by a fire or explosion which includes the following gases: Carbon monoxide, carbon dioxide, water vapor, nitrogen, oxygen, hydrocarbons and hydrogen.

The presence of these gases may be due to any of the following:

- After effects of blasting or other explosions.
- After effects of mine fires.
- Liberation from ore or country rock, as with methane.
- Decay of timbers in poorly ventilated areas.
- Absorption of oxygen by water or oxidation of timber or ore.
- Use of diesel and gasoline motors in enclosed areas.
- Gas carried with thermal water or carbon dioxide.

- Gas carried chemically by various chemicals and reagents.

Except in cases of fire, positive ventilating currents of sufficient quantity will prevent any dangerous accumulation of these gases. Gases may affect people either by their combustible, explosive, or toxic qualities, or, if inert, by the displacement of oxygen.

1. DIFFUSION

A gas diffuses into air and the rate of diffusion is proportional to the square root of its specific gravity (known as Graham's Law)

I.E., diffusion $\sqrt{\frac{1}{\text{sp. gr. of gas}}}$

In other words, the smaller the specific gravity, the faster the rate of diffusion. Thus a gas lighter than air will diffuse faster than the one heavier than air. In absence of diffusion, in stagnant air gases may stratify in layers. Methane layers are found near the roof whenever there is insufficient amount of air or insufficient turbulence in the flowing air.

2. THRESHOLD LIMIT VALUES (TLV)

The degree of effect of both gaseous and particulate contaminants depends largely upon the airborne concentration and the amount of exposure.

A list of threshold limit values (TLV's) is published yearly by the American Conference of Governmental Industrial Hygienists (ACGIH). These TLV's serve as guides for exposure concentrations which, it is believed, a healthy individual normally can tolerate for 8 hours a day, 5 days a week, without harmful effects.

Airborne particulate concentrations are generally listed as milligrams per cubic meter of air (mg/m^3) while gaseous concentrations are listed as parts per million or percent by volume.

3. PARTS PER MILLION (ppm)

All chemicals used in industry which have a toxic effect on workers have maximum allowable concentrations to which the employee can be exposed for an 8-hour day. These concentrations are expressed as parts per million (ppm).

For many of us, 1 part per million is about as hard to visualize as the national debt. The following helps indicate what 1 part per million really represents under various conditions:

1 ounce of sand in 31 tons

1 inch in 16 miles

The relationship between PPM and percent (%) is shown as follows:

PERCENT	PPM
1.0	10,000

0.1	1,000
0.01	100
0.001	10
0.0001	1

B. NITROGEN (N_2)

Nitrogen is a colorless, odorless, inert gas. It is not combustible and will not support combustion. Nitrogen is the main component of pure air (78.09) percent. It is slightly lighter than air.

Nitrogen has no physiological effect upon workers. It is only dangerous if present in concentrations high enough to dilute the oxygen content of air below safe limits. Dilution can result from the oxidation of various substances or from fire which consumes the oxygen in mine atmosphere. Oxygen can, therefore, be reduced to a low level and residual nitrogen can mix with products of combustion including carbon dioxide, carbon monoxide, sulfur dioxide, etc.

Nitrogen may be detected using the flame safety lamp, however, the flame safety lamp is of slight use in detecting concentrations of nitrogen because nitrogen has no effect upon the flame when accompanied by sufficient oxygen at normal atmospheres.

Nitrogen accumulations may be added to others, such as carbon dioxide in blackdamp. This can produce an oxygen deficient atmosphere which is indicated by the decrease in size of the flame or its extinction. The specific gravity of nitrogen is .967 and the weight of nitrogen per cubic foot at 60°F and 30.00 inches of mercury pressure is 0.0740. Nitrogen is approximately 4/5 of the atmosphere.

Threshold limit value (TLV) of nitrogen is 81 percent.

C. OXYGEN (O_2)

The most important gas in our world, oxygen, is nonflammable, yet nothing can burn without it. Oxygen is the element in air that supports normal combustion. In its pure state, however, when combined with fuel gases (acetylene, MAPP, hydrogen, propane) and combustible substances, oxygen causes them to burn fiercely at great speed.

Oxygen is colorless, odorless, and tasteless. It is the most important constituent of air (20.94 percent).

Oxygen is necessary to support life. By breathing, oxygen is absorbed by the blood and carried to the cells of the body. Workers breathe most easily and work best in air which contains approximately 21 percent oxygen. When the oxygen content is about 17 percent, miners at work will breathe a little faster and more deeply. The effect is about the same as going from sea level to an altitude of 5,000 feet. Workers breathing air containing as little as 15 percent oxygen usually become dizzy, notice a buzzing in their ears, have a rapid heartbeat, and often suffer headaches. Very few workers are free from these symptoms when the oxygen in air falls to 10 percent. The flame of a safety lamp is extinguished when the oxygen level falls to about 16 percent (16.25%).

Since oxygen is more soluble in water than nitrogen, air in a confined area when exposed to water will probably have a lowered oxygen content. For example: the oxygen content of the air from a hydraulic compressed air plant is lowered to about 17.7 percent oxygen--a consequent rise in nitrogen content occurs.

An oxygen percentage higher than 20 to 21 percent apparently has no negative effect on persons. This is found to be the case in using self-contained oxygen breathing apparatus. There is no noticeable effect after successive periods of wear. Oxygen in high percentages, as used with the oxygen breathing apparatus, helps people to work with less fatigue. However, it is dangerous to breathe pure oxygen at a pressure much higher than 15 pounds per square inch for a very long time. Irritating effects of oxygen are only found in humans after they have been exposed for 48 hours or more in an atmosphere containing 80 percent oxygen. Also, people who are medically diagnosed with Chronic Obstructive Pulmonary Disease (COPD) i.e. , pneumoconiosis (black lung), silicosis, emphysema, chronic bronchitis, etc., should not be administered, or exposed to high concentrations of oxygen as such high concentrations can result in oxygen toxicity, or affect the COPD person 's ability to breathe.

At approximately 7 percent oxygen, the face becomes leaden in color, the mind becomes confused and the senses dulled. When there is no oxygen in the atmosphere, loss of consciousness occurs in a few seconds without any warning symptoms. Oxygen previously in the lungs is rapidly removed and used up--loss of consciousness is quickly followed by convulsions and respiratory failure.

Some causes of oxygen deficiency include:

- Absorption by water or certain types of rock, ore, or fill
- Breathing of workers in confined space
- Displacement by carbon dioxide, carbon monoxide, or other gases
- Heating and combustion
-

Oxygen Deficiency

Oxygen Present (percent)	Effect
21	Breathing easiest.
17	Breathing faster and deeper.
15	Dizziness, buzzing noise,

	rapid pulse, headache, blurred vision.
9	May faint or become unconscious.
6	Movement convulsive, breathing stops, shortly after heart stops.

Mine air should not contain less than 19.5 percent Oxygen.

D. CARBON DIOXIDE (CO_2)

Carbon dioxide, an inert gas, is a product of the decomposition and/or combustion of organic compounds in the presence of oxygen. Carbon dioxide is also produced by respiration of people and animals. It is a colorless, odorless gas which, when breathed in large quantities, may cause a distinctly acid taste. It will not burn or support combustion. Carbon dioxide, being heavier than air, is often found in low places and abandoned mine workings and is a normal constituent of mine air. Carbon dioxide in mine air is increased by breathing, burning of flame lamps, fires, explosions, blasting or by escaping with thermal water. Carbon dioxide is also liberated from some coal beds. The specific gravity of carbon dioxide is 1.529.

Clinical investigations indicate that carbon dioxide influences the respiratory rate. This rate increases rapidly with increasing amounts of carbon dioxide.

The following table shows the effect upon humans of increased amounts of CO_2 in the air breathed:

Physiological Effects of Carbon Dioxide

Concentration of Carbon Dioxide (percent)	Increase in Respiration
0.05	Slight.
0.5	Maximum allowable for an 8-hour day.
2.0	50 percent.
3.0	100 percent.
5.0	300 percent and laborious.
10.0	Cannot be endured for more than a few minutes.
18.0	Rapid death.

Carbon dioxide in air has these effects when the oxygen content remains approximately normal and the individual is at rest. Moving around or working increases symptoms and danger is greater than when the individual is resting. Concentrations of over 5 percent carbon dioxide in air are usually accompanied by an appreciable lowering of the oxygen content.

Carbon dioxide in mine air should not be more than 0.50 percent.

E. METHANE (CH_4)

Methane is encountered in practically all coal mines. Flow of the gas is variable and is present in the pores of coal. Methane is formed by decomposition of organic matter in the presence of water and the absence of air or oxygen.

In a coal mine, methane may be emitted from the cleats or cracks of the coal, from "blowers" or "feeders", or from overlying or underlying strata. It is often released in large amounts from the coal when irregularities, such as clay veins, "horsebacks," or faults occur.

Once liberated from the strata, methane tends to accumulate near the mine roof or in high places where it mixes progressively with air currents and eventually may be found uniformly distributed across a cross section of airflow. Once mixed with fresh air, it will no longer separate into layers or form pockets of still gas. The specific gravity of methane is 0.555. The ignition temperature of methane is 1100-1380°F.

Methane is a colorless, odorless and tasteless gas. An odor caused by the presence of other gases such as hydrogen sulfide often accompanies it. Methane will burn with a pale blue nonluminous flame. Still air that contains 5 to 15 percent methane and 12 percent or more of oxygen will explode--this is its chief danger. The inflammable and explosive range of methane is variable. Occurrences of methane, if suspected or known, should be diluted with the help of adequate ventilation.

Methane is considerably lighter than air and when found at mines it is usually in high places near the roof. Accumulations of the gas may be encountered in poorly ventilated mine workings. Methane is most often detected by a methane detector.

Methane has no direct effect upon workers but it may displace the oxygen content of air to such an extent as to cause oxygen deficiency. An open flame or a spark may cause an explosion. Federal law requires electrical circuits to be isolated in any work area when the methane content in the general body of air in that area reaches 1.0 percent. Federal law also requires that all miners be withdrawn from any work area when the methane content of the general body of air in that area reaches 1.5 percent. No blasting or shot-firing is to be done when methane content exceeds 1.0 percent. Explosiveness of coal dust increases in the presence of methane and coal dust in turn decreases the lower explosive limit of methane.

<u>Law Regarding Measurement of Methane</u>

When requested by the mine inspector in the district, the mine foreman or the superintendent shall once each week direct and see that the methane content of the ventilating current or currents is determined by analyses, or by an instrument capable of accuracy to one-tenth of one percent. The samples or the determinations shall be taken on the return end of the air circuit or circuits just beyond the last working place, unless otherwise directed by the mine inspector in the district, and a correct report of these determinations shall be promptly furnished to the mine inspector in the district. Said determinations, or samples, shall be taken on days when the men are at work and recorded in a book provided for that purpose.

F. CARBON MONOXIDE (CO)

Carbon monoxide gas constitutes one of the greatest hazards to life in mining. Carbon monoxide is one of the products of combustion in normal blasting operations and in the use of diesel motors and is dangerous unless adequate ventilation is provided.

Carbon monoxide will burn and air that contains 12.5 to 74 percent of carbon monoxide will explode if ignited. It is only slightly soluble in water and is not removed from the air to any extent by water sprays. It is slightly lighter than air having a specific gravity of 0.967. The ignition temperature of carbon monoxide is 1100°F.

Carbon monoxide in excess of 0.01 percent, if breathed indefinitely, may eventually produce symptoms of poisoning; 0.02 percent will produce slight symptoms after several hours' exposure. When 400 parts per million (0.04 percent) is present and the exposure is from 2-to-3 hours, headache and discomfort usually occur. With moderate exercise, when 0.12 percent is present, slight palpitation of the heart will occur in 30 minutes, tendency to stagger in 1.5 hours, and confusion of mind, headache and nausea in 2 hours. In concentrations of 1.20 to 0.25 percent, unconsciousness usually occurs in about 30 minutes. The effect of high concentrations may be so sudden that one has little or no warning before collapsing. The carbon monoxide content of the air in which workers are employed for a period of 8 hours should not exceed 0.005 percent or 50 parts per million.

1. HOW CARBON MONOXIDE ACTS

Oxygen absorbed from air in the lungs is normally taken up by the blood in the form of a loose chemical combination with the red coloring matter (hemoglobin) of the corpuscles. Oxygen is carried in this form to the tissues where it is used. Hemoglobin forms a much more stable compound with carbon monoxide than with oxygen. Hemoglobin cannot take up oxygen when saturated with carbon monoxide.

Hemoglobin's affinity to carbon monoxide is about 300 times its affinity for oxygen. Therefore, even when a small percentage of carbon monoxide is present in inhaled air, hemoglobin will absorb the carbon monoxide in preference to the oxygen. Carbon monoxide, when absorbed by hemoglobin, reduces the capacity of hemoglobin for carrying oxygen to the tissues to a proportionate extent. This interference with the oxygen supply to the tissues produces the symptoms of carbon monoxide poisoning.

The symptoms of carbon monoxide poisoning more or less parallel the extent of blood saturation. The first definite symptoms, during rest, make their appearance when 20 to 30 percent of the hemoglobin is combined with carbon monoxide. Unconsciousness takes place at about 50 percent saturation and death occurs at about 80 percent.

According to experiments conducted by the United States Bureau of Mines, the symptoms produced by various percentages to carbon monoxide in the blood .are as follows:

Percentage of Blood Saturation	Symptoms
0-10	None
10-20	Tightness across forehead, possible headache
20-30	Headache, throbbing in temples
30-40	Severe headache, weakness, dizziness, dimness of vision, nausea, vomiting, and collapse
40-50	Same as 30-40, with more possibility of fainting and collapse, increased pulse and respiration
50-60	Fainting, increased pulse and respiration, coma with intermittent convulsions
60-70	Coma with intermittent convulsions, depressed heart action and respiration, possible death.
70-80	Weak pulse and slowed respiration, respiratory failure, and death

Symptoms decrease in number as the rate of saturation increases. If exposed to high concentrations the victim may experience but few symptoms. The rate at which a person is overcome and the sequence in which symptoms appear depends upon the concentration of gas, extent to which the person is exercising, the victim 's health, individual susceptibility, temperature, humidity and air movement. Exercise, high temperature, and humidity, with little or no air movement, tend to increase respiration and heart rate and consequently result in more rapid absorption of carbon monoxide.

2. TREATMENT FOR CARBON MONOXIDE POISONING

Carbon monoxide poisoning may occur suddenly or gradually depending upon concentration and period of exposure. The most important concerns in treatment are to rest and avoid further exposure to the gas.

The most important thing in treatment of acute carbon monoxide poisoning is to remove gas from the victim's blood as rapidly as possible.

Prompt action will decrease the possibility of serious after- effects or even loss of life from cardiac arrest and respiratory failure. The natural elimination of carbon monoxide will start as soon as the patient begins to breathe air free of carbon monoxide. This normal elimination is slow and often has serious effects. It requires possibly 8 to 15 hours to reduce the carbon monoxide hemoglobin to 10 percent of the total hemoglobin. Inhalation of pure oxygen will remove the carbon monoxide from the blood four or five times faster.

The use of oxygen alone in an oxygen therapy unit is a common practice because oxygen is usually readily available owing to its general use in industry.

Inhalation treatments are preferably given with oxygen therapy units, but oxygen may be administered by improvised apparatus or directly sprayed over the patient's face from a cylinder. Caution should be used to control oxygen flow when using gas directly from the cylinder. The cylinder should be opened and flow regulated before the gas is directed toward the patient. No improvised mask or device should be used in which pressure can build up and injure the victim.

In cases of severe carbon monoxide poisoning, the patient should be transported on a stretcher case for medical aid.

The steps for effective treatment of carbon monoxide poisoning are as follows:

1. Move the victim to fresh air as soon as possible.
2. If respiration has stopped, is weak and intermittent, or present only in occasional gasps, give artificial respiration until normal breathing is resumed, or until it is definitely established that the patient is dead.
3. Administer pure oxygen as soon as possible and continue as long as necessary (at least 20 minutes in mild cases and as long as 1 or 2 hour in severe cases). Immediate inhalation of oxygen for 20 to 30 minutes will significantly lessen the severe effects of carbon monoxide poisoning and lessen the chance of serious after-effects.
4. Aid circulation by rubbing the patient’s limbs (toward the heart). Keep the victim warm with blankets, hot water bottles, etc.
5. Keep the victim lying down to avoid strain on the heart and allow plenty of time to rest and recuperate.

Physiological Effects of Carbon Monoxide

PPM	Concentration of Carbon Monoxide (Percent)	Allowable Length of Exposure
50	0.005	Allowable for exposure of several hours.
400–500	0.04–0.05	Can be inhaled for 1 hour without appreciable effect.
600–700	0.06–0.07	Just noticeable effects after 1-hour exposure.
1000–1200	0.10–0.12	Unpleasant, but probably not dangerous after 1-hour exposure.
1500–2000	0.15–0.20	Dangerous for exposure of 1 hour.
4000 or more	0.4 or more	Death in less than 1 hour.

G. OXIDES OF NITROGEN (NO, NO_2, N_2O_4, N_2O_2, N_2O, N_2O_3, N_2O_5)

Oxides of nitrogen are formed in mines by burning of explosives and, to a lesser extent, by their detonation. Oxides of nitrogen can usually be detected by the burnt powder odor familiar to blasters and by the reddish color of nitrogen peroxide (NO_2) fumes, which are formed when nitric oxide (NO) produced by an explosion comes in contact with air. Gases collected from the burning of 40 percent gelatin dynamite contain 11.9 percent oxides of nitrogen. When explosives having properly proportioned components are completely detonated, they usually produce small percentages of oxides of nitrogen which are considered harmless. Explosives from which the wrapper has been removed may produce harmful percentages of oxides of nitrogen--even when detonated. Diesel engines also produce oxides of nitrogen.

Physiological Effects of Oxides of Nitrogen Concentration of Oxides of Nitrogen

PPM	Percent	Effect
5	0.0005	Maximum allowance for 8-hour day.
60	0.006	Minimum causing immediate throat irritation.
100	0.01	Minimum causing coughing.
100-150	0.01-0.015	Dangerous for even short exposure.
200-700	0.02-0.07	Rapidly fatal for short exposure.

A person exposed to dangerous concentrations of nitrogen dioxide may feel no discomfort for several hours after the end of exposure. As much as eight hours after exposure the victim's lungs may become filled with fluid resulting in asphyxia.

When air samples are analyzed for oxides of nitrogen, the results usually are reported in terms of nitrogen dioxide (N02), since this designation gives proper evaluation of the toxic properties of the atmosphere.

The threshold limit value (TLV) of oxides of nitrogen is 5 ppm.

H. SULFUR DIOXIDE (SO_2)

Sulfur dioxide gas is produced by burning sulfide ores (pyrites) or by blasting in sulfide ores or explosions of sulfide ore dust. Some diesel fuels produce sulfur dioxide. Sulfur dioxide is not combustible.

This gas has a strong sulfur smell which is suffocating and very irritating to breathe. Sulfur dioxide cannot be tolerated for any length of time in dangerous concentration. The specific gravity of sulfur dioxide is 2.264.

The gas affects the lungs in much the same manner as oxides of nitrogen and hydrogen sulfide.Irritation of the respiratory tract and lungs will cause edema.

Effects of Sulfur Dioxide Concentration of Sulfur Dioxide

PPM	Percent	Effect
5	0.0005	Maximum allowable for an 8-hour day.
20	0.002	Coughing and irritation to eyes, nose, and throat.
150	0.015	May be endured for several minutes.
400	0.04	Impossible to breathe.

Sulfur dioxide is highly soluble in water--in fact it is one of the most soluble gases found in mines. It is a very heavy gas and has a specific gravity of 2.264. It can, therefore, be expected to accumulate in low places. Sulfur dioxide is colorless with a distinctly acid taste.

The threshold limit value (TLV) of sulfur dioxide is 5 ppm.

I. HYDROGEN SULFIDE (H_2S)

Hydrogen sulfide is one of the most poisonous known gases. Only traces of it are ordinarily found in mine operations. In low concentration its distinctive rotten egg odor is noticeable. In high concentrations the sense of smell is quickly paralyzed by the action of the gas on the respiratory center. It is colorless and poisonous even in small concentrations. The gas has a specific gravity (SG) of 1.191 and may collect at low points in mines. The ignition temperature of Hydrogen sulfide is 700°F.

Hydrogen sulfide inhaled in a sufficiently high concentration produces immediate asphyxiation; in low concentrations it produces inflammation of the eyes and respiratory tract and sometimes leads to bronchitis, pneumonia, and edema of the lungs. The immediate effect of hydrogen sulfide is extreme irritation to the eyes.

Sub acute poisoning may be produced by long exposure to concentrations as low as 0.005 percent. Immediate collapse usually results from exposure to concentrations of 0.06 to 0.1 percent, and death quickly follows. The 8-hour daily exposure should not exceed 0.001 percent or 10 parts per million.

Hydrogen sulfide can be detected by hydrogen sulfide detectors. It can also be detected chemically by exposing a paper dipped in acetate which turns black immediately in presence of hydrogen sulfide.

When explosions of dust occur in blasting operations in sulfide ore bodies, the resulting gases may contain varying amounts of hydrogen sulfide, along with sulfur dioxide, and possibly other sulfur gases.

Hydrogen sulfide is highly explosive with an explosive range if 4.3 to 46 percent.

Physiological Effects of Hydrogen Sulfide Concentration of Hydrogen Sulfide

PPM	Percent	Time	Effect
10	0.001	----	Maximum allowable for 8-hour day.
50-100	0.005-0.010	1 hour	Subacute poisoning- 1. Mild eye irritation 2. mild respiratory irritation
200-300	0.02-0.03	1 hour	Subacute poisoning 1. marked eye irritation 2. marked respiratory irritation
500-700	0.05-0.07	5-1 hour	Subacute to acute poisoning--unconsciousness
1000-2000	0.10-0.20	Minutes	Acute poisoning- 1. unconsciousness 2. death

J. HYDROGEN (H_2)

Hydrogen is a colorless, odorless, and tasteless gas. It is very much lighter than air with a specific gravity of 0.0695 and is highly flammable. Hydrogen is explosive over a broad range of concentrations (4.1 to 74 percent). It will explode with as little as 5 percent oxygen in the air and is most violently explosive at concentrations of 7 to 8 percent.

Hydrogen is not a toxic gas and the only danger from inhaling it is when concentrations are such that the oxygen content of the air is reduced.

Hydrogen is normally found in mine air in only very small quantities. It can, however, be produced when mine fires heat rock to incandescence and also as a result of incomplete combustion.

MG-13

The most common source of hydrogen gas under normal circumstances is in the battery charging area. The electrolytic action which takes place during battery charging releases hydrogen gas. Charging stations must, therefore, be well ventilated and smoking, electric arcs, etc., must be avoided in them.

From a trace to as much as 9 percent can be found in crevices of a coal face after blasting. It is formed here as a result of incomplete combustion of explosives and by distillation of the coal caused by the explosion.

Hydrogen gas is usually present in amounts up to 2 percent in gas from ordinary mine fires and is always present after coal dust explosions.

Keep flames and sparks away from hydrogen, as with other fuel gases. Do not crack the valve of a hydrogen cylinder to blow out dirt, etc., it could be dangerous.

Hydrogen is detected only by chemical analysis.

MINE GASES
QUESTIONS AND ANSWERS

MG Q-1	Where is Methane most likely to be found in a mine?
MG A-1	Near the roof, as it is lighter than air.
MG Q-2	What percent of carbon monoxide can cause death in less than one hour?
MG A-2	0.4 percent or more.
MG Q-3	How do oxides of nitrogen cause death?
MG A-3	Causes fluid to accumulate in lungs resulting in asphyxia.
MG Q-4	What are the sources of Hydrogen in a mine?
MG A-4	Charging batteries, mine fires, and explosions.
MG Q-5	Hydrogen is always present after what type of explosion?
MG A-5	Coal dust explosions.
MG Q-6	Write down the chemical symbols of methane and hydrogen sulfide.
MG A-6	CH_4 and H_2S.
MG Q-7	What percentage of the earth's atmosphere is oxygen?
MG A-7	20.94 percent oxygen.
MG Q-8	What percent of the earth's atmosphere is nitrogen?
MG A-8	78.09 percent nitrogen.
MG Q-9	Once two or more gases mix uniformly will they separate or come apart?
MG A-9	No.
MG Q-10	When is the oxygen (O_2) level in air considered to be dangerous?
MG A-10	When oxygen (O_2) level falls below 16%.
MG Q-11	What is the chemical symbol of carbon dioxide?
MG A-11	CO_2
MG Q-12	What element in the air is essential for life?
MG A-12	Oxygen.

MG Q-13 How does the body receive oxygen?
MG A-13 By breathing, oxygen is absorbed by the blood and carried to the cells of the body.

MG Q-14 What is meant by the term "black damp"?
MG A-14 An atmosphere deficient in oxygen.

MG Q-15 How can methane gas be detected in a coal mine?
MG A-15 Chemical analysis, flame safety lamp and methane detectors.

MG Q-16 What is the explosive range of methane?
MG A-16 Five to fifteen percent.

MG Q-17 A flame safety lamp will go out when the oxygen percentage is below what?
MG A-17 16 percent.

MG Q-18 The explosive range of carbon monoxide is what?
MG A-18 12.5-74 percent.

MG Q-19 What distinctive odor does hydrogen sulfide gas smell like?
MG A-19 Smells like rotten eggs.

MG Q-20 What are the properties of hydrogen sulfide?
MG A-20 Poisonous and colorless with an odor like rotten eggs.

MG Q-21 What is a flammable mixture of methane and air which can either burn or explode when ignited called?
MG A-21 Firedamp.

MG Q-22 A sealed area of a coal mine after a period of time will be found to have the absence of what?
MG A-22 Oxygen.

MG Q-23 What is the specific gravity of carbon dioxide?
MG A-23 1.529

MG Q-24 How are oxides of Nitrogen formed?
MG A-24 From the use of explosives in mines

MG Q-25 How are results from analysis of oxides of Nitrogen reported?
MG A-25 In terms of nitrogen dioxide.

MG Q-26 What is the ignition temperature of methane?
MG A-26 1100-1380° F

MG Q-27 What is the ignition temperature of Carbon monoxide?
MG A-27 1100° F.

MG Q-28 Is hydrogen explosive?
MG A-28 Yes.

MG Q-29 What is the specific gravity of hydrogen?
MG A-29 It is the lightest of all gases with a specific gravity of 0.0695.

MG Q-30 What is the explosive range of hydrogen?
MG A-30 4.1 to 74 percent.

MGQ-31 How is hydrogen detected?
MG A-31 By chemical analysis.

MG Q-32 What is the principal poisonous gas produced by explosions?
MG A-32 Carbon monoxide.

MG Q-33 What effect does carbon monoxide have on life?
MG A-33 It is extremely poisonous.

MG Q-34 How does carbon monoxide cause injury to life?
MG A-34 By combining with the hemoglobin of the blood and excluding oxygen.

MG Q-35 What percentage of carbon monoxide will produce slight symptoms in several hours?
MG A-35 200 PPM (.02%).

MG Q-36 What percentage of carbon monoxide will produce discomfort in two or three hours?
MG A-36 400 PPM (.04%).

MG Q-37 What percentage of carbon monoxide will produce a tendency to stagger in one and one-half (l.5) hours?
MG A-37 1200 PPM (0.12%).

MG Q-38 What percentage of carbon monoxide will produce symptoms of unconsciousness in thirty (30) minutes?
MG A-38 2000-2500 PPM (.20%-.25%).

MG Q-39 How much greater affinity does hemoglobin have for carbon monoxide than for oxygen?
MG A-39 About three hundred (300) times.

MG Q-40 Why are small quantities of carbon monoxide injurious?
MG A-40 Because it is not easily eliminated and it accumulates in the blood.

MG Q-41 What is the specific gravity of carbon monoxide?
MG A-41 0.967.

MG Q-42 What is carbon monoxide (CO)?
MGA-42 It is a colorless, odorless, tasteless, combustible, and poisonous gas.

MG Q-43 How can carbon monoxide be detected?
MG A-43 By carbon monoxide detectors, and by analysis.

MG Q-44 Workers should not be employed for a period of 8 hours where the carbon monoxide content exceeds what?
MG A-44 50 PPM (0.005%)

MG Q-45 What percentage of carbon monoxide might produce symptoms of poisoning if breathed indefinitely?
MG A-45 0.01%.

MG Q-46 What is the source of carbon monoxide?
MG A-46 It is the product of incomplete combustion (combustion with an insufficiency of oxygen).

MG Q-47 When is carbon monoxide most likely to be found in mines?
MG A-47 When there is a mine fire or after an explosion.

MG Q-48 What instruments are most often used in detecting methane?
MG A-48 The flame safety lamp and methane detectors.

MG Q-49 What is the least percentage of methane that can be detected with a flame safety lamp?
MG A-49 About one percent (1%).

MG Q-50 What gas is odorless, tasteless, non-toxic, colorless and explosive in the concentration of 5%-15%?
MG A-50 Methane.

MG Q-51 What is the source of methane in coal mines?
MG A-51 It is liberated from coal and adjoining strata.

MG Q-52 What is the specific gravity of methane?
MG A-52 0.555.

MG Q-53 Where is methane usually found in mines?
MG A-53 Along the roof, to the rises, in the vicinity of working faces, in dead ends and above falls.

MG Q-54 Is methane an explosive by itself?
MG A-54 No. Oxygen is required to support combustion.

MG Q-55 Why can there be no explosion when the percentage of methane is greater than fifteen percent (15%)?
MG A-55 Because the amount of oxygen present is insufficient for rapid combustion to occur.

MG Q-56 What is the percentage of methane required for maximum explosive violence?
MG A-56 Ten percent (10%).

MG Q-57 What is the percentage of oxygen below which no explosion of a methane air-mixture can occur?
MG A-57 Twelve percent (12%).

MG Q-58 What effect does an atmosphere with a reduced oxygen content have upon the explosibility of methane?

MG A-58 A greater percentage of methane is necessary to start an explosion in an atmosphere which contains less than the normal percentage of oxygen.

MG Q-59 What effect does the presence of methane have upon the explosibility of coal dust?

MG A-59 The coal dust is more easily ignited and the force of the explosion is greater.

MG Q-60 What effect does coal dust in the air have upon the explosibility of methane?

MG A-60 The lower explosive limit is decreased.

MG Q-61 What dangerous gas is most likely to be encountered above a pillar fall?

MG A-61 Methane.

MG Q-62 Where might concentrated accumulations of carbon dioxide ordinarily be found?

MG A-62 Near the floor, in inadequately ventilated places.

MG Q-63 What effect does carbon dioxide have upon life?

MG A-63 Respiration is increased as concentration of carbon dioxide increases.

MG Q-64 How is carbon dioxide detected?

MG A-64 Usually by chemical analysis.

MG Q-65 What is carbon dioxide (C02)?

MG A-65 Carbon dioxide is a colorless and odorless gas formed by the chemical combination of carbon and oxygen.

MG Q-66 How is carbon dioxide formed in a mine?

MG A-66 By combustion, by breathing of miners and animals, by decay of vegetable and animal matter, by the oxidation of coal and by chemical action of acid water on carbonates.

MG Q-67 What is a product of complete combustion?

MG A-67 Carbon dioxide.

MG Q-68 Is carbon dioxide combustible?
MG A-68 No.

MG Q-69 How does the body receive oxygen?
MG A-69 Through breathing, the oxygen is taken up by the hemoglobin of the blood and carried to all parts of the body.

MG Q-70 What supports the chemical reaction that produces fires and explosions?
MG A-70 Oxygen.

MG Q-71 What percent oxygen can a person most easily work in?
MG A-71 20.9%

MG Q-72 What percent oxygen will a person breathe faster and deeper while at work?
MG A-72 17 percent.

MG Q-73 What is nitrogen?
MG A-73 It is a tasteless, odorless and colorless gas which will neither support life nor combustion.

MG Q-74 Is nitrogen combustible?
MG A-74 No.

MG Q-75 What effect does nitrogen have towards propagating an explosion?
MG A-75 None.

MG Q-76 What effect does nitrogen have upon life?
MG A-76 It has no effect, except when it depletes oxygen to the extent that there is a deficiency of oxygen.

MG Q-77 Does nitrogen have an ignition temperature?
MG A-77 No, nitrogen will not explode.

MG Q-78 How is sulfur dioxide formed in a mine?
MG A-78 By burning coal containing pyrites.

MG Q-79 What is the specific gravity of sulfur dioxide?
MG A-79 2.263.

MG Q-80 What is the particular danger of sulfur dioxide?
MG A-80 It is extremely poisonous even in small amounts.

MG Q-81 How is sulfur dioxide detected?
MG A-81 By the sense of smell and its effect on the air passages.

MG Q-82 What is the first effect on a person exposed to sulfur dioxide?
MG A-82 It is extremely irritating and suffocating and is intolerable to breathe.

MG Q-83 Is sulfur dioxide combustible?
MG A-83 No, it is incombustible.

MG Q-84 What mine gas can be detected by its odor?
MG A-84 Hydrogen sulfide.

MG Q-85 What is the origin of hydrogen sulfide?
MG A-85 It is liberated by burning explosives containing sulfur such as black powder or dynamite.

MG Q-86 How can hydrogen sulfide be detected other than by sense of smell?
MG A-86 By the hydrogen sulfide detector or by paper dipped in acetate of lead, which will turn black immediately on exposure to hydrogen sulfide.

MG Q-87 What is the specific gravity of hydrogen sulfide?
MG A-87 1.191.

MG Q-88 What is the explosive range of hydrogen sulfide?
MG A-88 4.3% to 45%.

MG Q-89 Is hydrogen sulfide poisonous?
MG A-89 Yes, it is extremely poisonous even in small amounts.

MG Q-90 What is the immediate effect of hydrogen sulfide on a person?
MG A-90 It is extremely irritating to the eyes.

MG Q-91 Is sulfur dioxide soluble in water?
MG A-91 Yes.

MG Q-92 What is noxious gas?
MG A-92 Any gas in the air which is harmful to life when inhaled.

MG Q-93 Which is the heaviest, one cubic foot of methane or one cubic foot of air?
MG A-93 One cubic foot of air.

MG Q-94 What gas is found near the roof and cavities on falls?
MG A-94 Methane (CH4).

MG Q-95 What does the presence of CO in a sealed mine area indicate?
MG A-95 A fire.

MG Q-96 What percentage of blood saturation by Carbon monoxide (CO) will cause death?
MG A-96 70% - 80%.

MG Q-97 Why will methane accumulate in an inadequately ventilated place?
MG A-97 It is lighter than air and will rise and stratify if not properly diffused.

MG Q-98 Can there be a mine fire or an explosion without the presence of oxygen?
MG A-98 No.

MG Q-99 What is the principle combustible gas usually found in coal mines?
MG A-99 Methane.

MG Q-100 Define the term diffusion of gases.
MG A-100 Diffusion is a phenomenon by which gases mix by natural forces.

MG Q-101 What is the law of diffusion?
MG A-101 The rate of diffusion varies inversely as the square root of specific gravity.

MG Q-102 What is stratification?
MG A-102 When gases do not diffuse completely, layers of gas stratify horizontally.

MG Q-103 Name the gas which is generated from a storage battery.
MG A-103 Hydrogen (H2).

MG Q-104 Name the non-explosive gases found in coal mines.
MG A-104 Carbon Dioxide (CO_2) and Nitrogen (N_2).

MG Q-105 Name the explosive gases found in bituminous coal mines in Pennsylvania.
MG A-105 Methane (CH_4), Carbon monoxide (CO), and Hydrogen Sulfide

MG Q-106 Is methane (CH_4) poisonous?
MG A-106 No.

MG Q-107 Among methane, carbon monoxide, and hydrogen sulfide, which one has the lowest ignition temperature?
MG A-107 Hydrogen sulfide (700°F).

MG Q-108 What type of atmosphere is easy for the detection of a gas - a diffused one or a stratified one?
MG A-108 A stratified one.

MG Q-109 What is the effect of black damp on flame safety lamp?
MG A-109 The flame of a safety lamp is dimmed or extinguished depending on concentration of gases present.

MG Q-110 What is the color of methane?
MG A-110 Methane is colorless.

MG Q-111 What is the most simple and safe test for the presence of black damp?
MG A-111 A flame safety lamp.

MG Q-112 Stink damp refers to which gas?
MG A-112 Hydrogen sulfide.

MG Q-113 White damp refers to which gas?
MG A-113 Carbon monoxide.

MG Q-114 Convert 0.01 percent of carbon monoxide to ppm.
MG A-114 100 ppm.

MG Q-115 List the threshold limit value for oxides of nitrogen.
MG A-115 5 ppm.

MG Q-116 What is the threshold limit value of nitrogen?
MG A-116 81%.

MG Q-117 What is TLV?
MG A-117 Threshold Limit Value.

MG Q-118 What is the threshold limit value of hydrogen sulfide?
MG A-118 10 PPM.

MG Q-119 Is hydrogen flammable?
MG A-119 Yes.

MG Q-120 Mine air should not contain less than what percentage of oxygen?
MG A-120 19.5%.

TABLE OF CONTENTS

MINE GASES SECTION 2

MINE GASES SECTION 2

BASIC CHEMISTRY CHARACTERISTICS OF EXAMPLE ELEMENTS AND GASES

SYMBOLS AND ATOMIC WEIGHTS OF ELEMENTS

Element	Symbol	Atomic Weight	Element	Symbol	Atomic Weight
Aluminum	Al	27	Manganese	Mn	55
Calcium	Ca	40	Mercury	Hg	201
Carbon	C	12	Nitrogen	N	14
Chlorine	Cl	35.5	Oxygen	O	16
Copper	Cu	63.5	Phosphorus	P	31
Gold	Au	197	Potassium	K	39
Helium	He	4	Silicon	Si	28
Hydrogen	H	1	Silver	Ag	108
Iodine	I	127	Sodium	Na	23
Iron	Fe	56	Sulphur	S	32
Lead	Pb	207	Tin	Sn	119
Magnesium	Mg	24	Zinc	Zn	65

EXAMPLE RATES OF DIFFUSION OF GASES COMPARED TO AIR

Gas Diffusion	Density	Specific Gravity	Rate of Calculated	
Observed			From Sp. Gr.	
Air ********	14.4	1.000	**********	
Acetylene ********	13	0.907	1.0503	
Carbon Dioxide	22	1.529	0.8090	0.812
Carbon Monoxide	14	0.967	1.0173	1.015

Ethane ********	15	1.049	0.9763	
Ethylene ********	14	0.975	1.0132	
Hydrogen	1	0.069	3.8168	3.830
Hydrogen Sulfide	17	1.190	0.9171	0.950
Methane	8	0.555	1.3428	1.344
Nitrogen	14	0.967	1.0173	1.014
Nitrogen Dioxide ********	23	1.589	0.7916	
Oxygen	16	1.105	0.9514	0.949
Sulphur Dioxide ********	64	2.264	0.6646	

MINE GASES SECTION 2

A. BASIC CHEMISTRY

1. MOLECULAR WEIGHTS

In comparing the weights of gases we assumed that, since hydrogen is the element with the least weight, we would give the hydrogen atom the weight of 1 unit. Now hydrogen molecules contain two atoms, if our previous statements can be accepted, so the weight of a hydrogen molecule is 2. Equal volumes of gases should, at the same temperature and pressure, contain equal numbers of molecules. If we weigh equal volumes of hydrogen, oxygen, carbon dioxide, or any other gas, their comparative weights should be representative of the comparative weights of the molecules. Hydrogen, oxygen, and carbon dioxide have weights of equal volumes in the ratio of 2, 32, and 44 respectively. We can say, then, that the molecules of hydrogen, oxygen, and carbon dioxide have weights in the ratio of 2, 32, and 44, respectively.

It was found that, when 2 grams of hydrogen (a gram is a unit of weight in the metric system), 32 grams of oxygen, or 44 grams of carbon dioxide were compared in volume, each gas has a volume of 22.4 liters (a liter is a unit of volume in the metric system, equal to slightly more than one quart). From this evidence it was decided that <u>the molecular weight of any gas in grams is equal to a volume of 22.4 liters at standard conditions of temperature and pressure</u> (0°C. and 760 mm.)

If we desire the molecular weight of a liquid or a solid, we can weigh a known volume of the substance at any temperature and pressure at which it is gaseous, then reduce this volume by rule to 0°C. and 760 mm. pressure, and calculate the weight of 22.4 liters.

Molecular Weight of Certain Gasses

	Mol. Wt.		Mol. Wt.
Hydrogen	2	Carbon monoxide	28
Oxygen	32	Nitrogen	28
Chlorine	71	Hydrogen sulfide	34
Carbon dioxide	44	Air	28.8

2. GAS DENSITIES

The density of a substance is its weight per unit of volume. For gases, the weight of one cubic centimeter at 0°C. and 760 mm. pressure is the density. It is customary, though, to use the relative weight of a volume of the gas compared to an equal volume of hydrogen as the density. This may be stated as, <u>the density of gas is the number of times it is heavier than hydrogen, volume for volume</u>.

Equal volumes of hydrogen, carbon dioxide, and chlorine weigh 2, 44 and 71 units of weight respectively; therefore, carbon dioxide has a density of $\frac{44}{2}$ or 22, and chlorine has a density of $\frac{71}{2}$ or 35.5.

It is evident from these examples that <u>the density of a gas is equal to one-half of its molecular weight</u>. Hydrogen with a molecular weight of 2 has a density of 1; chlorine with a molecular weight of 71 has a density of 35.5.

3. ATOMIC WEIGHTS

There are several methods by which the atomic weights, or weights of the atoms, may be found. These weights apply only to elementary substances, such as hydrogen, chlorine, oxygen, etc.

(a) If we assume that all elementary gases contain two atoms in each molecule, then the atomic weight for each gas is one-half the molecular weight. Thus, the atomic weights of hydrogen and oxygen are 1 and 16, respectively.

As mentioned previously, if the molecular weight of hydrogen is taken as 2, then that of oxygen is not 32 but 31.76, and the atomic weight of oxygen would be 15.98. It is customary to regard the atomic weight as 16, and that of hydrogen as 1.008. There are so many more compounds of oxygen than of hydrogen, and to have a whole number for the atomic weight of oxygen makes calculation of the atomic weights of other substances much easier.

(b) If we have an element which is not easily obtained in the form of a gas, but which forms gaseous compounds with other elements, these gaseous compounds can be analyzed, and their molecular weights found by weighing 22.4 liters of the gases at standard conditions of temperature and pressure. For instance, carbon forms many compounds with oxygen and hydrogen. An analysis of some of these compounds is given as follows:

	Mol. Wt.	Percent Carbon by Wt.	Wt. Carbon in Mol. Wt.
Carbon dioxide	44	27.2	12
Carbon monoxide	28	42.8	12
Acetylene	26	92.3	24
Ethylene	28	85.7	24
Ethane 30	30	80.0	24
Methane	16	75.0	12

The percent of carbon by weight in the gas is easily found from the combining weights. That is, 75 percent carbon and 25 percent hydrogen by weight form 100 percent of hydro-carbon called methane.

It will be noticed that 12 is the least weight of carbon in the molecular weight of any carbon compound in the above table; in fact, this is true for any carbon compound. The atomic weight of carbon is 12.

(c) Two scientists, Dolong and Petit, found by experiment that when the atomic weights of elements were multiplied by the specific heats of the substance in the solid condition, the products were approximately the same in all cases. This product equals 6.4 (approximately). Conversely, this value divided by the specific heat of an element should give the atomic weight of that element.

For example, the specific heat of iron at 100°C. is 0.115 calorie per gram. Then 6.4 divided by 0.115 equals 55.6, while the accepted atomic weight is 55.84 or 56 in round numbers. For copper, with a specific heat of 0.1 calorie per gram, 6.4 divided by 0.1 gives 64, while the accepted atomic weight is 63.57.

(d) The equivalent weight of an element has been found to be either the same as the atomic weight, or the atomic weight is some value obtained by multiplying the equivalent weight by some simple, whole number. For instance, in the case of carbon, the equivalent weight is 3, and the atomic weight is 4 times 3 or 12. The equivalent weight of oxygen is 8, while the atomic weight is 2 times 8 or 16.

Following are the atomic weights of a few elements:

	Atomic Wt.		Atomic Wt.
Hydrogen	1	Sulphur	32
Oxygen	16	Calcium	40
Nitrogen	14	Iron	56
Carbon	12	Silver	108

4. SYMBOLS

The statement of a chemical change is usually given in a form which can be read at a glance. Instead of using the names of the various elements or compounds involved, a symbol is used for each element and a combination of symbols is used for compounds. For instance, the letter 0 represents one atomic weight (16 units) of oxygen. In many cases the first letter of the name is used, as H for hydrogen, N for nitrogen, C for carbon, etc. When the names of several elements begin with the same letter, some will be designated by symbols of two letters, as Cl for chlorine, Ca for calcium, Mg for magnesium and Mn for manganese. In some cases the symbol is derived from the Latin term, as Cu for copper (cuprum), Fe for iron (ferrum), and Hg for mercury (hydrargyrum). The German language supplies Na for sodium (natrium) and K for potassium (kalium). Whenever a symbol is used in a chemical equation it represents one chemical unit weight (atomic weight) of the element. It is, of course, convenient at times to use the symbol in mentioning or indicating an element without any reference to its atomic form, as in speaking of oxygen we may use the symbol 0.

For convenience the following table gives the names, symbols, and atomic weights of the various elements with which the mining student will ordinarily deal. The weights are given as the nearest round figure.

Symbols and Atomic Weights of Elements

Element	Symbol	Atomic Weight	Element	Symbol	Atomic Weight
Aluminum	Al	27	Manganese	Mn	55
Calcium	Ca	40	Mercury	Hg	201
Carbon	C	12	Nitrogen	N	14
Chlorine	Cl	35.5	Oxygen	O	16
Copper	Cu	63.5	Phosphorus	P	31
Gold	Au	197	Potassium	K	39
Helium	He	4	Silicon	Si	28
Hydrogen	H	1	Silver	Ag	108
Iodine	I	127	Sodium	Na	23
Iron	Fe	56	Sulphur	S	32
Lead	Pb	207	Tin	Sn	119
Magnesium	Mg	24	Zinc	Zn	65

5. FORMULAE

A chemical formula is a short statement which indicates the kinds of atoms in a molecule, either of an element of a compound, and the number of each kind or the part by weight. For example, NaCl is the formula for common table salt or sodium chloride; it means that the molecule contains one atom of sodium and one atom of chlorine; this, in turn, means that 23 parts by weight of each molecule is sodium and 35.5 parts by weight is chlorine, and the total weight is 23 + 35.5 or 58.5 units of weight.

To continue, carbon dioxide is represented by CO_2, or each molecule has one atom of carbon and two atoms of oxygen. The number of atoms of any particular element in a molecule is indicated by a small number written on the line and to the right of the symbol for the element; this number is called a subscript. When the symbol has a subscript, the atomic weight of that element must be multiplied by the subscript to get the proper total weight of the element in the compound. In the case of CO_2, the weight of carbon in the compound is 12 (one atomic unit weight), and the weight of oxygen is 2 times 16 (the atomic weight) or 32; the weight of the molecule, or the molecular weight, is 12 plus 32 or 44.

In finding the molecular weight of a compound from its formula, it is best to first set down the formula, substitute the atomic weights, multiply by any subscript numbers, and add all weights to get the total weight. Let us use methane gas, or CH4, as an example:

CH_4 = one C atom + 4 H atoms
Atomic weight of C = 12
Atomic weight of H = 1
Then, $(1\ x\ 12) + (4\ x\ 1) = 16 = mol\ .wt.of\ CH4$

Regardless of how complicated the formula of a compound may be, its molecular weight can always be found by this method. The formulae of certain compounds with which mining students should be familiar are given in the following table:

FORMULAE AND MOLECULAR WEIGHTS OF COMPOUNDS

Name	Form	Formula	Molecular Wt.
Water	Liquid	H_2O	18
Sulfuric Acid	Liquid	H_2SO_4	98
Potassium hydroxide	Liquid	KOH	56
Methane	Gas	CH_4	16
Ethane	Gas	C_2H_6	30
Propane	Gas	C_3H_8	44
Carbon dioxide	Gas	CO_2	44
Carbon monoxide	Gas	CO	28
Hydrogen sulphide	Gas	H_2S	34
Sulphur dioxide	Gas	SO_2	64
Acetylene	Gas	C_2H_2	26
Lime	Solid	CaO	56
Carbide	Solid	CaC_2	64
Slaked lime	Solid	$Ca(OH)_2$	74

In working with gases which are elements, use the molecular formula for the gas instead of the atomic symbol. Thus, oxygen in O_2, nitrogen is N_2 etc. These gases are actually a collection of the molecules, so using the molecular formula is proper, just as we use CO_2 for carbon dioxide gas which is a collection of CO_2 molecules.

B. SPECIFIC GRAVITY AND DIFFUSION OF GASES

1. SPECIFIC GRAVITY AND DENSITY RELATIONS

The specific gravity of a gas is its relative weight compared with air; that is, how many times it is heavier than an equal volume of air. The specific gravity of air is taken as 1, and gases heavier than air will have specific gravities between zero and 1.

Specific gravity of CO_2 = 1.529

Specific gravity of CH_4 =0.555

The density of a gas is the number of times it is heavier than an equal volume of hydrogen. The density of hydrogen is taken as 1, and since hydrogen is the lightest of all gases, the densities of all gases will be greater than 1.

The molecular weight of hydrogen is 2, so the density of hydrogen is equal to one-half its molecular weight. In fact, the molecular weight of any gas is twice its density. Since the molecular weight of a gas is the weight of 22.4 liters at 0°C. and 760 mm. pressure, the molecular weight of air, or any other mixture of gases, can be found by weighing 22.4 liters of the gas mixture. The following table gives the molecular weights and densities of mine gases:

MOLECULAR WEIGHTS AND DENSITIES OF MINE GASES

Gas	Molecular Weight	Density
Air	28.8	14.4
Water vapor	18	9
Oxygen	32	16
Nitrogen	28	14
Carbon dioxide	44	22
Carbon monoxide	28	14
Nitrogen dioxide	46	23
Methane	16	8
Ethane	30	15
Ethylene	28	14
Acetylene	26	13
Hydrogen	2	1
Hydrogen Sulphide	34	17
Sulphur dioxide	64	32

Calculating the Specific Gravity of a Single Gas

Since the specific gravity of gas is its relative weight, volume for volume, compared with air, the densities of gases compared with the density of air will give specific gravities.

The molecular weight of any gas is easily calculated from its formula. For example, the molecular weight of CH_4 is (12 + 4) or 16, and the density will be one-half of 16 or 8. For oxygen gas, with a molecular weight of 32, the density is one-half of 32 or 16.

Many students confuse these calculated density values with atomic weights. It is true that the atomic weight of oxygen is 16, the same as its density value, but carbon dioxide does not have an atomic weight because it is a compound; so one-half the molecular weight of carbon dioxide is its density value of 44/2 or 22, not an atomic weight. This is true for methane, carbon monoxide, or any gas; specific gravity calculations involve densities, not atomic weights, and gas densities are equal in value to one-half the molecular weights.

Example: What is the specific gravity of CO?

Answer: $Mol.Wt.of\ CO = 12 + 16 = 28$

$$Density\ of\ CO = \frac{28}{2} = 14$$

$$Density\ of\ air = 14.4$$

$$Specific\ gravity\ of\ CO = \frac{14}{14}.4 = 0.972$$

It will be noticed that the calculated specific gravity is 0.972 as compared with the accepted specific gravity of 0.967. This is due to a slight error in stating the molecular weight of air, which is 28.96 instead of 28.8. The density of air, then, should be one-half of 28.96 or 14.48. If we use this value in our calculations, we will find that 14/14.48 equals 0.967. However, the use of 14.4 shortens the calculation considerably, and the results are close enough for practical purposes. The value of 14.4 as given in the table of molecular weights and densities will be continued in use in this section.

Example: Find the specific gravity of ethane, C_2H_6.

Answer: $Mol.Wt.of\ C_2H_6 = 24 + 6 = 30$

$$Density\ of\ C_2H_6 = 15$$

$$Sp.Gr.Of\ C_2H_6 = \frac{15}{14.4} = 1.041\ (True, 1.049)$$

Calculating the Specific Gravity of a Mixture of Gasses

We can find the specific gravity of a mixture of gases by either weighing 22.4 liters of the mixture and comparing this weight with an equal volume of air or calculating the specific gravity of the mixture from the percentage by volume of the various gases and their individual specific gravities. The second method will be illustrated by an example.

Example: Find the specific gravity of a mixture of 2 cu. ft. of air, 1 cu. ft. of nitrogen, and 3 cu. ft. of carbon dioxide.

Answer: We must first assume that the weight of 1 volume of air is 1 unit of weight. The weight of each gas in the mixture, then, will equal the volume of that gas times its specific gravity.

$$Wt. of\ air\ =\ 2\ x\ 1\ =\ 2$$
$$Wt. of\ nitrogen\ =\ 1\ x\ 0.967\ =\ 0.967$$
$$Wt. of\ carbon\ dioxide\ =\ 3\ x\ 1.529 =\ 4.587$$
$$Total\ wt. of\ mixture =\ 7.554$$
$$Total\ number\ of\ volumes\ \ 2\ +\ 1\ +\ 2\ =\ 5$$
$$Total\ wt. of\ same\ volume\ of\ air\ =\ 5$$
$$Specific\ gravity\ of\ mixture\ = \frac{7.554}{5} =\ 1.511.$$

A second method of calculating the specific gravity of a mixture uses the densities of the gases and their percentage by volume in the mixture. The volume of a gas times its density gives its relative weight compared to an equal volume of <u>hydrogen</u>. The total weight, then, of a mixture of gases which is found in this manner will bear a relation to the weight of an equal volume of hydrogen, and this relation when taken for one volume, will be the density of the mixture. The relation of this density to the density of one volume of air will give the specific gravity of the mixture.

Example: A sample of a mine damp contains 20% co2 and 80% CH4• Will the mixture be lighter or heavier than air?

Answer: $Volume\ of\ CO_2\ =\ 1;\ density\ of\ CO_2\ =\ 22$
$$Volume\ of\ CH_4\ =\ 4;\ density\ of\ CH_4\ =\ 8$$
$$Wt. of\ CO_2\ in\ mixture\ =\ 1\ x\ 22\ =\ 22$$
$$Wt. of\ CH_4\ in\ mixture\ =\ 4\ x\ 8\ =\ 32$$
$$Total\ wt. of\ mixture\ =\ 54$$
$$Total\ volumes\ of\ mixture\ =\ 1\ +\ 4\ =\ 5$$
$$Wt. of\ mixture\ per\ unit\ volume\ =\ density\ =\ \frac{54}{5}\ =\ 10.8$$
$$Density\ of\ air\ =\ 14.4$$
$$Sp. gr. of\ mixture\ = \frac{10.8}{14.4}\ =\ 0.75.$$

The proportions or percentages by volume of the gases which form a mixture can be found by reversing the procedures indicated in solving the two foregoing examples. In any statement concerning the mixture, the component gases must be indicated, otherwise it becomes necessary to assume that certain gases are present. For instance, it is not enough to say, "the firedamp has a sp. gr. of 0.85." The firedamp may have air, methane, and carbon dioxide in it. The statement should be, "the firedamp, containing air and methane, has a sp. gr. of 0.85."

Example: The specific gravity of air is 1. Calculate the percentage by volume of the oxygen and nitrogen in the air.

Answer: If we assume one volume of air, then the weight of oxygen will be its sp. gr. (1.105) times X, which is the quantity of oxygen in 1 volume. The weight of the nitrogen will be its sp. gr. (0.967) times (1-X), which is the quantity of nitrogen in 1 volume.

$$Wt. of\ oxygen\ =\ 1.105\ X$$
$$Wt. of\ nitrogen\ =\ 0.967\ (1-X) =\ 0.967\ -\ 0.967X$$
$$Wt. of\ air\ =\ 1\ (sp. gr.) times\ 1\ (vol. of\ air)\ =\ 1$$

The combined weights of the oxygen and nitrogen must equal the weight of the air, or,

$$l.105X\ +\ (0.967\ -\ 0.967X) =\ 1$$
$$l.105X\ +\ 0.967\ -\ 0.967X\ =\ 1$$
$$0.138X1\ -\ 0.967\ =\ 0.033$$
$$X\ =\frac{0.033}{0.138}\ =\ 0.239\ or\ 23.9\%\ of\ oxygen\ in\ air.$$
$$100\ -\ 23.9\ =\ 76.1\%\ of\ nitrogen\ in\ air.$$

(Note: The specific gravity of nitrogen is given as 0.967. This is for pure nitrogen. Atmospheric nitrogen, with which argon and other inert gases are mixed, has a specific gravity of 0.962. When this value is used, the percentages of oxygen and nitrogen in air become 21 and 79, respectively.)

2. THEORY OF THE DIFFUSION OF GASES

Gases will diffuse into each other if given the opportunity. For example, natural gas, which has a distinctive odor, if allowed to escape from a tank or a gas main into a room where the air is motionless, will soon penetrate all parts of the room and make its presence known by its odor. This diffusion or "wandering" of the gas molecules is a natural result of their activity. As explained in previous sections, these small particles are in a constant state of motion which causes them to fly in a straight line until they collide with another molecule or meet with the opposition of a container wall, when they rebound without any apparent loss in velocity to continue their headlong flight. If two gases are in contact with each other, they will diffuse into each other until the gases are thoroughly mixed. This will occur independently of any motion of the air, but is hastened by such motion.

True diffusion in coal mines is confined to those areas where the ventilating current does not penetrate, as in abandoned or worked-out areas, and in places blocked off by falls.bGases which are formed in ventilated areas, or which are drawn out of the coal seam and the strata overhead or underneath the seam, are mechanically mixed by the ventilated areas.

Gases diffuse into each other at different velocities. This is due to the nature of the molecules, some gases having extremely heavy molecules, while other gas molecules are light in weight. If two gases, one with heavy molecules (say, co2 which has a specific gravity of 1.529)and the other with light molecules (such as hydrogen which has a specific gravity of 0.069), diffuse into each other, the lighter gas molecules with their greater velocities penetrate the heavier gas more quickly than the heavy gas molecules penetrate the lighter gas. At the end of a short period of time the light gas would have diffused into the heavy gas to a much greater extent than the heavy gas would have diffused into the light gas. This, of course, would not continue for a great length of time because eventually the two gases would become thoroughly mixed and would have the same proportions of each in all parts.

Rate of Diffusion of Gases

The relationship between the velocity of gas molecules and their weight was discovered by Graham who stated, that the rate of diffusion of a gas is inversely proportional to the square root of its density. We know that the density of gas is one-half its molecular weight, so the formula of a gas will provide us with the information necessary to calculate its rate of diffusion.

Example: What is the rate of diffusion of methane as compared to that of air?

Answer: $Density\ of\ CH4 = \frac{12+4}{2} = 8$

$$Density\ of\ air\ =\ 14.4$$

Indicating the rate of diffusion by the letter R, R of CH_4: R of Air:: $\sqrt{14.4}$: $\sqrt{8}$, or as 3.79 : 2.83. Then CH diffuses $\frac{3.79}{2.83}$ or 1.34 times as rapidly as air.

Another statement for the rates of gas diffusion is that the rate of diffusion of a gas with respect to air is inversely proportional to its specific gravity. This is a restatement of Graham's Law with the specific gravity values being used in place of densities. It is used where the specific gravities of gases are more readily obtained than are the densities. Using the same example, the calculation of the relative diffusion rates of methane and air by use of the specific gravities of the gases is shown as follows:

Answer: $Sp.gr.of\ air 1$

$$Sp.gr.of\ CH4\ 0.555$$

Then, R of CH : R of Air :: $\sqrt{1} : \sqrt{0.555}, or\ as\ 1 : 0.745.$

Then methane will diffuse $\frac{1}{0.745}$ or 1.34 times as rapidly as air

The rates of diffusion of gases compared to that of air are given in the following table.

We can compare the rates of diffusion of two gases by their comparable rates with air. For instance, the rate of diffusion of methane with respect to air is 1.3428 (see table), while that of carbon dioxide is 0.8090. The rate of diffusion of methane with respect to carbon dioxide is 1.3428 0.8090 or methane diffuses 1.3428/0.8090 or 1.66 times as fast as carbon dioxide.

Example: Find the rate of diffusion of hydrogen as compared to hydrogen sulphide.

Answer: R of hydrogen: R of air: $\sqrt{14.4}$: $\sqrt{1}$" or as 3.79 : 1.
That is, hydrogen diffuses 3.79 times (3.8168 times, if sp. gr. is used) as fast as air.
R of hydrogen sulfide: R of air: $\sqrt{14.4} : \sqrt{17}$, or as 3.79 : 4.123.
That is, hydrogen sulphide diffuses 3.79/4.123 or 0.9192 times (0.9171 times, if sp. gr. is used) as fast as air. Then hydrogen will diffuse 3.79/0 .9192 or 4.12 times as fast as hydrogen sulphide.

RATES OF DIFFUSION OF GASES COMPARED TO AIR

Gas	Density	Specific Gravity	Rate of Calculated From Sp. Gr.	Diffusion Observed
Air	14.4	1.000	**********	********
Acetylene	13	0.907	1.0503	********
Carbon dioxide	22	1.529	0.8090	0.812
Carbon monoxide	14	0.967	1.0173	1.015
Ethane	15	1.049	0.9763	********
Ethylene	14	0.975	1.0132	********
Hydrogen	1	0.069	3.8168	3.830
Hydrogen sulphide	17	1.190	0.9171	0.950
Methane	8	0.555	1.3428	1.344
Nitrogen	14	0.967	1.0173	1.014
Nitrogen dioxide	23	1.589	0.7916	********
Oxygen	16	1.105	0.9514	0.949
Sulphur dioxide	64	2.264	0.6646	********

Application of Diffusion in Mines

Although the ventilation current in a mine causes gases to mix mechanically with the air, there are a few applications of diffusion of gases which may be well worth studying.

Suppose we have a mixture of 10 volumes of CO_2 and 15 volumes of CH_4. The specific gravity of the mixture will be found as follows:

Wt. of CO_2 in the mixture = $1.529 \, x \, 10 \; = \; 15.29$
Wt. of CH_4 in the mixture = $0.555 \, x \, 15 \; = \; 8.325$
Total wt. of 25 volumes of the mixture = 23.615
Wt. of 25 volumes of air = 25
Sp. gr. of the mixture = $\frac{23.615}{25} = 0.944.$

Such a mixture of gases, when thoroughly diffused into each other, would be lighter than air, and would naturally be found in high places. The action of a body of this gas in a place where the ventilating air current could hardly reach it would be similar to that described for flashdamp.

Another application of diffusion of gases is where a group of men who have barricaded themselves in a development entry or some other place after a gas or coal dust explosion are overcome by the carbon monoxide gas produced during the explosion. Carbon monoxide diffuses into air 1.0173 times as fast as air diffuses into carbon monoxide. The workers may have selected a place with only one entrance, and may have built what would be considered an effective barricade across this entrance. If the barricade or stopping is not air-tight, the carbon monoxide may diffuse into the place and gradually snuff out the lives of the workers.

BASIC CHEMISTRY

QUESTIONS AND ANSWERS

CH Q-1 What is the law of conservation of matter?
CH A-1 Matter can neither be created or destroyed.

CH Q-2 What is the law of conservation of energy?
CH A-2 Energy can neither be created nor destroyed.

CH Q-3 What is the chemical symbol for carbon dioxide?
CH A-3 CO_2

CH Q-4 How much does a gallon of water weigh?
CH A-4 8.342 pounds

CH Q-5 What is meant by Atomic Weight?
CH A-5 Atomic weight is defined as the average atomic mass for a naturally occurring element.

CH Q-6 What is meant by molecular weight?
CH A-6 Molecular weight is the sum of the atomic weights of all atoms in a molecule. If one assumes that all elementary gases contain two atoms in each molecule, then the molecular weight of each gas is twice the atom weight.

CH Q-7 How many atoms are there in a molecule of hydrogen?
CH A-7 2

CH Q-8 What is the chemical symbol for carbon monoxide?
CH A-8 CO

CH Q-9 What is the molecular weight of methane?
CH A-9 16

CH Q-10 The product of incomplete combustion is what gas?
CH A-10 Carbon monoxide.

CH Q-11 What is the chemical symbol for acetylene?
CH A-11 C_2H_2

CH Q-12 Which is the heaviest, one cubic foot of methane or one cubic foot of air?
CH A-12 One cubic foot of air.

CH Q-13 What is the chemical symbol for methane?
CH A-13 CH_4

CH Q-14 What is the specific gravity of methane?
CH A-14 0.555.

CH Q-15 What is the molecular weight of oxygen?
CH A-15 32.

CH Q-16 What is the atomic weight of hydrogen?
CH A-16 1.

CH Q-17 What is the chemical symbol for water?
CH A-17 H20.

CH Q-18 What is the molecular weight of water?
CH A-18 18.

CH Q-19 What is the color of methane?
CH A-19 Methane is colorless.

CH Q-20 What gas is used as a standard in determining the molecular weight of gases?
CH A-20 Hydrogen.

CH Q-21 Does a pH of 7 indicate alkaline, neutral or acid?
CH A-21 Neutral

CH Q-22 Which is the heaviest, 100 pounds of air or 150 pounds of methane?
CH A-22 150 pounds of methane.

CH Q-23 Does a pH of 9 indicate alkaline, neutral or acid?
CH A-23 Alkaline.

CH Q-24 Does a pH of 3 indicate alkaline, neutral or acid?
CH A-24 Acid.

TABLE OF CONTENTS

MINE GASES SECTION 3

SECTION 3

Basic Physics

Common Specific Gravities

SPECIFIC GRAVITIES OF EXAMPLE LIQUIDS AND SOLIDS

SOLIDS	Specific Gravity	Solids (cont'd)	Specific Gravity
Anthracite	1.4 - 1.8	Pyrite	4.95 - 5.01
Aluminum	2.7	Sandstone	2.14 - 2.36
Asbestos	2.0 – 2.8	Silver (cast)	10.42 – 10.53
Bituminous coal	1.2- 1.5	Wood (oak)	0.6 - 0.9
Charcoal (oak	0.57	Wood (white pine)	0.35 - 0.50
Charcoal (pine)	0.28 - 0.44	Wood (yellow pine)	0.37 - 0.60
Copper (cast)	8.30 - 8.95	Acetic Acid	1.06
Copper (hard-drawn)	8.89	Alcohol (methyl)	0.81
Glass	2.4 - 2.8	Benzene	0.90
Gold (cast)	19.3	Gasoline	0.66 - 0.69
Ice	0.92	Hydrochloric acid	1.20
Iron (pure)	7.85 - 7.88	Milk	1.03
Iron (gray cast)	7.03 - 7.13	Naphtha (petroleum)	0.66
Iron (wrought)	7.8- 7.9	Nitric acid	1.21
Iron (steel)	7.6- 7.8	Oil (linseed)	0.94
Limestone	2.68 - 2.76	Petroleum	0.88
Lead (solid)	11.01	Water (distilled)	1.00
Peat	0.84	Water (sea)	1.03

EXAMPLE SPECIFIC GRAVITIES OF GASES

(32°F. and 29.92 inches Mercury)

Gas	Specific Gravity	Gas	Specific Gravity
Air	1.00	Methane	0.555
Acetylene	0.907	Nitric oxide	1.037
Ammonia	0.596	Nitrogen (pure)	0.967
Butane	2.067	Nitrogen (atmosphere)	0.972
Carbon dioxide	1.529	Nitrous oxide	1.529
Carbon monoxide	0.967	Oxygen	1.105
Ethane	1.049	Propane	1.562
Ethylene	0.975	Sulphur dioxide	2.264
Helium	0.138	Smoke (bituminous)	0.102
Hydrogen	0.069	Steam (212°F)	0.488
Hydrogen sulfide	1.189	Water vapor	0.623

SECTION 3

BASIC PHYSICS

A. MINING PHYSICS - GENERAL

1. DENSITY

In comparing two substances we may say that "iron is heavier than aluminum." This does not mean that 3 pounds of iron weigh more than 3 pounds of aluminum, since we use the same unit of weight. What it means is, that "bulk for bulk, iron is heavier than aluminum;" or, "iron is denser than aluminum."

If we have a body weighing 6 pounds and its volume is 3 cubic feet, the matter of which the body is composed is said to have a density of 2 pounds per cubic foot. Density is mass per unit volume. We might use the term "weight per unit volume," since we determine the mass or quantity of matter by the attraction which gravity exerts on the body, and this attraction is measured by a scale or a balance which gives the measure as weight.

The volume of a solid body may be calculated from its dimensions. If the body is an irregular solid, its volume may be found by displacement of water in a graduated vessel, as follows:

Place some water in a vessel with a graduated scale which will indicate the volume of the vessel's contents. Read the volume indicated on the scale when only the water is in the vessel. Then place the irregular solid in the water so that it is completely covered. The water level will rise to a new reading on the scale. The difference between the two readings will give the volume of the solid.

Water has a density of approximately 62.5 pounds per cubic foot. This value is true when the temperature of the water is 4° C. or 39.1° F., and when it is pure water. A change in temperature will change the density; a rise in temperature will lower the density. Also, the mineral content will change the density of water; it may vary from 62.3 to 62.7. For ordinary work, the density of water may be taken as 62.5 pounds per cubic foot.

The weight of air will vary considerable with changes in temperature and pressure. The density of air, then, must be taken at a certain temperature and pressure if we are to have a value which may be used in calculations. Dry air has a density on 0.08072 pounds per cubic foot at 0°C. (32° F.) and 760 millimeters of mercury pressure (29.92 inches of mercury), or it weighs this much at these set conditions. The presence of moisture in air will change its weight considerably.

2. SPECIFIC GRAVITY

The specific gravity of a substance is the number expressing how many times that substance is heavier, bulk for bulk, than the substance taken as a standard. The specific gravities of liquids or solids are compared with water; the specific gravity of a gas is compared with air. In other words, the specific gravity is a measure of how many times the substance is "denser" than the standard substance.

For instance, the specific gravity of anthracite will vary from 1.41.8 which means that one cubic foot of anthracite weighs between 1.4 x62.5 or 87.5 pounds and 1.8 x 62.5 or 112.5 pounds. The specific gravity of methane at standard conditions (0°C. and 760 mm. of mercury pressure) is 0.555; one cubic foot of methane at these conditions will weigh 0.555 x 0.08072 or 0.04479 pounds. In each example the weight of the unit substance (water for liquids and solids, air for gases) was multiplied by the specific gravity to obtain the weight per cubic foot of anthracite and methane.

In using these tables, multiply the weight of one cubic foot of the standard substance by the specific gravity of the substance whose weight is desired. With solids and liquids, no attraction to temperature or pressure is necessary for common calculation. With gases, the resulting weight will be at standard conditions of temperature and pressure; if the weight is to be found for a different temperature- and pressure, additional calculations must be made, as will be explained in a later section.

SPECIFIC GRAVITIES OF LIQUIDS AND SOLIDS

SPECIFIC GRAVITIES OF EXAMPLE LIQUIDS AND SOLIDS

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Anthracite	1.4 - 1.8	Pyrite	4.95 - 5.01
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Charcoal (pine)	0.28 - 0.44	Wood (yellow pine)	0.37 - 0.60
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EXAMPLE SPECIFIC GRAVITIES OF GASES

(32°F. and 29.92 inches Mercury)

Gas	Specific Gravity	Gas	Specific Gravity
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Carbon monoxide	0.967	Oxygen	1.105
Ethane	1.049	Propane	1.562
Ethylene	0.975	Sulphur dioxide	2.264
Helium	0.138	Smoke (bituminous)	0.102
Hydrogen	0.069	Steam (212°F)	0.488
Hydrogen sulfide	1.189	Water vapor	0.623

Example: Find the weight of 3 cubic feet of bituminous coal with a specific gravity of 1.4.

Answer: $1.4\ x\ 62.5\ =\ 87.5$ pounds per cubic foot.

$3\ x\ 87.5\ =\ 262.5$ pounds weight for 3 cubic feet.

Example: Find the weight of 10 cubic feet of carbon dioxide at standard conditions of temperature and pressure.

Answer: $10\ x\ 1.529\ x\ 0.08072\ =\ 1.234\ pounds.$

The earth is roughly a huge sphere, and on its surface is an envelope of air extending outward. The thickness of this envelope is not definitely known, although estimates of 100 to 200 miles have been made. At the earth's surface the air reaches its greatest density, and it gradually becomes less dense or thins out at the distance from the surface increases.

The atmospheric pressure exerted on one square foot of area at the earth's surface, then, is due to the weight of a column of air one foot square and possibly 200 miles in height. If we measure the atmospheric pressure at the top of the mountain, we will find it to be less than at the seashore; there is less height to the column of air supported on top of the mountain. Again, if we measure the air pressure on the bottom of a deep shaft which is below the level of the sea in elevation, we would find it to be greater than at sea level; there is more height to the column of air supported. Atmospheric pressure decreases with the distance above sea level and increases with the distance below sea level. At sea level, when the temperature is 32°F (or 0°C), the atmosphere exerts a pressure of 14.697 pounds per square inch (commonly used as 14.7), and this pressure is called one atmosphere.

3. MEASUREMENT OF ATMOSPHERE PRESSURE

Atmospheric pressure is measured by an instrument called the barometer. It would be possible, of course, to measure air pressure with a long water column, but such an instrument could be transported only with difficulty so the mercury column is used instead.

The mercurial barometer is an instrument consisting of a glass tube about 36 inches long. This tube has been filled completely with boiled mercury (boiling expels all air bubbles), the one end being sealed. The tube filled with mercury is capped (thumb placed over the open end) and then is inverted and placed in a vessel containing mercury. When the open end (now the lower end) of the tube is uncapped, the mercury falls until the column stand approximately 30 inches above the mercury is as perfect a vacuum as it is possible to form. Since there is no pressure on top of the column, the pressure holding the column up in the tube must be that exerted by the atmosphere on the mercury in the open vessel, or the column measures the atmospheric pressure in inches of mercury.

4. PRESSURE OF THE ATMOSPHERE

The gases of the atmosphere are fluids, and they act like water or any other fluid by exerting pressure on a surface immersed in them. This pressure is due to the weight of the column of air standing on that surface.

We can readily prove that air has weight by first weighing a deflated toy balloon, then filling it with air so that it expands, then weighing it again. On a delicate scale the difference in weights will be readily indicated, the full balloon weighing more than the empty one.

A second instrument for measuring air pressure is the aneroid barometer (Figure PH-2). The mercurial barometer is well suited for stationary measurements at stations, while the aneroid barometer can be used as a portable as well as a stationary instrument. It consists of a circular metallic air-tight box (either of brass or aluminum), one side of which is covered with a thin corrugated plate. The air inside the box has been partially evacuated (drawn out) so that what is left will compensate for the lessened stiffness of the corrugated plate at higher temperatures. This plate rises and falls as the atmospheric pressure changes, and this movement is exaggerated and carried, through a system of levers, to a pointer on the face of the barometer. The face is calibrated with two scales; one indicates the pressure corresponding to inches of mercury and the other indicates the elevations above sea level corresponding to the atmospheric pressures.

Aneroid barometers vary in size from 1-3/4 inches to 5 inches in diameter which provides a variety of sizes to meet conditions of travel. They are also graduated to read from 31 inches of mercury down to 14 inches of mercury, offering a wide range in elevations from 1,000 feet below to 20,000 feet above sea level.

A third instrument for recording the pressure of the atmosphere is the barograph (Figure PH-3). This is essentially a barometer mounted on the same base as a revolving drum to which a chart is attached for recording the barometric pressure. This is done by means of a pen coupled to the indicating hand of the barometer. The drum is turned by clockwork making one complete revolution in one week. The fluctuation of the atmospheric pressure can easily be read on the chart or graph. Barographs are widely used in weather bureaus, and many mining companies now employ them in place of the mercurial or aneroid barometers. The use of the chart enables an observer to determine at a glance the trend of atmospheric pressure change; this would require many observations if the other types of barometer were used.

In comparison, the aneroid barometer answers more readily to changes in pressure than the mercurial barometer, which, unfortunately, has a tendency to lag behind atmospheric changes. On the other hand, aneroid barometers have the disadvantage that they must be standardized (that is, the accuracy of their readings must be verified) periodically. Barographs have this same disadvantage of needing periodic checks, but have the advantage of a full week's pressure indication on a chart which anyone may read. In case of a "falling barometer" or a drop in atmospheric pressure, there is a tendency for the explosive of noxious gases held back in abandoned or worked-out portions of mines which are ventilated by the pressure system to expand and escape into the working sections or roadways and thus create a hazardous condition. A glance at the barometer will show when such is taking place, and the blower fan can be speeded up to create additional pressure to counteract the drop in atmospheric pressure.

5. PRESSURE, AIR, WATER AND MERCURY COLUMNS.

There is a relation between the atmospheric pressure and the indication of this pressure by mercury or water columns. The mercurial barometer with a column 29.92 inches high (at standard conditions) indicates a pressure of one atmosphere at 14.697 pounds per square inch. If we used a water barometer in which water was used instead of mercury, we would find that the atmosphere would support a column of water 33.942 feet in height. It is evident, then, that one atmosphere of pressure is equal to 29.92 inches of mercury column or 33.942 feet of water column.

If we carry a barometer from sea level to a point 876 feet (approximately) above sea level, we will find that the height of the mercury column has dropped one inch. The dense air which is found close to the earth's surface has such weight that a column approximately 876 feet high produces a pressure equal to one inch of mercury column. If this measurement were taken several miles above the surface, the density of the air would have decreased to such an extent that it might take 1,600 feet or more of air column to equal one inch of mercury column. Since most of our calculations on air columns will be at or near the earth's surface, we can use this value of 876 feet for all practical purposes.

The relation between atmospheric pressure, air, water, and mercury columns is shown in the following table:

Pressure in pounds per square inch	Feet of air column	Feet of water column	Inches of Mercury column
14.697	26,220	33.942	29.921
0.491	876	1.134	1.000
0.433	772	1.000	0.882
0.036	64	$\frac{1}{2}$or 1 inch	0.074

Example: The pressure at the top of a shaft is 29.8 inches of mercury. If the pressure at the bottom is 30.3 inches, what is the depth of the shaft, the temperature remaining the same?

Answer: Difference in pressure = 0.5 inches of mercury.
1 inch = 876 feet of air column.
0.5 x 876 = 438 feet, depth of shaft.

Example: The water gauge at a fan drift indicates a ventilating pressure of 2 inches of water column. What is the air pressure in pounds per square inch, and in pounds per square foot?

Answer: 1 inch of water column = 0.036 pounds per square inch pressure.
2 x 0.036 = 0.072 pounds per square inch pressure.
144 (sq. in. in 1 sq. ft.)x 0.072 = 10.368 pounds per sq. ft.

Example: If the barometer indicates a change of 1.5 inches of mercury column, what is the change in pressure per square foot?

Answer: 1 inch of mercury column = 0.491 pounds per square inch,
or 70.704 pounds per square foot.
1.5 x 70.704 = 106.56 pounds per square foot.

B. THE EFFECT OF PRESSURE AND TEMPERATURE OF GASES

1. THEORY OF GAS PRESSURE AND VOLUME.

A gas, such as air or oxygen, which is enclosed in a vessel, exerts a pressure on the sides of its container. This pressure is the result of the bombardment which the minute particles or molecules of the gas give the sides of the vessel. The molecules at normal temperatures are flying about in all directions with enormous velocities; as they strike a wall, they rebound with the same velocity as they had before striking; likewise, they rebound from collisions with each other without losing velocity. The attack of the molecules, similar to the striking of a target by the bullets from a machine gun, produces the pressure on the container walls.

If the gas is placed in a large container, the space through which the molecules travel is greater and the number of times each molecule strikes the container walls will be less. Also, the number of collisions with rebounds to the walls will be fewer than before. This results in a lower rate of bombardment and the pressure exerted by the gas is not as great as in the previous case. However, if the volume of the vessel be decreased, the shorter distance for travel of the molecules results in their striking the walls a greater number of times. Also the number of collisions will be increased because the molecules will be crowded into a smaller space, so the possibility of a molecule rebounding from a collision and striking the container wall will be increased. As a result the pressure, which is the result of the molecular bombardment, will be greater.

2. RELATIONS OF PRESSURE AND VOLUME OF GASES

Robert Boyle first stated the exact relation of the pressure to the volume of a certain mass of gas. He found that when the pressure was doubled, the volume was halved; if the pressure was made five times as great, the volume was reduced to one-fifth; if the pressure was reduced to one quarter, the volume became four times as great. From these observations he stated, that if the temperature remains constant, the volume of a gas varies inversely as the absolute pressure. This is called Boyle's Law.

Absolute pressure is that above a perfect vacuum. Barometers always indicate absolute pressure. Pressure gauges on steam lines, air lines, etc., always register pressure above atmospheric or barometric pressure. If we are given a gauge pressure reading, we must add the barometric pressure to it to obtain absolute pressure.

Example: A barometer reads 29.92 inches of mercury when the gauge pressure is 80 pounds per square inch. What is the absolute pressure corresponding to the gauge reading?

Answer: 29.92 inches of mercury 14.7 lbs. per square inch
$14.7 + 80 = 94.7\ lbs$. per square inch absolute pressure.

The student should note that Boyle's Law states the volume varies inversely as the absolute pressure. In solving problems which involve these conditions, it is best to set their relations down as follows:

First Volume :	Second Volume:	Second Absolute Pressure:	First Absolute Pressure
V_1	V_2	P_2	P_2

Underneath these symbols place the values of the problem, making sure that pressure values are in absolute pressure, and that volume values are identical (that is, both are in cubic feet, cubic inches, etc.).

Example: The oxygen bottle on a breathing apparatus contains 2 quarts of oxygen at an absolute pressure of 120 atmospheres. What volume would this gas occupy at atmospheric pressure, the temperature being constant?

Answer: Atmospheric pressure = 1 atmosphere.

$$V_1: V_2 :: P_2: P: 1$$
$$2\ quarts : V_2 :: 1 : 120$$
$$V_2 = 120\ x\ 2 = 240\ quarts.$$

Example: What volume will 1,000 cubic feet of gas occupy if the pressure changes from 29 inches of mercury to 30.5 inches, temperature remaining unchanged?

Answer: $v_l: v_2 : : p_2: p_l$

$$1{,}000 : v_2 :: 30.5 : 29$$
$$30.5\ v\_2 = 29\ x\ 1{,}000 = 29{,}000$$
$$V = \frac{29{,}00}{30.5} = 950.8\ cubic\ feet$$

Example: The open parts of the abandoned workings of a mine contain 2,000,000 cubic feet of volume. If the barometer reads 29.8 inches in the morning, and the atmospheric pressure drops 0.36825 pounds per square inch, what volume of gas and air would be forced out of the old workings into the roadways of the mine, provided the temperature remains unchanged?

Answer: $0.491\ lbs. per\ sq. in. = 1\ in. of\ mercury.$
$$\frac{0.36825}{0.491} = .75\ in. of\ mercury$$
$$29.8 - 0.75 = 29.05\ in. of\ mercury, second\ pressure.$$
$$V_1 : V_2 :: P_2 : P_1$$
$$2{,}000{,}000 : v_2 :: 29.05 : 29.8$$
$$V_2 = \frac{29.8\ x\ 2{,}000{,}000}{29.05} = 2{,}051{,}634\ cubic\ feet$$

Then, 2,051,634 = 2,000,000 = 51,634 cu. ft. of gas and air that would be forced out into the roadways by the fall in atmospheric pressure.

3. THEORY OF GAS TEMPERATURE AND VOLUME

If we consider the gas which was held in a container during our discussion of the theory of pressure and volume, and apply heat to this gas, we find that the pressure exerted on the walls of the container increases. Likewise, if we lower the temperature of the gas, the pressure on the container walls decreases. Evidently, the application of heat has considerable to do with the pressure exerted by a gas.

We saw that confining the gas in a smaller container increased the activity of the molecules because of the smaller space in which they could move. The application of heat produces the same result; the molecules increase their velocity and strike the container walls more frequently because of this increased speed, and the pressure exerted on the container walls is thereby increased. If heat is withdrawn from the gas, or, as we say in common terms, if the gas is cooled, the action of the molecules becomes less, their velocity decreases, and since they strike the container walls less frequently, the pressure becomes less.

Suppose, instead of a container with fixed walls, we used one with movable walls, such as a balloon. When partially filled with gas, any application of heat will increase the gas pressure so that it causes the balloon to swell; the gas thereby occupies more space. Also, the cooling of the gases in the balloon will cause a decrease in pressure and the balloon will shrink in size.

4. RELATION OF TEMPERATURE AND VOLUME OF GASES

In considering the effects of a change of temperature on the volume of a given mass of gas it is necessary that the pressure be kept constant, just as we maintained a constant temperature when considering the relations between pressure and volume.

It has been found that if gases be heated and pressure is adjusted so as to be constant always, all gases expand at the same rate, and the volume of a gas varies directly as the absolute temperature. This is called Charles' Law.

Experiment has shown that at 32°F a perfect gas expands 1/491.64 part of its volume when the temperature is increased 1°F, the pressure remaining constant. If the temperature should be reduced 491.64°F, the gas volume would, theoretically, be reduced to a point where no volume would exist. However, all gases liquefy or become liquids long before they reach this temperature and not being gases any more, they no longer follow this law. This temperature is called absolute zero; on the Fahrenheit scale it is 491.64° below the freezing point of 32°, or is 459.64° below zero; on the Centigrade scale it is 273° below the freezing point or zero. The relation of these temperatures may be shown by four thermometric scales, as in Figure PH-4.

The absolute temperature, then, is 459.64° (460° is normally used) plus any indicated temperature on the Fahrenheit scale, or is 273° plus any indicated temperature on the Centigrade scale.

Examples: $60°F = 460 + 60 = 520°F\ absolute$

$$10°C = 273 + 10 = 283°C\ absolute$$
$$-10°F = 460 - 10 = 450°F\ absolute$$
$$-5°C = 273 - 5 = 268°C\ absolute$$

In solving problems where the temperatures and volumes of gases are involved, it is good practice to set down first the relations according to Charles 'Law. That is, the volume will vary directly as the absolute temperature.

First Volume:	Second Volume:	First Absolute Temperature:	Second Absolute Temperature:
V_1	V_2	T_1	T_2

Underneath these symbols place the problem values, making certain that volumes are given in the same units and that temperatures are given as absolute values.

Example: 20 cubic feet of a gas at 76°F are cooled to a temperature of 45°F, the pressure being constant. What is the second volume?

Answer: $76°F = 460 + 76 = 536°F\ absolute$

$$45°F = 460 + 45 = 505°F\ absolute$$
$$V_l : V_2 :: T_l : T_2$$
$$20 : V_2 :: 536 : 505$$
$$V_2 = \frac{20x505}{536} = 18.04$$

5. RELATION OF VOLUME, TEMPERATURE AND PRESSURE OF GASES

The relation of the volumes occupied by gases which undergo both temperature and pressure changes is found by combining Charles' and Boyles' laws. That is, the volume change is directly proportional to the absolute temperature change and inversely proportional to the absolute pressure change. This can be set down as follows:

First Volume: Second Volume: First Absolute Temperature: Second Absolute Temperature:

Volume Absolute Temperature

x Second Absolute Temperature: x First Absolute Temperature

Absolute TemperatureAbsolute Temperature

$$V_l: V_2:: T_l P_2: T_2 P_l$$

Example: The air and other gases in an abandoned section occupy a volume of 500,000 cubic feet at 65°F and 29.9 inches of mercury pressure. If the barometer drops to 29 inches, and heating of the gob increases the temperature of the gases to 75°F, what will be the change in volume?

Answer: $65°F = 460 + 65 = 525°F\ absolute$

$$75°F = 460 + 75 = 535°F\ absolute$$

$$V_l: V_2:: T_l P_2: T_2 P_l$$

$$500{,}000: V\_2:: 525\ x\ 29: 535\ x\ 29.9$$

$$V_2 = \frac{535\ x\ \ 29.9\ x\ 500{,}000}{525x29} = 525{,}336\ cubic\ feet$$

Example: The air entering a mine at a temperature of 32°F and atmospheric pressure equals 350,000 cu. ft. per minute. What volume will this air occupy if the temperature rises to 50°F and the pressure increases to 15 pounds per square inch?

Answer: $Atmospheric\ pressure\ -\ 14.7\ lbs. per\ sq. in.$

$$31°F = 460 + 32 = 492°F\ absolute$$

$$50°F = 460 + 50 = 510°F\ absolute$$

$$V\_l: V\_2:: T\_lP\_2: T\_2P\_l$$

$$350{,}000: V_2:: 492\ x\ 15: 510\ x\ 14.7$$

$$V_2 = \frac{350{,}000\ x\ 510\ x\ 14.7}{492\ x\ 15} = 355{,}548\ cubic\ feet$$

C. WEIGHT AND SOLUBILITY OF GASES

1. WEIGHT OF AIR

One cubic foot of dry air at standard conditions of temperature and pressure (32°F or 0°C; 29.92 inches or 760 mm of mercury) weighs 0.08072 pound. This value should be noted and remembered because it is used in the calculation of the weights of air and other gases under varying conditions of temperature and pressure.

If the barometer be reduced to one inch of mercury, the volume of one cubic foot would be increased to 29.92 cubic feet, temperature remaining constant. At this pressure the density of the air would be

$\frac{0.08072}{29.92} = 0.0026977\ lbs\ per\ cubic\ foo$t

If the pressure rose to 20 inches of mercury, the density of air would be

$$20\ x\ 0.0026977\ =\ 0.05395\ lbs\ per\ cubic\ foot$$

In general, if the temperature remains constant at 32°F, the weight of one cubic foot of air at a barometric pressure of B inches of mercury will be (0.0026977 x B) pounds.

In considering the effect of temperature, let us suppose the barometer is held constant at one inch of mercury so that a cubic foot of air weighs 0.0026977 pound. When the temperature changes from 32°F to 33°F, the volume of one cubic foot will increase by 1/492 (Charles' Law). One cubic foot of air under these conditions will weigh

$\frac{0.0026977}{1+\frac{1}{492}}\ pound.$ If the temperature is raised to 34°F, the weight becomes $(0..0026977)/(1 + 2/492)$ pound. At any temperature T°F above 32°F, the weight of one cubic foot of air will be

$$\frac{0.0026977}{1\ +\frac{T}{492}} = \frac{0.0026977\ x\ 492}{492\ +\ T} = \frac{1.3273}{492\ +\ T}$$

Since Fahrenheit temperature readings begin at 0°F, and this is 32° below freezing point, the formula should be revised to allow T to have values read directly from the Fahrenheit temperature scale. The formula then becomes$\frac{1.3273}{460+T}$. This is the weight of one cubic foot of dry air at any temperature T and under a barometric pressure -of one inch of mercury. At any barometric pressure B, in inches of mercury, and at any temperature T, in degrees Fahrenheit, <u>the weight in pounds of one cubic foot of dry air is given by the formula</u>.

$$W\ =\ l..\frac{3273B}{460+T}$$

This formula is the one which normally will be used, for in normal mining practice temperature is calculated in degrees Fahrenheit and pressure is given in inches of mercury. One exception is when the pressure is given in pounds per square inch, and in this case, --

$$W\ = \frac{2.7P}{460\ +\ T}$$

The factor 2.7 is obtained by dividing the constant 1.3273 of the first formula by the weight of one cubic inch of mercury, this being 0.4912 pounds.

If the temperature is given in degrees Centigrade and the pressure in inches of mercury, the weight of a cubic foot of air is given by the formula,

$$W = \frac{0.7364B}{273+T}$$

Example: Find the weight of 10 cubic feet of air at 70°F and 28 inches of mercury pressure.

Answer: $w\ =\ 10\,\frac{1.3278\ x\ 28}{460+70} = 0.7012$ pound.

Example: The temperature in a 300-foot upcast shaft of a furnace ventilated mine is 320°F, and that of the downcast is 65°F. The ventilating pressure per square foot is equal to the difference in weight of a cubic foot of the air in the two shafts times the depth of the downcast, which is 300 feet. Find the ventilating pressure if the barometer reads 29 inches of mercury.

Answer: $W\ (downcast) = \frac{1.3273\ x\ 29}{460\ x\ 65} = 0.0733\ pound.$

$W\ (upcast) = \frac{1.3273\ x\ 29}{460\ x\ 320} = 0.0493\ pound.$

Excess weight per cubic foot in the downcast air = 0.0733 - 0.0493 pound-.024 pound. The ventilating pressure per square foot, then, will equal this excess weight per cubic foot times the shaft depth, or 300 x 0.024 =7.2 pounds per square foot.

Example: Find the weight of 100,000 cubic feet of air at 15 pounds per square inch and 10°F.

Answer: $w = 100{,}000\ \frac{2{\bullet}7\ x\ 15}{460+10} = 8{,}617\ pound$

Example: Find the weight of 5 cubic feet of air at 750 mm. pressure and 10°C.

Answer: $\frac{750}{25.4} = 29.52\ inches\ of\ mercury$

$W = 5\frac{0\ .7364\ x\ 29.52}{273 + 10} = 0.38407\ pound.$

2. THE RELATION OF WEIGHT AND VOLUME OF A GAS

If the weight of a gas is 0.0912 pound per cubic foot, then in one pound of the gas there will be 1/0.912 or 10.96 cubic feet.

The weight and volume of the gas V = l/Weight or l/W. gas are reciprocals. We have shown that Likewise, if we know the number of cubic feet of gas in one pound, we can find the weight by using the formula W = l/V.

Example: There are 11.5 cubic feet of gas in a pound. What is its density?

Answer: $W = \frac{l}{V} = \frac{1}{11.5} = 0.0869\ pound.$

3. WEIGHT OF GASES

In determining the weight of any particular gas at a certain temperature and pressure we first find the weight of one cubic foot of air for that temperature and pressure, then multiply by the specific gravity of the gas in question, then multiply by the quantity of gas involved.

Example: Find the weight of 10 cubic feet of carbon monoxide at 60°F and 28 inches of mercury pressure.

Answer: $Sp.gr.of\ carbon\ monoxide\ 0.967$

$Wt.of\ 1\ cu.ft.air = \frac{1.3273\ x\ 28}{460\ x\ 60} = 0.07147\ pound.$

$Wt.of\ 1\ cu.ft.carbon\ monoxide = 0.967\ x\ 0.07147 = 0.0691\ lb.$

$Wt.of\ 10\ cu.ft.carbon\ monoxide = 10\ x\ 0.0691 = 0.691\ lb.$

Example: The return air from a mine contains 1/2 of one percent methane. If the exhaust fan is passing 100,000 cubic feet per minute at 14.7 pounds per square inch pressure and 65°F, what will be the weight of methane removed from the mine in 24 hours?

Answer: $Sp.Gr.of\ methane\ =\ 0.555$

$$Wt.of\ 1\ cu.ft.of\ air = \frac{2.7\ x\ 14.7}{460 + 65} = 0.0756\ pound.$$

$$Wt.of\ 1\ cu.ft.of\ carbon\ monoxide\ =\ 0.555\ x\ 0.0756\ =\ 0.04196\ lb.$$

$$Volume\ of\ methane\ passed\ per\ minutes\ =\ 0.005\ x\ 100{,}000\ = 500\ cu.ft.$$

$$Volume\ of\ methane\ passed\ in\ 24\ hours\ =\ 500\ x\ 60\ x\ 24\ = 720{,}000\ cu.ft.$$

$$Weight\ of\ methane\ passed\ in\ 24\ hours\ 720{,}000\ x\ 0.04196\ = 30{,}212.2\ pounds.$$

4. SOLUBILITY OF GASES IN WATER

The solubility of a mine gas in water depends on the conditions surrounding the gas and water. First, the temperature of the water affects the amount of gas which can be dissolved. Second, the concentration (amount per unit of volume) of a gas dissolved in water depends on the pressure exerted by the gas.

If the student were asked to what extent carbon dioxide was soluble in water, he would have to know the conditions of temperature and pressure. If at a certain temperature and pressure 1 cubic foot of gas would dissolve in 1 cubic foot of water, then if the pressure were doubled while the temperature remained constant, the quantity of gas that would dissolve would also be doubled. <u>The concentration of the saturated solution of a gas is proportional to the pressure at which the gas is supplied. (Henry's Law)</u>

The amount of gas which will remain dissolved in water decreases as the temperature increases. Thus, water near 32°F will retain a greater amount of co2 in solution than when the temperature is 62°F. For instance, approximately 5 parts of oxygen will dissolve in 100 parts of water at 32°F at one atmosphere of pressure while approximately 3 parts of oxygen will dissolve in water at 67°F under the same pressure.

Since the student is concerned only with the solubility of water at normal atmospheric pressure and at temperatures around 60°F (normal mine temperature), the following table has been prepared to give solubility values largely under those conditions.

D. MOISTURE IN THE ATMOSPHERE

1. EVAPORATION

At normal atmospheric pressure and at all conditions of temperature below their boiling points all liquids tend to evaporate. The evaporation of water from streams, lakes and oceans keeps a certain amount of moisture in the air. Near such bodies of water the amount of water vapor in the air is naturally large, while over portions of the land surfaces of the earth which are far removed from large bodies of water there is comparatively little vapor in the air.

The importance of atmospheric moisture in mining studies lies in its influence on the health of the workmen who are employed underground, on the behavior of moisture-absorbent roof strata, and on the flammability of coal dust.

2. VAPOR TENSION AND PRESSURE OF LIQUIDS

When a liquid evaporates, the molecules of the liquid, while not so active as gas molecules, are nevertheless in a constant state of motion, and some at the surface are moving so rapidly that they break away from the liquid and enter the atmosphere. If the liquid is in a closed container, the vapor molecules above the surface act like any other gas, colliding with each other and the container walls in their flight. Eventually a point is reached where the space over the liquid is so crowded with vapor molecules that some rebound into the liquid surface and thus make room for other molecules to pass from the surface to the space above.

The vapor pressure of a liquid is the pressure exerted by the vapor molecules over the surface. When there is an equilibrium established between the molecules leaving and entering the liquid, the vapor pressure reaches its maximum value. The amount of this maximum pressure depends on (a) the type of liquid, (b) the temperature, and (c) the atmospheric pressure.

We know that gas molecules vary in their inherent energy, so that some have greater velocities than others. The same holds true for liquids, some vaporizing more readily than others, and the rate of evaporation depends on the activity of the liquid molecules. For instance, ether evaporates quite rapidly, water less rapidly, and crude oil least rapidly of the three liquids. The ability to evaporate is a distinctive quality which we may associate with a liquid. The maximum vapor pressure, then, is a measure of the maximum evaporation ability and represents the vapor tension of the liquid.

The effect of temperature on a liquid is to increase the activity of its molecules, thus increasing the rate of evaporation. With each change in temperature there is a new maximum vapor pressure and a new vapor tension. The temperature at which the vapor tension of a liquid equals the vapor pressure of the atmosphere is called the "boiling point" of the liquid.

Atmospheric pressure affects the boiling point of a liquid. At sea level the boiling point of water is 212°F. On top of a high mountain, the lower atmospheric pressure may require the water temperature to be raised only to 180°F to have the vapor tension equal the atmospheric pressure.

If a current of air passes over a body of water and reduces the quantity of vapor in the atmosphere, the pressure over the water is decreased and the rate of evaporation increases. We use fans during warm weather to move the hot air and permit evaporation of the moisture on the surface of our bodies. This evaporation cools our bodies and gives us some degree of comfort.

3. QUANTITY OF MOISTURE IN SATURATED AIR

The atmosphere cannot hold more than a certain quantity of water vapor at any particular temperature and pressure. When this condition is reached, we say that the air is saturated with moisture. Since the quantity varies with the temperature and pressure, we would expect there to be a new saturation point for each condition of temperature and pressure. We will normally be occupied with problems of moisture in air at atmospheric pressure and may neglect any change in pressure. The following table gives the amount of moisture in saturated air at different temperatures and standard atmospheric pressure (29.92 inches of mercury).

4. DEW POINT

If we study the table showing the moisture content of saturated air, we will note that warm air will hold more moisture than cool air. When warm air, which contains moisture but is by no means in a saturated state, is cooled to a temperature at which the now cooler air is saturated, then, any additional cooling causes the air to deposit its excess moisture. This temperature is called the "dew point."

We are familiar with this deposition of moisture in the form of "dew" on grass and other vegetation. Inside mines the same process during the summer months results in the condition known as "sweating.” The warm air entering the cool mine deposits its excess moisture on the bottom, sides, top, timbers and other objects, but it is most evident when beads of water hang suspended from the roof rock. Reasoning from the facts which we know, the cool air which is exhausted from the mine in the summer months should be able to absorb moisture as it warms in the atmosphere; certainly, any one standing in the draft of such mine air is quickly chilled because of the evaporation of the moisture on the skin, partially by the movement of air but also by reason of the moisture absorbing quality of the air. In the cold months, some of the moisture absorbed from the warm mine workings is deposited by the air upon reaching the colder atmosphere outside, and if the temperature is below freezing, gives evidence of this fact by the coating of ice which can be seen on vegetation or objects in the path of the air current.

Moisture in Saturated Air		
(Pressure of 29.921 inches of mercury)		
Temperature °F	Wt. in lbs. per cu ft.	Gallons per 100,000 cu ft.
-20	0.000024	0.284
-10	0.000041	0.488
0	0.000069	0.825
10	0.000111	1.328
20	0.000177	2.118
30	0.000276	3.303
40	0.000409	4.895
50	0.000587	7.025
60	0.000829	9.922
70	0.001152	13.788
80	0.001576	18.863
90	0.002132	25.517
100	0.002848	34.087
110	0.003763	45.039

5. RELATIVE HUMIDITY

Relative humidity is a term used to indicate the ratio of the amount of moisture in the air at a particular moment to the amount which the air would hold if in a saturated condition. It is usually expressed as a percentage.

Example: If the moisture content of the atmosphere is 0.000715 pounds per cubic foot at 60°F, what is the relative humidity?

Answer: At 60°F saturated air holds 0.000829 pound per cubic foot.

$$Re1ative\ Humidity = \frac{0.000715}{0.000829} = 0.86 = 86\%$$

Measurement of Relative Humidity

The amount of moisture in the air can be measured by devices which operate either on chemical or physical principles. Since the physical measuring devices are the type normally used around coal mines, we will limit our discussion to the principles involved in such devices.

If a little quantity of air can be cooled -- not separated from the rest of the atmosphere -- and its dew point determined, then the relation of the two temperatures and the vapor pressures at these temperatures will allow us to calculate the relative humidity of the air. We have said that "relative humidity of the air is the ratio of the amount of moisture actually present to the amount if the air were saturated." This may be restated as "relative humidity is the ratio of the pressure of the water vapor present to the pressure of the water vapor if saturated at the same temperature." Tables are available from which these values can be obtained. Since it is more convenient to use a chart than to select values from tables, the practice of finding the relative humidity will be described in which the final answer is found on a chart or graph. This graph contains essentially the same information as the vapor pressure tables but in a form adapted for use with instruments for finding the relative humidity by physical means.

The instruments employing physical means to find the relative humidity of the air at any temperature are called hygrometers. There are two general types, one of stationary design and the other a portable type, but both employ the same principles. The instrument consists of two thermometers attached to a frame, and for mine service the thermometer scales usually read in degrees Fahrenheit. One thermometer bulb has a muslin sack attached to it, the other is uncovered. When the muslin sack is wet, and air flows around the bulbs at a rate of 10 to 15 feet per second, the wet bulb thermometer shows a lowered temperature which finally becomes stationary and indicates the dew point of the air. At the same time the dry bulb thermometer gives the temperature of the air. The two readings, when used with the chart in Figure PH-5, give the relative humidity of the air.

Example: If the dry bulb temperature of a hygrometer is 70°F and the wet-bulb temperature is 60°F, what is the relative humidity?

Answer: 70° - 60° 10°F depression of wet bulb.

Using the chart in Figure 73, we select on the lower edge the point indicating 70°F. From this point we trace a vertical line until it intersects the line running horizontally from the point on the left side which indicates 10°F wet-bulb depression. This intersection falls between the curves for 50% and 60% and indicates approximately 55% relative humidity.

The water in the wet muslin sack is continually evaporating, because of the passage of air around it, and this cools the bulb around which the sack is wrapped. The wet-bulb readings, therefore, are normally lower than the dry-bulb readings. The amount of cooling depends upon the rate of evaporation, and this in turn depends upon the amount of moisture in the air; the rate of evaporation is greatest when the moisture content of the air is least. If we had a saturated atmosphere, there would be no evaporation from the muslin sack and the two thermometers would give the same readings.

Stationary hygrometers obtain evaporation by having the muslin sack extend into a jar or reservoir filled with water. The sack draws moisture by capillary action. The successful use of this type depends on the amount of air velocity obtainable; the velocity should be as near 15 feet per second as possible in order to obtain the dew point with accuracy.

Portable hygrometers, or as they as known, sling psychrometers (see Figure PH-6), are used to a greater extent in mining practice than the stationary type. Evaporation is obtained by first wetting the muslin sack, then whirling the frame to which the thermometers are attached around the handle. This action produces the same effect as the velocity of the air had on the stationary hygrometer; the velocity of the thermometer bulbs when the psychrometer is swinging should be at least 15 feet per second. Accuracy demands that several readings be obtained in order to check results. The muslin sack must be wetted before each test. The sling psychrometer gives accurate tests in still as well as moving air and is considered the most accurate of the relative humidity measuring devices.

E. BASIC PHYSICS QUESTIONS AND ANSWERS

PH Q-1: What does a thermometer indicate?
PH A-1: Temperature

PH Q-2: What is specific gravity?
PH A-2: The specific gravity of a substance is the number expressing how many times that substance is heavier, volume for volume, than a substance taken as a standard.

PH Q-3: 32 degrees Fahrenheit temperature is what degree Centigrade temperature?
PH A-3: Zero degrees.

PH Q-4: Absolute temperature is equal to the Centigrade temperature plus what number?
PH A-4: 273

PH Q-5: How many gallons of water are there in a cubic foot?
PH A-5: 7.48 gallons

PH Q-6: How many cubic inches are there in a gallon of water?
PH A-6: 231 cubic inches

PH Q-7: The weight of a water column one inch square and one foot high is?
PH A-7: 0.434 pounds

PH Q-8: A cubic foot of water weighs how much?
PH A-8: 62.5 pounds

PH Q-9: One horsepower equals how many watts?
PH A-9: 746

PH Q-10: What is meant by the density of a substance?
PH A-10: Density is mass per unit volume.

PH Q-11 Find the weight of 20 cubic feet of bituminous coal with a specific gravity of 1.4.
PH A-11 $1.4 \ x \ 62.5 \ = \ 87.5 \ pounds \ per \ cubic \ foot$
$20 \ x \ 87.5 \ = \ 1750 \ pounds \ weight \ for \ 20 \ cubic \ feet$

PH Q-12 How much will 7 cubic feet of carbon dioxide weigh at standard conditions?
PH A-12 $7 \ x \ 1.529 \ x \ 0.08072 \ = \ 0.864 \ pound$

PH Q-13 Find the weight of 20 cubic feet of methane at standard conditions (0°C and 760 mm pressure).

PH A-13 20 x 0.555 x 0.08072 = 0.896 pound

PH Q-14 How is atmospheric pressure measured?

PH A-14 Atmospheric pressure is measured by an instrument called the barometer. The different types of instrument for measuring atmospheric pressure are: (i) mercurial barometer, (ii) aneroid barometer, and (iii) barograph.

PH Q-15 What is Charles 'Law?

PH A-15 Charles ' Law $\frac{Q_1}{Q_2} = \frac{T_1}{T_2}$

It has been found by experiment that under constant pressure that the volume of a gas is directly proportionate to the temperature.

PH Q-16 What is Boyle's Law?

PH A-16 Boyle’s Law $\frac{Q_1}{Q_2} = \frac{P_2}{P_1}$

It has been found by experiment that under constant temperature that the volume of a gas is indirectly proportionate to the pressure.

PH Q-17 If 5 cubic feet of air at a temperature of 41°F is heated under constant pressure to a temperature of 55°F, what will be the new volume?

PH A-17 41°F=460 +41=501°F absolute

55°F=460 +55=515°F absolute

$V_l : V_2 :: T_l : T_2$

$5 : V_2 :: 501 : 515$

$V_2 = \frac{5x515}{501} = 5.14 \; cubic\ feet$

PH Q-18 The air entering a mine measured at the inlet is 150,500 cubic feet per minute, while the measurement in the return airway shows a volume of 160,800 cubic feet per minute. The mine is ventilated by an exhaust fan producing a water gauge of 2.8 inches. If the temperature of the air in the intake is 40 degrees Fahrenheit and that in the return is 65 degrees Fahrenheit:

a. What estimated volume of gas is being produced in the mine per minute?

b. What is the percentage of gas in this current?

Given:		Find:
Quantity Intake (Q_I)	=150,500 c.f.m.	a. Quantity Gas (Q_G)
Quantity Return (Q_R)	=160,800 c.f.m.	....
b. Percent Gas (X_G)		
Water Gauge (i)	=2.8 inches	
Temperature Intake (T_I)	=40°F	Work problem in absolute temp.
Temperature Return (T_R)	=65°F	A°= 460°+F°

PH A-18 a. Increased volume to temperature

$$\frac{Q_1}{Q_1} = \frac{T_I}{T_R}; \frac{Q_1 x\, T_R}{T_I} = 150{,}500\ c.f.m.x \frac{460° + 65°}{460° + 40°}$$

$$Q_1 = \frac{150{,}500\ c.f.m.x\ 525°}{500}° = \frac{79{,}012{,}500\ c.f.m.°}{500} = 158{,}025\ c.f.m.$$

Increased volume due to fan pressure

$$P_1 = Atmospheric\ Pressure\ -\ 14.7\frac{lb}{in^2}\ x\ 144\frac{in^2}{ft^2} = 2116.8\frac{lb}{ft^2}$$

$$P_2 = Atmospheric\ Pressure - Fan\ Pressure$$

$$Fan\ Pressure = 2.8\ in\ W.G.x\ 5.2\frac{lb}{ft^2}\ \ per\ inch\ W.G. = \ 14.56\frac{lb}{ft^2}$$

$$P_2 = 2116.8\frac{lb}{ft^2}\ -\ 14.56\frac{lb}{ft^2} = 2102\ .24\frac{lb}{ft^2}$$

$$Q_2 = \frac{158{,}025\ c.f.m.x\ 2116.8\frac{lb}{ft^2}}{2102.24\frac{lb}{ft^2}} = \frac{334{,}507{,}320\ c.f.m.\frac{lb}{ft^2}}{2101.24\frac{lb}{ft^2}} = 159{,}119\ c.f.m.$$

$$Q_G = Q_R - Q_2 = 160{,}800\ c.f.m. - 159{,}119\ c.f.m. = 1681\ c.f.m.$$

b. $\%_G = \frac{Q_G}{Q_R}x100 = \frac{1681\ c.f.m.}{160{,}800\ c.f.m}x100 = 0.0105x100 = 1.05\%$

PH Q-19 What would be the weight of one cubic foot of air if the barometer reads
29 inches of mercury and the temperature is 65 degrees Fahrenheit?

Given:		Find:
Barometer Reading (B)	=29 inches	Weight of 1 ft^3 of air
Temperature Fahrenheit (T_F)	=65°F	

PH A-19

Temperature Absolute (T_A) =460° +T_F

T_A =460° +65°=535°A

$$Weight\ of\ Air\ = \frac{1.3273\ lb\ \frac{°A}{in} x\ B}{T_A} = \frac{1.3273\ lb\ \frac{°A}{in}\ x\ 29\ in}{525°A}$$

$$= \frac{38.4917\ lb\ °A}{525°A} = 0.0733\ lb$$

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 1

SECTION 1

A. ELECTRICAL DEFINITIONS

AC: Alternating current. An electrical current that reverses its direction and changes in amplitude at regular intervals.

Arc: Intense light and heat created when an electric circuit is opened.

Ammeter: An instrument that is used to measure current flow in a circuit.

Ampacity: The current carrying capacity of a conductor. Ampere: The unit of current of flow is an ampere.

Approval plate: The current produced by one volt applied across a resistance of one ohm. A metal plate, the design of which meets the regulatory requirements, for attachment to an approved machine or accessory, identifying it as satisfactory for use in anthracite and bituminous mines.

Battery: Source of DC electricity created from a chemical reaction.

Branch circuit: A branch circuit shall be any tap taken off a main circuit.

Cable: Insulated copper or aluminum wires contained in an insulated outer jacket used to conduct power to electric equipment.

Capacitor: An electrical device that will store electrical energy. A capacitor opposes a change in voltage.

Circuit: A network of electrical components connected in series or in parallel through which current can flow.

Circuit-breaker: A device which may be controlled by relaying or protective equipment for interrupting a circuit between separable contacts under normal or abnormal conditions.

Combustible: Capable of burning, i.e., coal, methane, wood, etc.

Conductance: 1/resistance. The readiness with which a conductor transmits electrical current.

Conductor: Materials which offer little opposition to the flow of current. Examples are copper, aluminum, and iron.

Cross-sectional area: The area of the end of a conductor when cut at right angles to its length.

Current: Movement of electrons in a conductor. The unit of measurement is an ampere. The electrical symbol is an I.

DC: Direct current. . An electrical current that flows in only one direction.

De-energize: To disconnect an electrical circuit from its source of power.

Derived Neutral: A neutral point of connection established through the use of a "zig-zag" or grounding transformer with a normally ungrounded delta power system.

Diode: An electrical device which allows current flow in one direction only. Diodes are commonly used as rectifiers to change AC to DC.

Direct neutral: A neutral conductor that does not have to be derived. A direct neutral is connected directly to the transformer neutral point.

Distribution box: An enclosure through which one or more portable cables may be connected to a source of electrical energy, and which contains circuit breakers with tripping devices which will function on overload, phase fault and ground fault for each outgoing cable.

Efficiency: Output/input.

Electric circuit: An electric circuit is a closed path through which an electric current can flow.

Electrolysis: An action of an electrical current which carries away the particles of a conductor.

Electrocution: Death from an electric current flowing through the body. Electron: A negatively charged atomic particle. Enclosure (electrical): A metallic box or frame that encloses electrical equipment or circuits.

Energized: A circuit that has electric power connected to it.

Energy: Is the rate of which power is utilized. It is measured in watt-hour.

Explosion or flame proof: Explosion or flame proof casings or enclosures are those which, when completely filled with a mixture of methane and air, and the same exploded, are capable of either entirely confining the products of such explosion within the casing or of so discharging them from the casing that they cannot ignite a mixture of methane and air, combined in proportions most sensitive to ignition and entirely surrounding the points of discharge, and in most intimate proximity therewith.

Face Equipment: Mobile or portable mining machinery having electric motors or accessory equipment normally installed or operated inby the last open crosscut in any entry or room.

Flame arresting path: Two or more adjoining or adjacent surfaces between which the escape of flame hot enough to ignite an explosive mixture of methane is prevented.

Flame resistant: Material that will burn when held in a flame, but will cease burning when the flame is removed.

Frequency: Number of cycles per second. Unit is Hertz. The standard frequency of AC in the United States is 60 Hertz.

Gassy mine: Notwithstanding any other provision of Law, the distinction between gassy & non-gassy mines is eliminated, & all underground bituminous mines shall comply with the requirements for gassy mines.

Grounded circuit: A circuit that is either intentionally or accidentally connected to the ground.

Hazard: Any condition that may result in, or contribute to an accident.

Hazard location: An area in which flammable or explosive gases, vapors, and dust can be encountered.

Hazardous reduction: As the probability of ignition cannot be brought to zero, such systems used to reduce the overall hazard are termed as hazard reduction techniques.

High voltage: Any voltage greater than 1000 volts.

Horsepower: A unit of measurement of power. One horsepower equals 746 watts.

Hydrometer: An instrument used to determine the specific gravity of liquids. A hydrometer is used to check the electrolyte in lead acid batteries used in mining.

Imminent danger: The existence of any condition or practice which could reasonably be expected to cause death or serious physical harm before such practice or condition can be abated.

Impedance: Total ohmic opposition to current flow in an AC circuit. The unit of measurement of impedance is the ohm. The symbol is Z.

Inactive workings: Includes all portions in a mine in which operations have been suspended for an indefinite period, but have not been abandoned.

I**nsulation**: Non-conducting material such as rubber or plastic used to cover electric conductors.

Insulator: Insulators are materials that offer opposition to the flow of electrical current. Examples of insulators are air, glass, rubber, and wood.

Intrinsically safe: Incapable of releasing enough electrical or thermal energy under normal or abnormal conditions to cause ignition of a flammable mixture of methane or natural gas and air of the most easily ignitable composition. Available energy from any component must be less than about 0.25mJ.

Kilowatt: l kW = 1000 watts.

KVA: Kilo-volt-amps. It is the product of volts times amperes divided by 1000.

Line current: The current that flows through the line.

Low voltage supply: Where the conditions of the supply of electricity are such that the difference of potential between any two points in the circuit cannot exceed 660 volts.

Medium voltage supply: Where the conditions of the supply of electricity are such that the difference of potential between any two points in the circuit may at any time exceed 660 volts, but cannot exceed 1000 volts.

Megohmmeter: An instrument which supplies a voltage for testing purposes. A Megohmmeter is sometimes called a Megger because it is calibrated to read very high resistance values. Megohmmeters are normally used for testing insulation resistance in electrical machines.

Multimeter: An electrical testing meter capable of measuring AC and DC volts, amperes, and ohms. The meter is usually provided with adjustable settings and ranges in order to equip the electrician for a number of electrical testing functions.

Ohmmeter: An instrument to measure electrical resistance in ohms.

Ohm: The unit of electrical resistance of a material is termed as an ohm.

Ohm's Law: The current is directly proportional to the electromotive force and inversely proportional to resistance.

Open circuit: An open or break in an electric circuit either intentional or unintentional.

Overload: Operation of equipment in excess of normal, full load rating, or of a conductor in excess of rated ampacity which, when it persists for a sufficient time would cause damage or dangerous overheating.

Parallel circuit: An electrical circuit having more than one path for current flow.

Phase Current: Current that flows through a particular phase winding of a wye or delta connected transformer (or an AC motor).

Potential Difference: The difference of electrical pressure or electromotive force existing between any two points of an electrical system, or between any point of such a system and the earth, as determined by a voltmeter or other suitable instrument. The terms "potential" and "voltage" are synonymous and mean electrical pressure.

Power: Power is the rate of doing work. In electrical measurement the unit of power is watt.

Power factor: The ratio of watts divided by volt-amperes expressed in decimal. Real power divided by apparent power.

Rectifier: A device that converts alternating current into direct current.

Qualified: Electrician: An electrician, who through training and experience, has shown that he can perform electrical work safely and efficiently.

Resistance: The electrical resistance of a conductor is a measure of the opposition which the electrons experience as they try to find their way through the conductor.

Resistor, Grounding: A resistor connected in series with the system neutral and the ground wire for the purpose of limiting the fault current that would flow in case of a line to ground fault.

Resistor, Potentiometer: A variable resistor which uses three connections.

Resistor, Rheostat: An adjustable resistance.

Semiconductor: A material having electrical properties between those of good electrical conductors and those of insulators.

Series circuit: A series circuit has only one closed path through which the current can flow.

Series-parallel circuit: A series-parallel circuit consists of two or more electrical devices connected in parallel which in turn is connected in series with other electrical devices.

Shock (electrical): Sensation caused by an electric current flowing through the body.

Short circuit: A short circuit occurs when current takes a path short of its intended circuit. The current therefore -fails in its full circuit and also fails to perform its function.

Single phase: If only one current is alternating, it is known as single phase current. This system requires two power conductors and a grounding wire.

Splice: Mechanical joining of conductors that have been separated. Cable jacket repairs involving conductors or conductor insulation are considered temporary splices.

Switch: Device that is used to disconnect power from an electric circuit.

Sulfuric acid: Solution used in batteries to create electric power, which can severely burn human tissues.

Three phase: If three phases are alternating, it is known as three phase current. Three phase systems require three power conductors and a grounding wire.

Trailing Cable (Portable): A portable cable is a flexible cable or cord used for connecting mobile, portable, or stationary equipment in mines to a trolley system or other external source of electric energy where permanent mine wiring is prohibited or is impracticable.

Transformer: An electrical device used to change the voltage of one circuit to a different value for another circuit. A transformer may also be used to isolate one circuit from another circuit.

Trolley Wire: Bare copper conductor supported from the mine roof that supplies DC power to electric locomotives or off track equipment.

Trolley feeder wire: Bare copper or aluminum conductor paralleling the trolley wire that supplies DC electric equipment.

Volt or Voltage: The unit of electromotive force, potential difference is the voltage or volt.

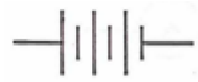
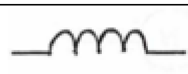
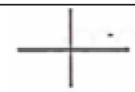
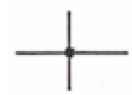
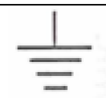
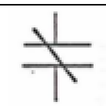
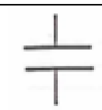
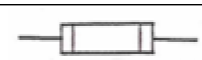
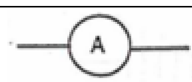
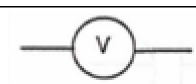
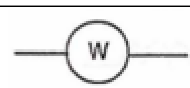
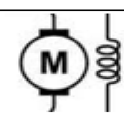
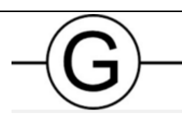
Voltage drop: The amount of voltage used in forcing current flow through any resistance. In a complete circuit, all the applied voltage is always used. No pressure returns to the power source.

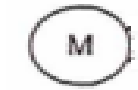
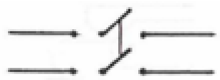
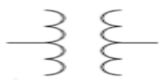
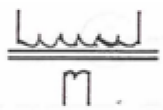
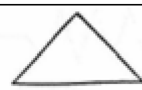
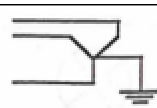
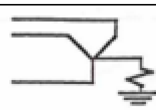
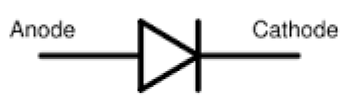
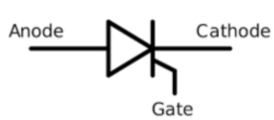
Voltmeter: The instrument to measure the electromotive force, or potential difference. The unit is Volt.

Watt: A unit of electrical power equal to a current of one ampere under one volt of pressure.

Work: Work is done whenever resistance is overcome. The work accomplished is measured in foot-pounds.

B. ELECTRICAL SYMBOLS

Device	Standard Symbol
Battery	
Capacitor	
Coil, nonmagnetic core	
Coil, magnetic core	
Wire crossing (No connection)	
Wire connection	
Ground connection	
Contacts, normally closed	
Contacts, normally open	
Fuse	
Ammeter	
Voltmeter	
Wattmeter	
DC motor or generator armature	
Single-phase generator	

Electric motor	
Adjustable resistor (rheostat)	
Switch, single-pole, single-throw	
Switch, double-pole, double-throw	
Push button, normally open	
Push button, normally closed	
Transformer	
Current transformer (CT)	
Potential transformer (PT)	
Delta connection	
Wye connection	
Solidly grounded	
Resistance grounded	
Semiconductor diode	
Thyristor or SCR	

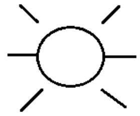
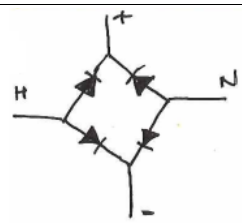
Light bulb	
Resistor	
Circuit breaker	
Rectifier	
Lighting arrestor	
Overload Device	

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 2

SECTION 2

MINE ELECTRICAL FUNDAMENTALS

A. ELECTRICAL FUNDAMENTALS

Electricity is used in the coal mining industry for many purposes. It serves, broadly, to furnish light, heat, and power. Some common applications of electricity in mining are the operation of lights, signal devices, controls, and electric motor-operated equipment. The advantage in the use of electricity as a medium of transfer of energy lies in the ease of control, adaptability to a multiplicity of functions, and the comparative ease of distribution. This wide application necessitates a basic knowledge of electrical principles on the part of those persons who must supervise the operation of electrically-operated machines and electrical distribution systems.

1. GENERAL ELECTRICAL EFFECTS

Electricity is not visible, but it does make itself evident by one of the following effects:

- An electric current, when flowing in a coil or wire, produces a magnetic effect very similar to that of the ordinary horseshoe magnet with which we are familiar.
- An electric current, when flowing through a conductor, produces heat.
- A current flowing through a liquid produces a chemical effect.
- The physical effect of shock when a wire carrying electric current is touched needs no explanation.

The study of electricity is based on these effects, and these effects can take place only when electricity is in motion. Therefore, throughout this study of electricity, keep in mind that this is the study of electricity in motion.

2. ELECTRICITY AS A FLOW OF ELECTRONS.

According to modern theory, which has been substantiated by the experimental results of many investigators, the atoms of all matter consist of a positively charged nucleus around which infinitesimal negative charges rotate with high angular velocity. The individual negative charges, called electrons, are found to be identical for all matter. Most of the electrons in matter are parts of atoms, but it is possible for the electron to exist in a free state apart from the atom. Free electrons exist to some extent in gases, liquids, and solids, but are much more plentiful in some substances than others. Metals contain a greater quantity of free electrons than non-metallic substances such as rubber and glass, which accounts for the better conductivity of electricity in metals than in substances having fewer electrons. Apparently this is due to free electrons, which are continually in a state of motion or agitation, seeking an affinity with a positive charge; if some means is provided to attract or repel them in quantity, a flow of electricity results. The existence of these free electrons makes possible a flow of electricity. It is believed that the actual travel of individual electrons is not very rapid, probably only a few inches per second, but as each one moves it so influences the next one that the pulsation travels through the solid with very great velocity. It takes only about 1/17 of a second for this pulsation to travel along a telephone wire from Boston to San Francisco, approximately 3000 miles.

3. CURRENT AND AMPERES.

If a stream of water flows through a pipe, the rate of flow might be taken as the number of gallons that pass a given point in a minute. In like manner, when a current of electricity flows through a wire, the rate of flow is the number of electrons that pass a given point per second. It does not matter how fast the individual electrons, which make up the current, are traveling when they pass a point. What does matter is how many of them pass in one second. The unit of current flow known as the <u>Ampere</u>. It is, of course, impossible to count the number of electrons, but a reliable measure or their effect can be secured by a device as they pass through it. This device is called an <u>ammeter</u>.

4. RESISTANCE AND OHMS.

Electrons flow along an electric conductor when an external force tends to make them move. The progress of an electron, however, is impeded by many collisions with other electrons crossing or moving along its path. The electrical resistance of a conductor is a measure of the difficulty which electrons experience as they try to find their way through the conductor. Heat is always developed in a conductor because of this resistance, and the temperature will rise as a result of current flow if the heat developed is not carried away.

5. LAWS OF RESISTANCE.

The resistance of a material depends upon four factors:

- **THE LAW OF LENGTH**. The resistance of a conductor is directly proportional to its length. This means that 1000 feet of a wire has twice as much resistance as 500 feet of the same wire.
- **THE LAW OF AREA**. The resistance of a conductor is inversely proportional to its cross-sectional area or to the square of its diameter. This means that the larger a wire is, the less is its resistance.

The diameters of wires are expressed in mils, and the cross-sectional areas are expressed in circular mils. A mil is 1/1000 inch, and a circular mil is the area of a circle having a diameter of one mil. A number 10 (A.W.G.) wire is approximately 100 mils in diameter and has a cross-sectional area of 100 x 100 or 10,000 circular mils. The square of the diameter in mils is equal to the cross-sectional area in circular mils.

- **THE LAW OF MATERIAL**. The resistance of a conductor is dependent upon the material of which it is composed. Examples of good conductors are: copper, silver, aluminum, gold, lead, and carbon. Materials that are poor in conducting electricity are called insulators. Examples of good insulators are: air, oil, glass, plastics, and rubber.

- **THE LAW OF TEMPERATURE FOR METALLIC CONDUCTORS**. The resistance of all metallic conductors increases with an increase of temperature. For example, a tungsten lamp may have ten times as much resistance when hot as when cold.

The resistance of some materials becomes less when they are heated. Carbon and most insulating materials are in this class. There are some alloys that have almost the same resistance, whether hot or cold.
Resistance is given as ohms per foot for a given conductor size, this is needed in calculation to determine correct gauge conductors for an application.

$$Conductance = \frac{1}{Resistance}$$

6. ALGEBRAIC EXPRESSION OF THE RESISTANCE LAWS.

By knowing the effect of the length, diameter, temperature, and the material on the resistance of a conductor, the amount of resistance can be computed. The relationship of these factors to resistance (R) can be expressed algebraically as, $R = \frac{KL}{D^2}$; where K is the resistance per mil-foot at a given temperature, L is the length in feet, and D is the diameter in mils.

Example: Find the resistance of a copper wire 2000 feet long, 10 mils in diameter, and at 20 C. K for copper at 20 C is 10.4 ohms per mil-foot.

Solution: $R = \frac{KL}{D^2}$ $R = \frac{10.4 \, x \, 2000}{10 \, x \, 10} = 208 \, ohms$

7. POTENTIAL DIFFERENCE, ELECTROMOTIVE FORCE, AND VOLTAGE.

The electrons flowing in an electric circuit are under the influence of a physical force which pushes them constantly. If this force were removed, the electrons would slow down and finally stop because the resistance they encounter robs them of their energy. In order to have a steady rate of flow of electrons through a conductor offering a resistance, there must be an energy difference between any two points of the conductor. This difference in energy-level is called the potential difference which constitutes the electromotive force (emf) to push the electrons along the conductor. The unit of electromotive force, potential difference, or voltage is termed a volt. The volt is the electrical potential which, when steadily applied to a conductor whose resistance is one ohm, will cause a current of one ampere to flow.

8. OHM' S LAW.

There is a definite relation between current, voltage, and resistance. The relationship of the electrical circuit is expressed by Ohm's Law, which states that "the current is directly proportional to the electromotive force and inversely proportional to the resistance.

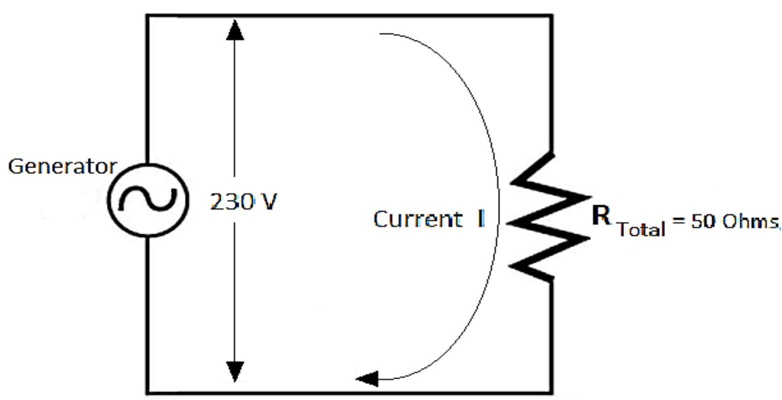
I	=	CURRENT	=	AMPS
E	=	VOLTAGE	=	VOLTS
R	=	RESISTANCE	=	OHMS

$Amperes = \frac{Volts}{Ohms}$ or $I = \frac{E}{R}$ where I represents the current in amperes, E represents the emf in volts, and R represents the resistance in ohms.

If we know the values for any two of the factors in the equation, the third can be found. A simple way to remember the relationship of E, I, and R is to set them up in the following manner:

When it is desired to find an unknown factor, place a finger over the known in this form and read the relationship of the other two.

Example: A generator having an emf of 230 volts sends current through a circuit having a total resistance of 50 ohms. What will the current be?



Solution: $I = \frac{E}{R} = \frac{230}{50} = \frac{23}{5} = 4.6\ Amperes$

Example: What emf is needed to send a current of 125 Amp through a circuit having a total resistance of 1.8 Ohms?

Solution: $E = I \, x \, R = 125 \, x \, 1.8 = 225 \, volts$

Example: If 230 Volt draws a current of 40 Amp, what is the resistance of the motor?

Solution: $R = \frac{E}{I} = \frac{230}{40} = 5.75 \, Ohms$

B. ELECTRIC CIRCUITS.

An electric circuit is a closed path through which an electric current can flow. Electric circuits usually include a source of emf and one or more devices which convert electrical energy into some other form or energy, such as light, heat, mechanical movement, or sound. In order to understand how mining electrical equipment functions, it is necessary to be able to recognize the several simple types of electrical circuits and to be able to perform simple calculations in connection with these circuits.

1. UNITS IN SERIES.

Some mining electrical devices are used in a series circuit, illustrated by Figure MES 2-1. A generator and three load units are shown connected in series.

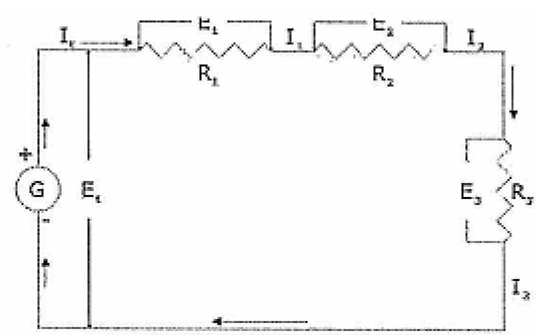


Figure MES 2-1: Series Circuit

A series circuit has only one closed path. There are no points along it where the current may branch off. It consists of electrical units which are connected in tandem so that the charge must flow through each in succession. The outstanding feature of a series circuit is that the current, in amperes, must necessarily have the same value in every unit in the circuit. With this fact in mind, the equation expressing the relationship of the total current in a series circuit to the current in each electrical device becomes: $I_t = I_1 = I_2 = I_3, = etc.$ where I_t is the total current across the power source and $I_1, I_2, and\ I_3$ represent the current values in each load unit.

As the charge progresses from the positive side of the generator to the ground, it loses energy in each unit. It follows that the total voltage drop must be divided between the several load units. The algebraic equation showing this relationship is:

$$E_t = E_1 + E_2 + E_3 + etc.$$

Where E_t is the total emf and $E_1, E_2, and\ E_3$ are the emf values across the individual load units in the total circuit.

Through the use of Ohm's Law and the characteristics of the current and voltage in a series circuit, the resistance in ohms can be calculated by use of the formula, $R = \frac{E}{I}$.. Since the current in a series circuit is the same in all parts, the resistance of the units will be found to be in the same ratio to each other as the observed voltage drops. In other words, if the potential difference across an electrical device is large with a unit flow of current, the resistance of the device is also large. The total resistance of a series circuit is equal to the sum of the individual resistances in the circuit. This is expressed by the formula $R_t = R_1 + R_2 + R_3 + etc.$ where R_t is the total resistance of the circuit and $R_1, R_2, and\ R_3$ are the values of the individual resistors in the circuit.

Example: The resistance of a conductor is 5 Ohms, a faulty junction offers 10 Ohms resistance, and a motor has a 60 Ohm resistance. If these are connected in series to a 75-Volt circuit, what is the total current?

Solution: $Total\ Resistance = R_t = R_1 + R_2 + R_3 = 5 + 10 + 60 = 75\ Ohms$

$$Total\ Current = I_t = \frac{E_t}{R_t} = \frac{75}{75} = 1\ ampere$$

2. UNITS IN PARALLEL.

The majority of the electrical devices in modern mining are operated in parallel, being connected together in the manner shown in Figure MES 2-2. This figure schematically illustrates a generator and three load devices. One side of each device is grounded; the other side is connected to the positive side of the generator. Switches and meters have been omitted from the diagram for simplicity.

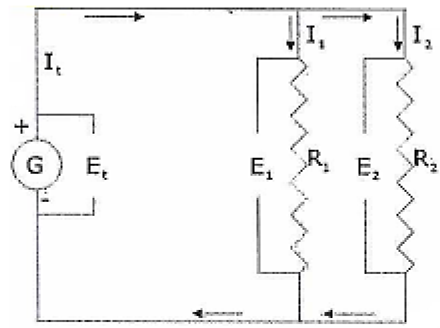


Figure MES 2-2: Parallel Circuit

When two or more electrical devices are thus connected, with one terminal of each attached to a common point or to the same conductor, and their other terminals similarly attached to another common point or conductor, they are said to be in parallel. A parallel circuit is one in which there are two or more load units in parallel. The full emf of the energy source is therefore applied separately to each load unit regardless of the nature of the electrical device in each of the branch circuits. In other words, if the leads of a voltmeter are placed in contact with the terminal of any load unit designed for parallel operation with other units, the meter will show almost the same deflection in every case. The emf values will decrease slightly as the testing points move away from the generator, because of the line losses.

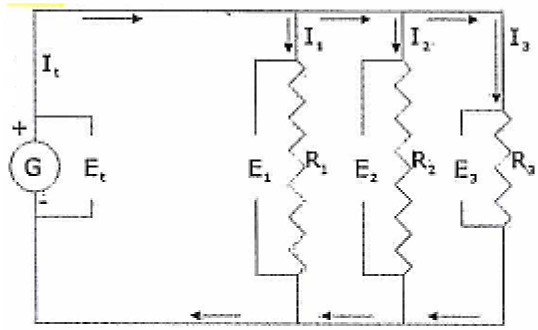
The relationship of the total emf across the power source terminals in a parallel circuit to the emf in each parallel branch can be shown by the equation: $E_t = E_1 = E_2 = E_3, etc.$ Where E_t is the total emf, and E_1, E_2, E_3 are the emf values across the individual parallel paths of the total circuit.

The total current is equal to the sum of the currents in the individual parallel branch circuits. The equation expressing this relationship is: $I_t = I_1 + I_2 + I_3 + etc.$ where I_t is the total amperes being pushed through the circuit, and $I_1, I_2, and\ I_3$ represents the current values of the individual branches.

The resistance of a parallel circuit, whether it is the total resistance or the resistance of any part of the circuit, can best be determined by using Ohm's Law, providing the corresponding values of current and voltage are known. In many cases the unknown resistance must be found from the values of other resistances. The total resistance or total resistance of the individual parallel paths may be calculated by use of the formula:

$$\frac{I}{R_t} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3} + etc, \qquad R_t = \frac{1}{\frac{1}{\frac{1}{R_1}+\frac{1}{R_2}+\frac{1}{R_3}+etc.}}$$

Example: Three resistances of 10.8, and 12 Ohms respectively are connected in parallel across 120 volts. What is the total resistance of this circuit?



Solution: $R_t = \frac{1}{\frac{1}{\frac{1}{R_1}+\frac{1}{R_2}+\frac{1}{R_3}+etc.}} = \frac{1}{\frac{1}{10}+\frac{1}{8}+\frac{1}{12}} = \frac{1}{\frac{37}{120}} = \frac{120}{37} = 3.25\ ohms$

A working knowledge of the above information has supplied a few stated facts:

a. The total resistance of two resistors in parallel is equal to the product of the individual resistances divided by their sums.

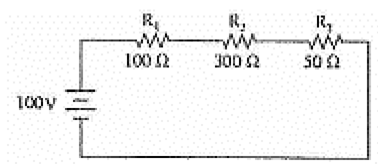
b. The total resistance of two resistors of equal value connected in parallel will be one half of either resistor value.

c. The total resistance is less than the resistance of the smallest branch.

The main comparison that needs to be taken away from series and parallel connections is that the current stays the same in series; whereas, voltage is the same across each load in parallel.

The circuits shown below are to illustrate the relationship of current, voltage drop, and total resistance of series and parallel circuits.

Series Circuit:



$$R_{total} = R_1 + R_2 + R_3 = 100\Omega + 300\Omega + 50\Omega = 450\Omega$$

$$Current = V = IR, \qquad I = \frac{V}{R_{total}} = \frac{100v}{450\Omega} = 0.22\ Amps$$

$$Voltage\ Drop\ Across\ R_1 = V = IR = (0.22A)(100\Omega) = 22v$$

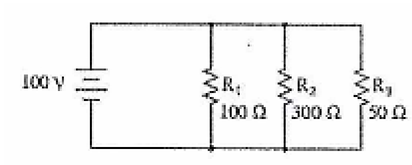
$$Voltage\ Drop\ Across\ R_2 = (0.22A)(300\Omega) = 66v$$

$$Voltage\ Drop\ Across\ R_3 = (0.22A)(50\Omega) = 11v$$

$$Source\ Voltage\ is\ equal\ to\ the\ total\ voltage\ drop\ across$$
$$R_1 + R_2 + R_3 = 22v + 66v + 11v = 100v$$

Key Points: In a series circuit, the current through each resistor is the same. The voltage across each resistor depends upon the resistance value of each load and the sums of the voltage drops are equal to the source voltage.

Parallel Circuit:



$$R_{total} = \frac{1}{\frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3}} = \frac{1}{\frac{1}{100\Omega} + \frac{1}{300\Omega} + \frac{1}{50\Omega}} = \frac{1}{0.033\Omega} = 30.0\Omega$$

Current through: $R_1 = V = IR, I = \frac{V}{R} = \frac{100v}{100\Omega} = 1\ Amp$

$R_2 = \frac{100v}{300\Omega} = 0.33\ Amp$

$R_3 = \frac{100v}{50\Omega} = 2\ Amp$

Key Points: In a parallel circuit, the current in each branch splits and maybe different based on the branches' effective resistance. The voltage drop across each branch is the same and is equal to the source voltage.

3. UNITS IN SERIES-PARALLEL.

A series-parallel circuit consists of two or more electrical devices connected in parallel which in turn is connected in series with other electrical devices. Figure MES 2-3 is an example of such a circuit. The current flows from the positive side of the generator through the conductor whose resistance is indicated by R_1 to a common junction box. Here the current is divided into the four parallel paths. Each part is formed by the electrical circuit of one of several machines in a coal production unit with each circuit represented in the illustration by $R_2, R_3, R_4, and\ R_5$. The currents from the parallel branches rejoin at the common ground and flow back to the negative side of the generator.

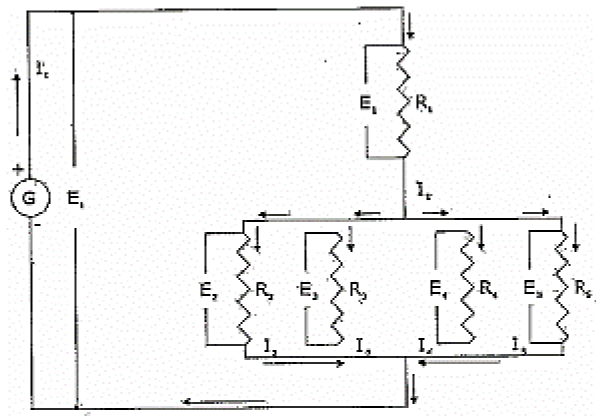


Figure MES 2-3: Series-Parallel Circuit

Series-parallel circuits may be solved by application of the rules already given for simple series and parallel circuits. To do this, the series-parallel circuit is reduced to an equivalent, simplified circuit. Each parallel group is first replaced by its equivalent single resistance value, and the entire circuit is treated as a series circuit.

Example: Three resistors are connected as follows: R_1(=24 ohms) in series with R_2(=18 ohms) and R_3(=16 ohms) which are in parallel. Calculate the equivalent resistance, total resistance and current flow.

Equivalent resistance: R_2 and R_3 reduces to $\frac{18\ x\ 36}{18\ +\ 36}\ =\ 12\ ohm$

Total resistance = R_1 in series with R_2 and R_3 which are in parallel = 24 + 12 = 36 ohm

$$Current\ =\ I\ = \frac{V}{R} = \frac{98\ v}{36}\ =\ 2.72\ A$$

4. HEATING EFFECTS OF DIRECT CURRENT ELECTRICITY.

We have learned that when electricity flows through a conductor it expends electrical energy in overcoming resistance and, hence, generates heat. This transformation of electrical energy into heat energy is quite useful to the mining industry, making possible the use of heater units, various safety devices, electric welders, and others. It is the undesirable formation of large quantities of heat in mining equipment, transmission cables, and the like that results in the creation of many hazards and excessive operating costs.

Experimental facts have proved a definite relationship between electrical energy and heat energy. The law developed from these facts states: The amount of heat generated in a conductor is directly proportional to the resistance, the time, and the square of the current. If H stands for heat, R for resistance, t for time in seconds, and I· for current, we can express the relationship in the form of an equation as follows:

$$H = I^2Rt \; (in\, Joules)$$

A Joule is defined as a unit of energy or work equivalent of energy expended in one second by an electric current of one ampere in a resistance of one ohm.

5. ELECTRICAL POWER CALCULATIONS.

WORK. Work is done whenever resistance is overcome. Lifting a weight, pulling a car along a track, or turning a wheel against a roller bearing requires a certain amount of work in each case to overcome friction. The work accomplished in each case would be measured in foot-pounds. In lifting a weight of 50 pounds to a height of 5 feet there is 5 x 50 or 250 foot-pounds of work done. It does not matter how long it takes to do a certain piece of work since times does not enter into the measurement or work.

Work done electrically may be measured in joules. This would indicate that energy and work have the same units of measure and can be calculated by the same equation. The equation used to determine the heat loss in an electrical circuit may read $H = W = I^2Rt \; (in\, Joules)$ where W indicates work.

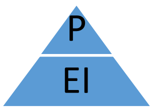
POWER. Power is the rate of doing work. As applied to any means of absorbing energy, we may say that power is the rate at which energy is absorbed or transformed.

In electrical measurements the unit of power is the Watt. If the above equation is divided by for time, we have

$$P = \frac{H}{t} = \frac{W}{t} = \frac{I^2Rt}{t} = I^2R$$

in which P is power. Therefore, the power absorbed by a conductor through which an electric current is flowing is equal to the product of the resistance of the conductor and the square of the current. If I is in amperes and R in Ohms, P will be in joules per second or watts.

Since from Ohm 's law, E =IR, in which E is the potential difference between terminals of the resistance R, E may be substituted for IR in the equation above, thus obtaining



Generalizing from this result, we may say that the power absorbed by any receiving circuit, whatever its nature, through which a steady current is flowing, is equal to the product of the current and the potential difference between the circuit terminals. Similarly, the power delivered to any circuit by an electric generator is equal to the product of the current it supplies and the potential difference between the generator terminals.

There is a definite relation between mechanical and electrical power. One horsepower (mechanical) equals 746 watts (electrical). This means that 550 ft-lbs per second is equal to 746 watts.

The watt is too small a unit to deal with in ordinary electrical calculations so a larger unit called the kilowatt, equal to 1000 watts, is used. The horsepower is equal to $\frac{746}{1000}$ watts or equals 0.746 kilowatts (kW).

Example: A direct current generator is supplying a power line with 250-volt power. A motor on the line is taking 42 amperes of current. What power is the generator supplying?

Solution: $P = EI = 250\,x\,42 = 10{,}500\,watts = 10.5\,KW$

Example: A pump motor is taking 25 amperes and is located 500 feet from the trolley wire. The leads to the pump have a resistance of 1 Ohm per 1000 feet. If the voltage at the trolley wire is 220, find (a) the voltage at the motor, (b) the power consumed by the motor, and (c) the power lost in the pump leads.

Solution: (a) Resistance in two pump leads 1.0 Ohms. Voltage drop between the trolley wire and pump = $25\,x\,1 = 25$ Volts. Voltage at pump motor = $220 - 25 = 195$ Volts.

(b) Power consumption by motor = EI = 195 x 25 = 4875 watts = 4.875 KW

(c) Power consumption at the trolley wire 220 x 25 = 5500 Watts = 5.5 KW

Power lost in pump leads: 5500 - 4875 = 625 Watts = 0.625 KW

Table MES 2-1 is a summary of the relationships between E, I, R, and P and the methods for their measurement. The formulas apply, as given to direct current circuits. Modifications of these formulas must be made for application to alternating current circuits.

Electrical term	Nonelectrical language	Unit	Symbol	Formula	Measure by	How meter is connected in circuit	Series Circuit	Parallel Circuit
Current	Rate of flow	Ampere	I	$I = \frac{E}{R}$	Ammeter	Series	$I_t = I_1 \; or \; I_2 \; or \; etc.$	$I_t = I_1 + I_2 + etc.$
Potential difference	Difference in energy level	Volt	E	E=IR	Voltmeter	Parallel	$E_t = E_1 + E_2 + etc.$	$E_t = E_1 \; or \; E_2 \; or \; etc.$
Resistance	Opposition	Ohm	R	$R = \frac{E}{I}$	Ammeter and voltmeter or ohmmeter	Series and parallel-series	$R_t = R_1 = R_2 + etc.$	$R_t = \frac{R_1 R_2}{R_1 + R_2}$
Power	Rate of work	Watt	P	$P = EI = \frac{E^2}{R} = I^2 R$	Wattmeter or ammeter and voltmeter	Series and parallel		

Table MES 2-1

Tabulation of Electrical Fundamentals

C. MINE ELECTRICAL FUNDAMENTALS QUESTIONS AND ANSWERS

MES-2 Q-1: One horsepower equals how many watts?
MES-2 A-1: 746

MES-2 Q-2: What is the unit of electrical pressure?
MES-2 A-2: Volt

MES-2 Q-3: What is the formula to find horsepower if the voltage and amperage is given?
MES-2 A-3: $hp = \frac{E \, x \, I}{746}$

MES-2 Q-4: If the voltage is 250 DC and the horsepower is 10, what is the amperes?
MES-2 A-4: $\frac{10 \, x \, 746}{250} = 29.84 \, Amps$

MES-2 Q-5: One horsepower equals how many foot pounds per minute?
MES-2 A-5: One horsepower equals 33,000 foot pounds of work in one minute.

MES-2 Q-6: A D.C. generator is supplying power at 250 volts. A motor operating from this line is taking 42 amperes. What power is supplied to the motor? Show the formula and work. Give answer in kilowatts.
MES-2 A-6: $P = E \, x \, 1 = 250 \, x \, 42 = 10{,}500 \, watts$

$$\frac{10{,}500}{1{,}000} = 10.5 \, kilowatts$$

MES-2 Q-7: If a fan is being driven by a 200 horsepower 2200 DC volt motor, through which there is flowing 50 amperes, what is the actual horsepower consumed?
MES-2 A-7: $hp = \frac{W}{746} - \frac{2200 \, x \, 50}{746} = 147.45$

MES-2 Q-8: What is the cost of operating the fan for 24 hours if the power consumed cost $00.05 per kilowatt hours?
MES-2 A-8: V x A = W or 2200 x 50 = 110,000 watts
Kw = W/1000 = 110 kw. 110 x .05 x 24 = $132.00 per day

MES-2 Q-9: A 150 hp mining machine has a rated voltage of 250 DC. What is the full load current on this machine?

MES-2 A-9: $150\ x\ 746\ =\ 111{,}900\ watts$

$\frac{111{,}900}{250}\ =\ 447.6\ amps$. Full load.

MES-2 Q-10: If the current of the DC motor, having a load of 50 hp. is 200 amperes, what will be the voltage of the load?

MES-2 A-10: $50\ x\ 746\ =\ 37{,}300\ watts$

$$\frac{37{,}300}{200}\ =\ 186.5\ volts$$

MES-2 Q-11: There are three resistors connected in parallel across a 120-volt circuit. The resistors are carrying currents as follows: A - 10 amperes, B - 15 amperes, C - 20 amperes. Find the voltage across the combination, the resistance of each part and the resistance of the combination.

MES-2 A-11: Voltage across the combination equals line voltage - 120 volts.

A = $\frac{120}{10}$ = 12 ohms, B = $\frac{120}{15}$ = 8 ohms; C = $\frac{120}{20}$ = 6 ohms.

Total current of combination = 10 + 15 + 20 = 45 amps. $\frac{120}{45}\ =\ 2-2/3$ ohms resistance of the combination.

MES-2 Q-12: If a 75 hp motor is rated at 500 volts DC, what is the full load current of this motor?

MES-2 A-12: $Current\ =\frac{hp\ x\ 746}{v} - \frac{75\ x\ 746}{500}\ =\ 111.9\ Amperes$

MES-2 Q-13: What would be the proper cable ampacity for this motor?

MES-2 A-13: The accepted rule is 125% of full load current or 111.9 x 1.25 = 139.185 or 140 amps.

MES-2 Q-14: If the voltage of a power system is 500, the DC power transmitted is 500 kw and the allowable voltage drop is 10%, what would be the number of amperes flowing in the circuit and what would be the number of volts lost?

MES-2 A-14 Watts = kw x 1,000 = 500 x 1,000 = 500,000

Amperes = $\frac{Watts}{v}\ = \frac{500{,}000}{500}$ = 1,000 amperes

10% x 500 = 50V DC lost

MINE ELECTRICIAN EXAMINATION

Here are some formulas that you may need in working the test problems.

Legend	
E – Volts	eff – Efficiency (%)
I – Amps	P.F. – Power Factor (%)
R-Ohms	W – Work
P – Power (Watts)	K – Resistance (per mil ft.)
L – Length (ft.)	D – Diameter (mils)
Ohm's Law	$E = I \, x \, R$
Power	$P = I \, x \, E$
Power	$P = I^2 \, x \, R$
Power	$P = \frac{V^2}{R}$
Amperes (3ϕ)	$I = \frac{746 \, x \, Horsepower}{1.732 \, x \, E \, x \, eff \, x \, P.F.}$
Amperes (1ϕ)	$I = \frac{746 \, x \, Horsepower}{E \, x \, eff \, x \, P.F.}$
Motors	
Synchronous	$RPM = \frac{Hertz \, x \, 120}{No.of \; Poles}$
Resistance	$R = \frac{K \, x \, L}{D^2}$
Work	$W = I^2 \, RT \; (joules)$
Electric Circuits	
Parallel	$E_t = E_1 = E_2 = E_3, etc.$
	$I_t = I_1 + I_2 + I_3 + etc.$
	$R_t = \frac{1}{Rt = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3} + etc.}$
Series	$E_t = E_1 = E_2 = E_3, etc.$
	$I_t = I_1 = I_2 = I_3 = etc.$
	$R_t = R_1 + R_2 + R_3 + etc.$
Horsepower	1 HP = 746 watts
	$1HP = \frac{550 \; ft\text{−}lbs}{second}$
	$1 \, HP = \frac{33{,}000 \; ft\text{−}lbs}{minute}$
	$1 \, HP = \frac{E \, x \, 1}{746}$

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 3

SECTION 3

FUNDAMENTALS OF AC SYSTEMS

A. AC SYSTEMS

Alternating current (AC) flow is constantly changing in amplitude, and it reverses its direction at regular intervals. An AC current produces a varying magnetic field around a conductor. This varying magnetic field will induce a voltage in another conductor placed nearby. This is the principle that enables a transformer to be utilized in AC circuits. A transformer gives us the advantage of transmitting power at high voltages with little line loss, and it allows us to efficiently change the voltage from one level to another when needed.

Three phase AC is more efficient than single phase AC in most transmission and power applications. Three phase motors are more rugged and dependable, and they present fewer starting problems than other types of motors. Also, as compared to single phase power, 1.732 times as much power can be transmitted in three phase with the same number of pounds of wire used.

1. REACTANCE IN AC CIRCUITS

In a DC circuit, resistance is the only property of a conductor that affects the flow of direct current. In an AC circuit, however, in addition to the resistance, there are two other properties of a conductor that affect the flow of current. These are inductance and capacitance. However, unlike resistance, they do not result in a consumption of energy.

2. INDUCTANCE AND INDUCTIVE REACTANCE

The inductance of a conductor causes the current in an AC circuit to lag behind the voltage. Thus, in a purely inductive circuit (a circuit with only inductance and no resistance), an alternating current lags behind the voltage by 90°. A coil in an electrical circuit will provide a high inductance. Inductance is denoted by L, and the unit of inductance is henry, H.

Inductive reactance is mathematically expressed:

$$X_L = 2\pi fL$$

Where X_L represents the inductive reactance in Ohms, f is the frequency of the alternating current in cycles per second (Hertz), and L is the inductance of the circuit in henrys.

3. CAPACITANCE AND CAPACITIVE REACTANCE

Capacitance is a property of a conductor that causes the current to lead the voltage in an AC circuit. In a purely capacitive circuit (a circuit with only capacitance and no resistance), and alternating current leads the voltage by 90°. The unit of capacitance is a farad.

The net effect of capacitance in an AC circuit, termed capacitive reactance, depends upon the frequency of the current. It is denoted by X_C. The value of capacitive reactance is expressed mathematically:

$$X_C = \frac{1}{2\pi fC}$$

where X_C represents the capacitive reactance in Ohms, f is frequency of the current in cycles per second, and C is the capacitance in farads.

4. IMPEDANCE

Impedance represents the combined effect of circuit resistance, inductance, and capacitance on the flow of current.

The series circuit consists of an inductance of 0.1 henry, a resistance of 6 Ω, and a capacitance of 100 microfarads which is subjected to 110 V at 60 cps. Calculate circuit impedance. Calculating the inductive reactance,

$$X_L - 2\pi fL = 2\,(3.14)\,60\,(0.1) = 37.68\ Ohms$$

Calculating the capacitance reactance

$$X_C = \frac{1}{2\pi fC} = \frac{1}{2\,(3.14)60\,(100\ x\ 10^{-6})} = 26.54\ Ohms.$$

Therefore, the impedance can now be calculated,

$$Z = \sqrt{R^2 + (X_L - X_C)^2} = \sqrt{(6^2) + (37.68 - 26.54)^2} = \sqrt{36 + 124} = \sqrt{160} = 12.65\ Ohms$$

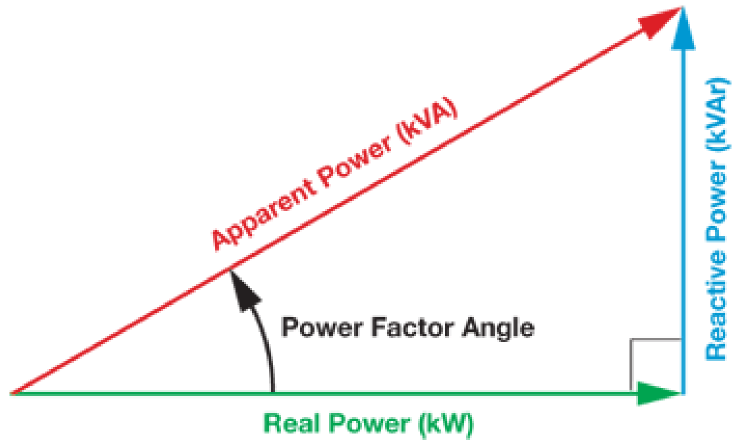
5. POWER, REACTIVE POWER AND POWER FACTOR

Since, in an AC circuit, the current is usually out of phase with the voltage by a phase difference which ranges between 0° and 90°, the power has three components : (1) apparent power--the power supplied by the generator, expressed in voltamperes, or kilovoltamperes (kVA); (2) useful or real power--the power used by the load for producing useful work, expressed in watts or kilowatts (kW); and (3) wattless or reactive power--the power utilized for overcoming the effects of circuit reactances, expressed in voltamperes reactive or kilovoltamperes reactive (kVAR).

The ΔABC is called a power triangle because the side AC represents the apparent power in volts amps (VA) or kilovoltamperes (kVA), AB represents the useful or real power in watts or kilowatts (kW), and BC represents the reactive power in voltamperes reactive (VAR) or kilovoltamperes reactive (kVAR). Since the power ΔABC is a right triangle:

$$apparent\ power\ = \sqrt{(useful\ power)^2 + (wattless\ power)^2}$$

Or $$kVA\ = \sqrt{(kW)^2\ +\ (kVAR)^2}$$



The horsepower of an AC motor is expressed thus:

$$Power\ Factor\ Angle\ = \sigma$$

$$Hp\ = \frac{useful\ power}{746\ eff}\ = VI\cos\sigma$$

6. POWER FACTOR

Only a part of the apparent power is useful power. The power factor when multiplied by apparent power gives the value of useful power. The power factor is expressed thus:

$$power\ factor\ = \frac{useful\ power}{apparent\ power}\ = \cos\sigma\ = \frac{AB}{AC}$$

When the power factor (PF) is one (1), the entire apparent power is utilized as useful power and the wattless power becomes zero (0). This happens when the circuit inductive reactance is equal to the capacitive reactance. If the inductive reactance is greater than the capacitive reactance, the current lags behind the voltage and the power factor is said to be lagging. If the capacitive reactance is greater than the inductive reactance, the current leads the voltage and the power factor is said to be leading. In the mine power systems, the usual power factor is lagging. Due to the inductive nature of the loads, therefore, shunt capacitors are added to produce leading power factor load. Shunt capacitors are used to improve the power factor close to unity.

7. Transformers

These are pieces of apparatus applicable to AC systems whereby a high voltage may be reduced or a low voltage increased. The transformer used to raise the voltage is termed a "step-up" transformer, while that used to lower the voltage is known as a "step-down" transformer.

Transformers operate on the principle that a current can be induced in a stationary conductor when it is cut by a magnetic field that is changing in strength and direction. A conductor carrying an alternating current is surrounded by a magnetic field constantly changing in strength and direction, and if another conductor is placed in this field, an alternating emf and current are induced in it. The conductor connected to the supply or input power circuit is termed the primary coil; the conductor connected to the output circuit is termed the secondary coil, and each coil surrounds the same iron core.

The primary coil sets up an alternating magnetic field with magnetic lines of force which are confined to the iron core because of the high permeability of the iron. The lines of the field travel around the core and cut the secondary coil, thereby inducing an alternating emf and current in the secondary coil.

It has been found that the induced emf in the secondary and the emf of the primary are in the direct ratio of the number of turns in the respective coils, while the currents are practically in the inverse ratio of the number of turns. Also, the power transformed to the secondary is equal, except for a small loss, to the power supplied at the primary.

Transformers are used in the mining industry at (a) power generating stations as step-up transformers to raise the generated voltage for transmission so as to reduce the size of cables and the line losses, (b) at surface substations as step-down transformers to lower the voltage to that required, (c) at underground substations to step-down "high" electrical pressure to medium or low pressures where necessary, and (d) as step-down transformers for lighting circuits.

In a transformer the following relation helps to identify various voltage and winding currents:

$$\frac{V_P}{V_S} = \frac{I_S}{I_P} = a = transformer\ turns\ ratio$$

where p represents the primary and s represents the secondary. By measuring the secondary voltage for a given primary voltage, the transformer turns ratio can be determined.

Example: Find the turns ratio and winding currents of a 150 KVA, $\frac{2400}{240V}$ single phase transformer at unity power factor.

$$Turns\ ratio\ = \frac{2400V}{240V}\ =\ 10$$
$$Primary\ Current, I_P = \frac{150{,}000VA}{2400V}\ =\ 62.5A$$
$$Secondary\ Current, I_S\ =\ (62.5\ x\ 10)\ =\ 625$$

8. POLYPHASE CONNECTIONS

Most transformers in and around mine installations are of the three-phase or polyphase type. Polyphase transformers are more compact and require less space than a bank of three single-phase transformers. The connections are usually delta, wye, or a combination of these. Mine power transformers are practically always connected in the delta Δ configuration on the primary side and in the wye configuration on the secondary side.

A delta connection is a parallel arrangement of the windings. When transformers are connected in the delta fashion, the line current will be 1.732 times the phase-winding current and the line voltage will be the same as that of the phase windings. The delta connection used on the primary side of a three-phase transformer allows the three high-voltage lines to be brought into the transformer to obtain the maximum line voltage.

The secondary coil connections in the wye configuration permit the use of a neutral for ground fault relaying. When the secondary windings are wye-connected, the line voltage will be 1.732 times the phase-winding voltage and the line current will be the same as the phase-winding current.

Example: Study voltage and currents in various sections of a delta-wye transformer. Primary line-to-line voltage is 7200 and line current is 80A. Step down ratio is 20:1.

$$Primary\ line-to-line\ =\ phase\ voltage\ =\ 7200V$$
$$Secondary\ phase\ voltage\ = \frac{7200V}{20}\ =\ 360V$$
$$Secondary\ line-to-line\ voltage\ =\ 360\ x\ 1.732\ =\ 623.5V$$
$$Primary\ line\ current\ =\ 80A\ (given)$$
$$Primary\ phase\ current\ = \frac{80A}{1.732}\ =\ 46.2A$$
$$Secondary\ phase\ current\ =\ (46.2A\ x\ 20)\ =\ 923.8A$$
$$Secondary\ line\ current\ =\ 923.8A$$

Summarizing in Delta Connection:

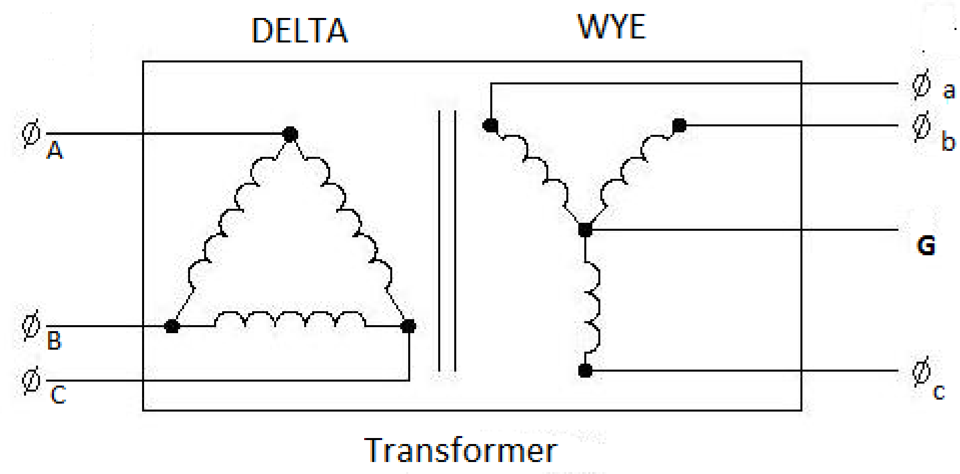
$$Phase\ voltage\ =\ line - to - line\ voltage.$$

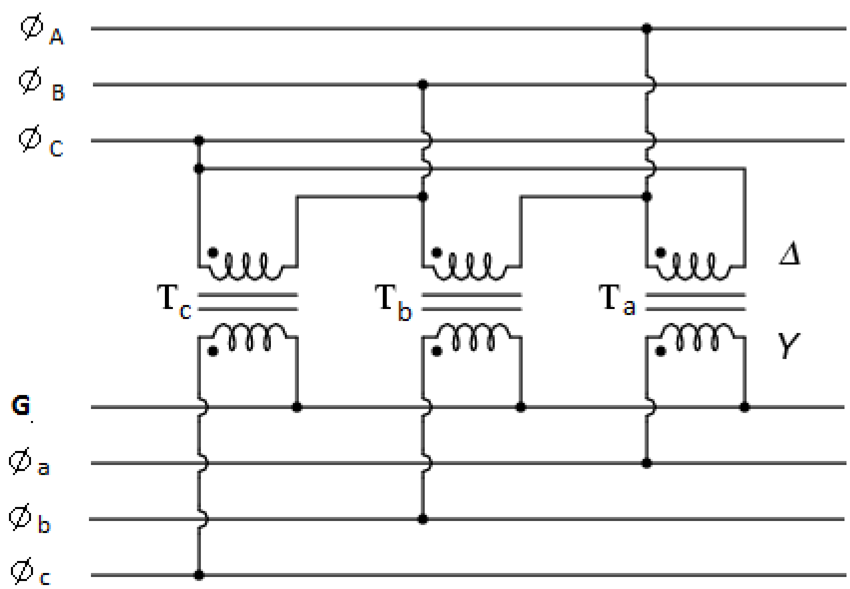
$$Phase\ current\ =\frac{line\ current}{1.732}$$

Summarizing in Wye-Connected Transformers:

$$Line - to - line\ =\ (phase\ voltage)\ x\ 1.732$$

$$Phase\ Current\ =\ line\ current$$





B. QUESTIONS AND ANSWERS
AC SYSTEMS

MES-3 Q-1: What is AC?

MES-3 A-1: Alternating current flow is constantly changing in amplitude, and it reverses its direction at regular intervals. An AC current produces a varying magnetic field around a conductor. This varying magnetic field will induce a voltage in another conductor placed nearby. This is the principle that enables a transformer to be utilized in AC circuits.

MES-3 Q-2: What are the advantages of a transformer?

MES-3 A-2: A transformer give us the advantage of transmitting power at high voltages with little line loss, and it allows us to efficiently change the voltage from one level to another when needed.

MES-3 Q-3: What is the function of a transformer?

MES-3 A-3: These are pieces of apparatus applicable to a-c systems whereby a high voltage may be reduced or a low voltage increased. The transformer used to raise the voltage is termed a "step-up" transformer, while that used to lower the voltage is known as a "step-down" transformer.

MES-3 Q-4: In a perfect transformer, what is the relation between the primary and secondary currents?

MES-3 A-4: The primary coil sets up an alternating magnetic field with magnetic lines of force which are confined to the iron core because of the high permeability of the iron. The lines of the field travel around the core and cut the secondary coil, thereby inducing an alternating emf and current in the secondary coil.

It has been found that the induced emf in the secondary and the emf of the primary are in the direct ratio of the number of turns in the respective coils, while the currents are practically in the inverse ratio of the number of turns. Also, the power transformed to the secondary is equal, except for a small loss, to the power supplied at the primary.

MES-3 Q-5: Name some applications of step-up transformers?

MES-3 A-5: Transformers are used in the mining industry at power-generating stations as step-up transformers to raise the generated voltage for transmission so as to reduce the size of cables and the line losses.

MES-3 Q-6:	Name some applications of step-down transformers?
MES-3 A-6:	Transformers are used in the mining industry at surface substations as step-down transformers to lower the voltage to that required, at underground substations to step-down "high" electrical pressure to medium or low pressures where necessary and as step-down transformers for lighting circuits.

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 4

SECTION 4
DC SYSTEMS AND CONVERSION EQUIPMENT
A. D.C. SYSTEM

Direct current (DC) is defined as the electrical current flowing in one direction. There are three sources of DC power in underground mines. They are:

a) Battery
b) Motor-generator sets
c) Rectified DC from AC.

DC power is obtained from the battery through chemical reactions. Of the two remaining sources, AC power is converted into DC and is described under the heading Conversion Equipment. DC power is used in underground mines in the following applications. Some of which are:

a) Trolley locomotive
b) Battery charging
c) For driving DC scoop motors
d) For driving the tram motors of some mining equipment
e) Welding applications

In general, DC motors are suitable for variable speed applications and high torque requirements. The disadvantage of DC is that it cannot be transmitted over longer distances without appreciable voltage drop.

1. BATTERY

A "storage battery" can be defined as a battery in which the electrochemical action is reversible. That is, after an output of electrical current ("discharge"), the battery can be returned to the original state ("recharged") by passing current through it in the opposite direction. The basic unit of the battery is the "cell", simply consisting of positive plates, negative plates, and electrolyte. One or more cells are connected together to form the battery, and the connection is usually in series (parallel and series-parallel combinations are also used). The battery voltage, often given as an open-circuit value, is the sum of the series cell voltages. The capacity, commonly expressed as ampere-hours (Ah) or kilowatt hours (kWh), is mainly dependent upon the plate size (surface area and voltage).

Two types of storage batteries have been employed in underground traction applications. The "nickel-iron" or "Edison cell" is an "alkaline-type" cell because of the electrolyte used. The plates for this battery are constructed of nickel oxide and iron, immersed in an electrolyte of potassium hydroxide and lithium hydroxide. Having high-reliability and minimum-maintenance characteristics, Edison batteries were once popular in the mining industry. However, lead-acid batteries have replaced the Edison type as a result of their high-energy per unit volume and high-power capability. The basic lead-acid cell utilizes a lead peroxide (PbO_2) positive plate and a sponge lead (Pb) negative plate. These plates are suspended in a solution of diluted Sulfuric Acid (H_2SO). Lead-acid batteries are commonly used in face haulage vehicles.

A power rectifier consists of alternating current switchgear, rectifier transformer, rectifier, and direct current switchgear. Experience with rectifier equipment has shown that they are more efficient than other conversion equipment over the entire range of load, are easily moved, and require very little maintenance because there are no moving parts.

It should be noted that where alternating current mining equipment is used underground, conversion equipment is not required, and the establishment of a transformer station of sufficient capacity will suffice. Usually the overall efficiency of the transformer-rectifier set is very high. They are also less costly, and require little maintenance for the most part.

2. DC SUPPLY TO RAIL HAULAGE

The negative side of most direct current power systems is known as the return or the grounded side. It is an important feature of an electrical system to maintain good returns. All of the aforementioned practices of keeping the conversion unit as near the load center as possible and providing ample size copper in the positive side would be useless without an adequate return.

Theoretically, the negative side of a mine power system should be equal in size to the positive line. The main objective in providing returns is to secure the least resistant path to the negative side of the conversion unit. Usually the return conductors are of sufficient size (especially where the track is used as a return) to maintain a low voltage drop. However, it is the resistance of the terminal contacts that increases the total resistance above the desired value. Thus, frequent terminal resistance tests as well as tests at the time of installation, should be made to determine the condition of the return path.

3. RETURN CONDUCTORS

Where track haulage is used, the rails provide the return path. The rails, not being continuous throughout the mine must be welded or bonded at the joints. In addition to welding and bonding of one or both rails, electric conductors may be laid parallel to the rail. An example of this practice is to tie a 500 kcmil auxiliary return cable into two 60 pound bonded and cross bonded rails at frequent intervals on the main lines and 4/0 wire laid parallel to 40 pound rail in the panels.

With conveyor haulage, adequate return capacity is secured by using the same size of return conductor as the feeder. However, the practices vary here as well as in the feeder system from a one line return to many parallel return lines. An example of this latter practice would be the use of three 4/0 lines; one for both pieces of equipment and the other for a general neutralizing conductor. Regardless of the method used, every possible means should be employed to insure a continuous and low resistance return path.

Underground metallic pipe lines or ground circuits paralleling haulage roads or conveyor lines shall be electrically continuous throughout and shall be electrically connected to the track or conveyors at frequent intervals. This precaution reduces stray currents, removes the possibility of a difference of potential between a grounded circuit and the pipe line, prevents the pipe line from becoming charged, prevents corrosion of the pipe line from electrolytic actions, and at the same time makes the pipe line a part of the return system.

4. BONDING

In considering the terminal contact in the resistance value of a return circuit, the welding or bonding at all rail joints must be electrically efficient to reduce the loss to a minimum. The improper welding and maintenance of rail joints or bond terminals increases the resistance and transforms electrical energy into heat. The cost of these electrical losses will often times more than pay for the labor required to provide good installation and maintenance of the return path. Where the electrical load is heavy and the track is used for the return, cross bonding is desirable. Cross bonding provides insurance against a broken return current path and gives some assurance of a continuous system. A continuous path for the current is necessary at all times from the standpoint of safety as well as maximum efficiency. Inefficient bonding and broken rails will dissipate large quantities of heat with the flow of current. The overheating at these points may cause ignition of combustible material in contact with them. Furthermore, increased resistance increases the voltage drop in the circuit, and low operating voltages induce heating in motor windings and trailing cables.

B. TROLLEY WIRES

1. POSITIVE SIDE

It is general practice in all direct current powered mines to use the trolley line paralleled by a feeder line to transmit the energy toward the working faces. These conductors are usually bare wires, although the feeder line may have an insulated covering broken only where it connects the trolley wire and the sectionalizing switches. They shall be installed on the side opposite the clearance side and shelter holes.

The trolley and bare feeder wires shall be supported securely on insulated hangers. All trolley wires and feeder lines installed on underground haulage roads shall be placed as far to one side of the passageway as is practicable, but not less than six inches outside of line of rail, and securely supported upon hangers which shall not be more than twenty-four feet apart and efficiently insulated. It is further suggested that ample support be given by considering the tensional strength and weight of the wire to be supported and the possible restricting of the sag to 3 inches from the horizontal. The hangers on curves should be so spaced that the trolley wire at any one hanger may be disconnected completely without endangering the locomotive operator or make contact with the roof, rib, cross bars, or door frames.

Trolley wires shall be hung not less than 6 inches outside the rail and kept as near this distance as possible. The height above the rail should be as uniform as possible. All hangers, switches, splicers, frogs, and other connections should be of such a type as to eliminate any jumping of the trolley. This method of hanging wire will reduce the tendency of trolley poles of locomotives to jump off the wire and thus reduce the hazards incident thereto.

All branch trolley lines shall be fitted with either a trolley switch, circuit breaker, or section insulator and line switch, or some other device that will allow the current to be shut off from such branch headings. Switches or circuit breakers shall be provided on haulage roads to de-energize all trolley and feeder lines at intervals not to exceed two thousand feet. It is good practice to mark the positions of these switches with suitable signs so they may be found easily in emergencies. Officials, haulage crews, electricians, and others should be familiar with the location of these cut-out switches.

Each branch trolley circuit should be provided with a frog where it leaves the main line. The branch sectionalizing switch or breaker should be near the frog. A great deal of care should be given to the installation of the frog, for a poorly installed frog is both uneconomical and hazardous. A definite method should be employed to determine the exact location of every trolley frog according to the frog's turn-out angle, the location of the track, and the type of locomotive in use.

With the location established, installing the frog in the wire is a simple matter. For best results, it is recommended that the straight-through wire be strung uncut and under tension before the frog is placed in the wire. First, place the frog in position on top of the straight-through wire and mark its length on the wire with a file or hacksaw blade. Next, take the tension off the marked portion using wire eccentrics and a block and tackle. Cut the marked section out of the wire and install a tip at each of the open trolley wire ends. Then bolt the tips to the frog body, release the tension, and the frog is in place on the straight-through wire. Turnout wires are handled in the same manner except that they may be attached to the frog before the wire is tensioned.

The ends of all trolley and other bare wires should be securely anchored and properly insulated. It is good practice to have the last 5 or 6 feet of a trolley wire and the insulating turnbuckle protected by guard boards, because at this point trailing cables are connected and the erection of guards will reduce the shock hazard. The ends of trolley or feed wires shall not extend beyond the last open crosscut and shall be kept at least 150 feet from pillar workings, advancing workings, or any open, partly caved places. They are also installed in intake air in all mines.

The size of trolley wires should not be less than 2/0 (American Wire Gauge) and 4/0 is preferable. In many mines trolley wire of 350 kcmil area are installed on main haulageways, and in a few mines they are as large as "No. 9 deep section grooved."

Where feeder cable parallels a trolley wire, the cable size is restricted by two factors: (a) the allowable voltage drop, and (b) the allowable temperature rise. With these factors in mind, the allowable current-carrying capacity in amperes of a 350 kcmil trolley wire and 1,000 kcmil cable is approximately 1500 Amps. Where the cable size is quite large because of the high current to be transmitted, a number of other points are worthy of consideration. They are (1) the amount of ventilation provided over the cable, (2) the number and type of connections to be made, (3) the maximum degree of curvature the cable will make, etc.

C. CONVERSION EQUIPMENT

The use of conversion equipment in mine service was much more common when primary distribution and utilization was de. Today, since most underground mine electrical service is distributed and utilized as ac, its use is much more limited. Conversion of AC to DC is normally limited to use for transportation systems, welding services, battery charging, and to certain electric motors in some mining equipment. Motor generators, rotary convertors, and rectifiers are all found as conversion equipment in underground coal mines.

In selecting the proper conversion equipment to supply a specific mining requirement, there are four basic decisions to be made: (1) where the conversion equipment will be located, (2) the type of equipment best suited to the requirements, (3) the rating in kilowatts and voltage of each unit, and (4) the number of conversion units required.

In making each of these decisions, five factors must be considered: (1) the operating characteristics of the conversion equipment; (2) the suitability of the equipment for the environment; (3) the cost of the equipment and its installation; (4)the operating cost; efficiency, power factor, maintenance; and (5) the ease of operation, installation, and maintenance.

1. MOTOR GENERATOR SET

Motor-generator sets represent the oldest conversion equipment still found in mine service. A set generally consists of a DC shunt or compound generator driven by an AC synchronous motor. Both motor and generator are independent machines connected by flexible couplings and mounted on a common bedplate. For underground applications, the motor-generator set can be mounted on a rail car. An additional car is required for the associated controls and switchgear. This arrangement facilitates moving the equipment for ever-changing underground operations.

Smaller size units will use an AC squirrel cage as the prime mover because of its simplicity, and rugged construction. The efficiency of the set depends on the loading conditions.

The motor-generator set requires maintenance on bearings, lubrication, commutator, slip rings and brushes, winding insulation level, and set alignment. Some of these maintenance operations require periodic equipment shutdown.

2. RECTIFIERS

Rectifiers are devices that change AC power into DC power with the help of diodes. Diodes are devices that allow current flow in one direction only under normal conditions. These are used in single phase or three-phase rectifiers.

D. DC SYSTEMS
QUESTIONS AND ANSWERS

MES-4 Q-1: Define DC current?
MES-4 A-1: Direct current (DC) is defined as the electrical current flowing in one direction.

MES-4 Q-2: Give some examples of DC power sources.
MES-4 A-2: There are three sources of DC power in underground mines. They are:
(a) Battery
(b) Motor-generator sets
(c) Rectified DC from AC

MES-4 Q-3: Are there any DC power applications in underground mines?
MES-4 A-3: Yes

MES-4 Q-4: Give some examples of DC power applications in underground mines.
MES-4 A-4: DC power is used in the underground mines in the following applications. They are:
(a) Trolley locomotives
(b) Battery charging
(c) For driving DC motors in some mining equipment
(d) Welding applications

MES-4 Q-5: What is a battery?
MES-4 A-5: A "storage battery" can be defined as a battery in which the electrochemical action is reversible. That is, after an output of electrical current ("discharge"), the battery can be returned to the original state "(recharged") by passing current through it in the opposite direction. The basic unit of the battery is the "cell", simply consisting of positive plates, negative plates, and electrolyte. One or more cells are connected together to form the battery, and the connection is usually in series (parallel and series parallel combinations are also used). The battery voltage, often given as an open-circuit value, is the sum of the series cell voltages. The capacity, commonly expressed as ampere-hours (Ah) or kilowatt hours (kWh), is mainly dependent upon the plate size (surface area and voltage).

MES-4 Q-6: What are two types of storage batteries used in the mines?
MES-4 A-6: Two types of storage batteries have been employed in underground traction applications. The "nickel-iron" or "Edison cell" is an "alkaline-

type" cell because of the electrolyte used. The plates for this battery are constructed of nickel oxide and iron, immersed in an electrolyte of potassium hydroxide and lithium hydroxide. Having high-reliability and minimum-maintenance characteristics, Edison batteries were once popular in the mining industry. However, lead-acid batteries have replaced the Edison type as a result of their high energy per unit volume and high-power capability. The basic lead-acid cell utilizes a lead peroxide (PbO_2) positive plate and a sponge lead (Pb) negative plate. These plates are suspended in a solution of dilute Sulfuric acid (H_2SO). Lead-acid batteries are commonly used in the face haulage vehicles.

MES-4 Q-7: In selecting the proper conversion equipment to supply a specific mining requirement, what are the decisions to be made?

MES-4 A-7: In selecting the proper conversion equipment to supply a specific mining requirement, there are four basic decisions to be made : (1) where the conversion equipment will be located, (2) the type of equipment best suited to the requirements, (3) the rating in kilowatts and voltage of each unit, and (4)the number of conversion units required.

MES-4 Q-8: Before making each decision, what factors must be considered?

MES-4 A-8: In making each of these decisions, five factors must be considered: (1) the operating characteristics of the conversion equipment; (2) the suitability of the equipment for the environment; (3) the cost of the equipment and its installation; (4)the operating cost; efficiency, power factor, maintenance; and (5) the ease of operation, installation, and maintenance.

MES-4 Q-9: What are the advantages of power rectifiers?

MES-4 A-9: Rectifier equipment is more efficient than other conversion equipment over the entire range of load, are easily moved, and require very little maintenance because there are no moving parts.

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 5

SECTION 5
A. MINE POWER CABLES

Most of the mine equipment power is supplied through cables. Cable dependability is determined by: (1) the type selected, (2) the installation, (3) the use, and (4) maintenance. Sensible conservative practices will add to mine safety, the life and service of cables and to mine profits. The components in the system are connected together by various cables that must be insulated against different voltage levels. For example, high-voltage cable of 4160 V, 7200 V, or 12470 V will connect components between the surface substation and the section load center. The power must be supplied efficiently and safely to the mobile production equipment operating at the face or between the load center and the face. The machinery requires the extension of power by means of a flexible and portable type of cable.

Every machine operated from an external source of power must have a portable cable of adequate length and current-carrying capacity. The cable must have an outer sheath of rubber or equivalent material that will be highly resistant to abrasion, moisture, and flame. Since the cable must contain the frame ground conductors, and often a monitoring conductor, and the main conductors, they all must be separated by adequate insulation material capable of withstanding the flexing required by the mobility of the equipment.

The increased number of portable cables in service and the magnitude of the current capacity of individual cables has made it necessary for the governmental regulatory agencies to establish and enforce more rigid restrictions in connection with cables carrying electrical power to mobile machines.

In the mine, it is important that all controller casings, motor enclosures, etc., be so sealed to prevent electric arcs or the flames of explosions within the enclosures of electric elements from reaching a possible explosive atmosphere surrounding the machine. In order to maintain such an explosion-proof condition, the clearance between the cable and the sides of the opening leading into an enclosure must be small enough to help restrict the arc or flame from reaching the outside of the casing. Since mining conditions result in short lives for portable cables, the number of replacements is rather high.

The current demand of an electrical mining machine is fixed by the power requirements of the motors installed in it and the voltage of the power circuit. Portable cables conducting the current flowing to and from the machine must be capable of carrying this load safely. The size and number of conductors within a cable, the arrangement of the conductors, and the type and amount of insulation used all govern the current-carrying capacity of the cable.

Portable cables have a shorter life than those installed in fixed positions. They receive much greater mechanical abuse and are often overheated due to the various electrical conditions under which they must supply the load. If these two principal causes for rapid deterioration of portable cables are not carefully considered, production by the mining machines will fall far short of expectancy, and the cost of cable repairs and replacement will be rather high.

Mobile machines tramming at speeds in excess of 2.5 mph must be equipped with suitable mechanically, electrically, or hydraulically driven reels on which to wind portable cable. A cable placed on a reel or coiled into several layers and then required to carry its normal current capacity or greater is subjected to a heating condition. The heat developed cannot be dissipated fast enough to prevent its causing breakdown of the insulation. In order to maintain reasonably safe and continuous operation of a machine equipped with a cable-reel mechanism, the cable should have a higher current-carrying capacity than would normally be required.

It should be realized that the current rating of a cable cannot be the sole factor to consider in selecting a cable. Physical properties such as weight, overall dimensions, flexibility, tensional strength and insulation must also be considered. The standard of color coding of distribution power cables for polarity identification for both AC and DC power are as follows:

1. Multiple conductor power cables for DC distribution service.
   Black – positive
   White – negative
   Green - safety

   The negative conductor shall always be considered the grounded conductor.

2. Multiple conductor power cables for single phase low voltage AC distribution service.
   Black - phase wire
   White – neutral
   Red - phase wire
   Green - safety ground

3. Multiple conductor power cables for three phase low voltage AC distribution service.
   Black - phase wire
   White - phase wire
   Red - phase wire
   Green - safety ground

1. VOLTAGE RANGES

The Bituminous Coal Mining Law of Pennsylvania define "low voltage" supply as the conditions of the supply of electricity such that the difference of potential between any two points in the circuit cannot exceed 660 volts. "Medium voltage" supply ranges from 661 to 1000 volts. "High voltage" supply includes voltage higher than 1000 volts (Art. III, Sec. 302).

2. PORTABLE CABLES

Portable machine cables, unless stated otherwise, are those subjected to constant flexing and reeling while they connect the load centers to shuttle cars and other mobile mining equipment such as continuous miner, cutting and loading machines. These cables are for use in circuits not exceeding l000V but insulated for up to 2000V, a 2:1 safety ratio, with a maximum conductor temperature of 20°C.

Flat twin type W cable is used where grounding of equipment is not required. It consists of two insulated conductors assembled parallel with a neoprene jacket compound between the conductors as an integral part of the overall neoprene jacket.

Flat twin type G cable consists of two insulated conductors assembled parallel with a flat grounding conductor between them with neoprene compound between each insulated conductor and grounding conductor as an integral part of the overall jacket.

Three-conductor flat type G cable consists of three insulated conductors with one flat ground wire between the middle and each outer conductor and with reinforcing braid over the assembly and a vulcanized neoprene jacket over the braid.

Three-conductor flat type G-GC cable has three insulated conductors assembled in parallel with a flat grounding conductor covered with a separator between the white and red power conductors and the ground check conductor between the black and white power conductor with a reinforcing braid over the assembly and then vulcanized to a neoprene jacket.

Three-conductor round type G-GC cable has three insulated power conductors connected together, with the ground check conductor placed in the valley (interstices) between the black and white conductors with one grounding conductor in each of the other two valleys. The two braid- covered grounding conductors are uninsulated, and the ground check conductor is covered with yellow insulation. The entire assembly is covered with a reinforcing neoprene jacket.

Three-conductor, EPR-insulation, type SHD-GC cable is used with longwall miners and for other heavy duty use where the individual insulated phase conductors are required to be shielded, and grounding conductors and a ground-check conductor are required. Three insulated and shielded conductors are cabled together with suitable fillers and with the insulated and yellow-braid-covered ground-check conductor in the valley between the black-and-white conductors and one of the uninsulated grounding conductors in each of the other two valleys. A cable tape is applied over the assembly which is then encased in an overall, two-layer, reinforced thermosetting jacket.

In addition to two-conductor DC cables and three-conductor AC cables, portable cables are made both round and flat and in a variety of configurations of ground conductors and ground-check conductors. These reflect an attempt to provide a compact arrangement of the conductors in a single jacket and also to eliminate induced differences in potential in the ground conductors.

3. CABLE SELECTION

Cables are selected on the basis of: (1) load; (2) their conductor-operating temperature; and (3) voltage drop caused by current flowing through them.

The selection of voltage levels is probably the most important decision made in designing a mine power system. Because higher voltages, in most cases, mean lower costs, greater flexibility, and greater margin for expansion, higher voltages tend to be selected both for the primary transmission system and for portable cables.

Since a lower price for cable results from the fact that a higher voltage permits the use of smaller conductors to supply the same load at the same distance, primary voltages have gone from 4160 to 7200 to 12470V. The cost of the equipment for a 7200V system will only be slightly more than for 4160V, but it can supply the same load at three times the distance of a 4160V system using the same size cable.

What has been said for the high-voltage cable holds equally for the portable cable, perhaps even more so. The increasing horsepower requirements of continuous miners call for such a large portable cable that it becomes difficult to handle. The solution is to increase the utilization voltage that the portable cables must handle. Thus 440V to 480V has given way to 550V to 600V and, more recently, to 950V.

The following high-voltage installation practices are suggested guidelines although all local laws should be adhered to:

1) High voltage cables shall be installed only in intake airways. They may be installed on intake haulage-ways only with written approval of the department. Such cable may be installed by hanging on suitable hooks or clamps, or by supporting by a suitable messenger cable, or by burying or by installation in metal conduit. When suspended, distance between

supports shall not exceed twenty feet and they shall be so placed that they do not damage the cable jacket. When hung in haulage entry containing a trolley wire, the cable shall be installed at least twelve inches from the trolley wire or feeder wires and away from the track.

2) Any excess cable which is connected and supplying a load shall be coiled, stored on a reel, or otherwise stored, at a place near the load where it can be protected by dangering off the place. Such cable shall not exceed one thousand feet in length.

Cables will be sectionalized to convenient lengths which are fitted with appropriate couplers or splice boxes. These must be carefully installed and must be kept clean to prevent maintenance problems, parallel ground paths.

Cable life may be increased and cable costs reduced by observing the following rules:

1. Excess cable, whether it is high voltage or low voltage, should be stored in a large figure 8 in a safe place so that maximum ventilation air flows over it.

2. Cables should not be buried under timbers, slate, coal, or other minerals, or in any manner covered up to prevent the free passage of ventilation air over them.

3. Energized high voltage transmission cable shall not be handled.

4. Cables should not be punctured with nails or other similar sharp objects, and every effort should be made to keep them from resting on sharp rocks or other things that may pierce their jackets.

5. Even though cables are of waterproof construction, every effort should be made to keep them away from water holes and water dripping from the roof.

6. Cables should not be kinked.

7. Cables should not be twisted.

8. Cables should not be tied in knots or bent sharply.

9. Cables must not be run over by rubber-tired vehicles, or any other type of vehicle. In fact, an effort should be made to avoid even walking on cables.

10. Cables should not be pulled by a machine.

11. Cables should be examined frequently (each trailing cable in use shall be examined every shift by the machine operator) and repaired immediately when any damage or fault is found.

12. All repairs should be vulcanized or, in some equally effective manner, sealed mechanically and electrically. Never use friction tape to replace factory insulation in splices. Always use a water-resistant, rubber insulating tape.

13. When cables are installed on reels, the level wind should be arranged so as to lay the first turns in the place they are to occupy and all subsequent turns should be laid on in an easy, natural manner. The level wind should not fight the cable.

14. Slack cable should be reeled up carefully to avoid snapping or stretching it, or its slipping.

15. Cable reel torque settings should be based on manufacturers ' recommendations.

16. Shuttle-car cable anchor snubs should be fitted with springs to reduce the effects of jerks on the cable.

17. Cables should be anchored by suitable clamps around their jackets. Anchoring one conductor only causes the cable to twist.

18. Where possible, all cables should be sectionalized in short lengths.

B. MINE POWER CABLES
QUESTIONS AND ANSWERS

MES-5 Q-1: Define the three voltage ranges.

MES-5 A-1: The BMS has defined three voltage levels: low voltage is up to and including 300V, medium voltage is 661 to 1000V, and high voltage is in excess of 1000V.

MES-5 Q-2: What are mine cables used for?

MES-5 A-2: Most of the mine equipment power is supplied through cables.

MES-5 Q-3: What are the differences between fixed cables and portable cables?

MES-5 A-3: Portable machine cables, unless stated otherwise, are those subjected to constant flexing and reeling while they connect the load centers to shuttle cars and other mobile mining equipment such as continuous miners, cutting and loading machines. These cables are for use in circuits not exceeding l000V but insulated for up to 2000V, a 2:1 safety ratio, with a maximum conductor temperature of 20°C.

Portable cables have a shorter life than those installed in fixed positions. They receive much greater mechanical abuse and are often overheated due to the various electrical conditions under which they must supply the load. If these two principal causes for rapid deterioration of portable cables are not carefully considered, production by the machines will fall far short of expectancy, and the cost of cable repairs and replacement will be rather high.

MES-5 Q-4: How should you select cables?

MES-5 A-4: Cables are selected on the basis of: (1) load; (2) their conductor-operating temperature; and (3) voltage drop caused by current flowing through them.

MES-5 Q-5: How should you install high-voltage cable?

MES-5 A-5: The following high-voltage installation practices are suggested guidelines although all local laws should be adhered to:

(1) High voltage cables shall be installed only in intake airways. They may be installed on intake haulage-ways only with written approval of the secretary. Such cable may be installed by hanging on suitable hooks or clamps, or by supporting by a suitable messenger cable, or by burying or by installation in metal conduit. When suspended, distance between supports shall not exceed twenty feet and they shall be so placed that they do not damage the cable jacket. When hung in haulage entry containing a trolley wire, the cable shall be installed at least twelve inches from the trolley wire or feeder wires and away from the track.

(2) Any excess cable which is connected and supplying a load shall be coiled, stored on a reel, or otherwise stored, at a place near the load where it can be protected by dangering off the place. Such cable shall not exceed one thousand feet in length.

MES-5 Q-6: What are the standard of color coding of distribution power cables for polarity identification for both AC and DC power?

MES-5 A-6: The standards are as follows:

1. Multiple conductor power cables for DC distribution service.
   Black – positive
   White – negative
   Green - safety

The negative conductor shall always be considered the grounded conductor.

2. Multiple conductor power cables for single phase low voltage AC distribution service.
   Black - phase wire
   White - neutral
   Red - phase wire
   Green - safety ground

3. Multiple conductor power cables for three phase low voltage AC distribution service.
   Black - phase wire
   White - phase wire
   Red - phase wire
   Green - safety ground

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 6

SECTION 6
MINE ELECTRIC MOTORS AND STARTERS
A. ELECTRIC MOTORS

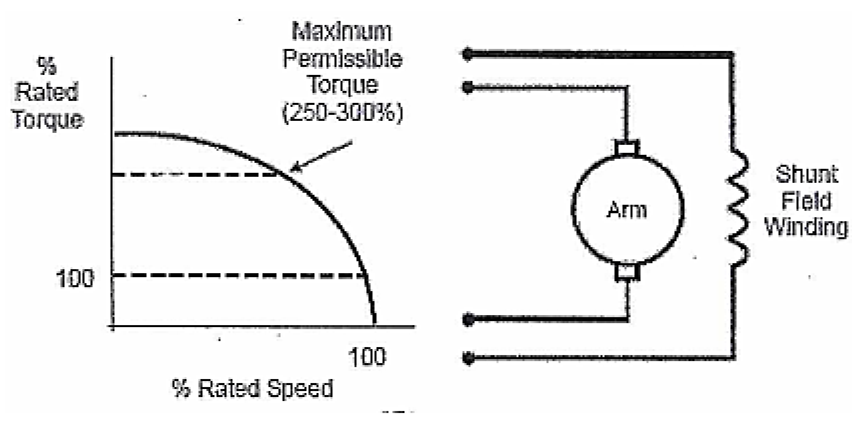
Electrical motors are preeminent as the means for converting electrical energy to mechanical energy. This is accomplished by utilizing the magnetic principles of attraction and repulsion to cause an interaction between a magnetically charged stationary winding called a stator, and another magnetized rotating element called a rotor. The basic differences among motors are the methods employed to produce proper relative polarity between the rotors and stators. For the use of mine electrical power systems, there are two different types of motors:

A) DC motors
   1) Shunt motor
   2) Series motor
   3) Compound motor

B) AC induction motors
   1) squirrel-cage motor
   2) wound-rotor motor
   3) Synchronous motor

C) Motor Starters

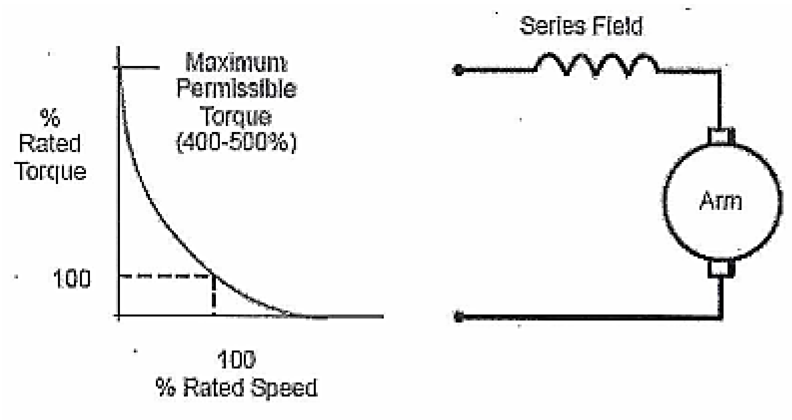
1. DC SHUNT MOTOR



In shunt motors, as the torque increases, the speed drops, but the motor never loses much of its speed. Normally, the speed has no more than 10% loss under full-load conditions.

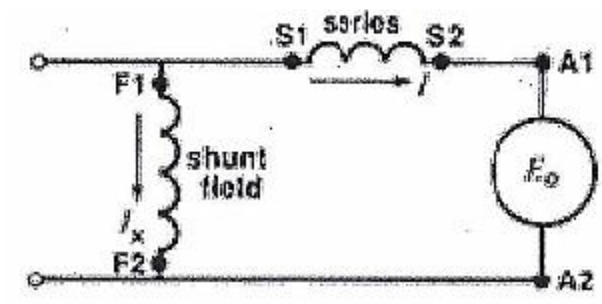
Shunt motors are used where a substantially constant speed is required, for example, as a conveyor, pump, and blower drive. They are better adapted than any other type of motor to speed control and speed adjustment. They are not very popular in the mining industry.

2. DC SERIES MOTOR



The speed of series motor varies over a wide range with the change of torque. The motors provide a large starting torque from standstill, so these types of motors are often used in traction application and scoops.

3. DC COMPOUND MOTOR



In a compound motor both the shunt and series fields are provided. There are two types of compound motors.

a) Normally the shunt and series field coils are connected causing the flux produced by the series field to add to the flux produced by the shunt field. A motor with such a field windings is known as a cumulative compound motor.

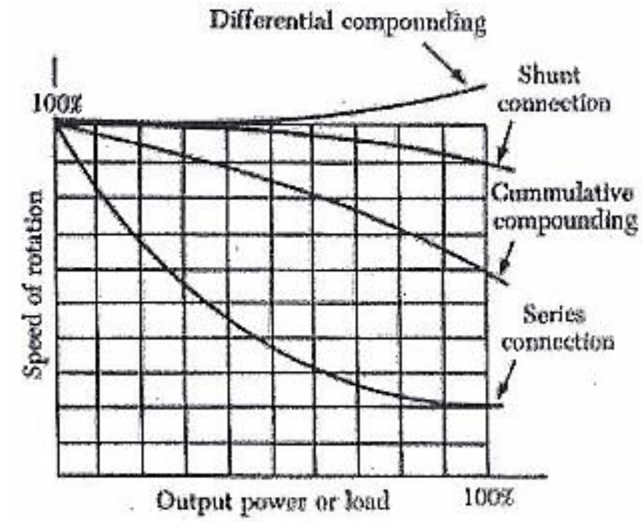
b) Under certain conditions, it may be advantageous to have the series field oppose the shunt field for the purpose of regulating the motor speed under full load conditions. Motors of this type are known as differential compound motors. There is little demand for this type of motor around coal mines.

This type of motor is used where the characteristics of either the series or shunt motor are not suitable for the work to be performed.

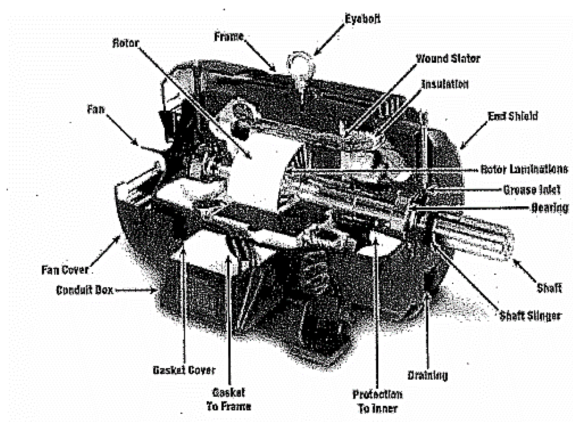
AC induction motors available for mining applications are basically three-phase motors with the stator usually connected in wye or delta. They are:

a) squirrel-cage motor, and

b) wound-rotor motor



4. AC SQUIRREL-CAGE MOTOR



Squirrel Cage Motor

There are four classes of squirrel cage motors available according to National Electrical Manufacturer's Association (NEMA). They are:

a) NEMA Design A

b) NEMA Design B

c) NEMA Design C

d) NEMA Design D

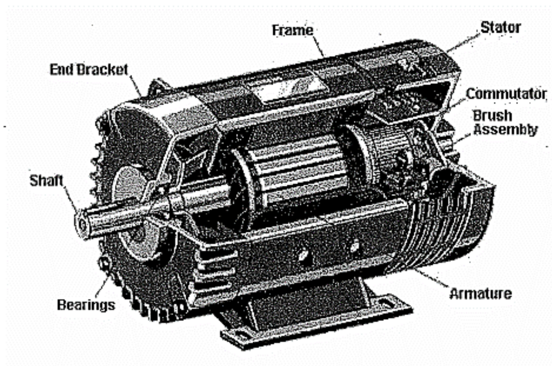
Usually, NEMA Design B is commonly used in industrial application. Since the mining applications, such as conveyors, require heavy starting torque, NEMA C and NEMA D designs are used in the mining industry. Typical mining application for squirrel cage induction motors are:

a) Conveyor belt

b) Pump motor and cutting motors in the continuous miner
c) Ventilation fan motor

The rotor for the squirrel cage motors is die-casted.

5. AC WOUND-ROTOR MOTORS



Wound Rotor Motor

The wound-rotor motor has a phase wound secondary (rotor) having as many poles as the stator windings connected to its slip rings. These slip-rings are in turn connected to an external resistor.

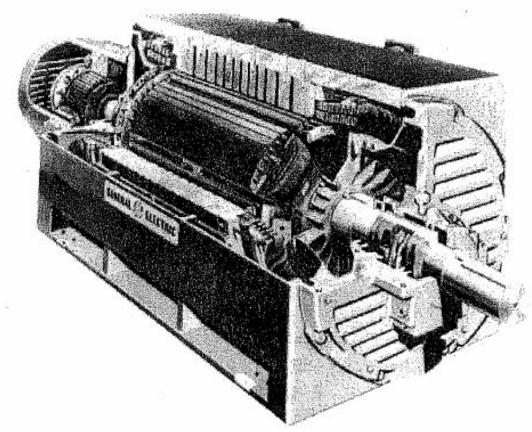
The torque-speed characteristics of this machine can be varied by changing the value of the external resistor in the rotor circuit. Wound-rotor induction motors are used in the conveyor belt applications.

The synchronous speed (N) of an induction motor may be calculated using the following formula.

$$N = \frac{l20f}{P}$$

where f is the supply frequency (60Hz in U.S.) and P is the number of poles. For a two pole machine, the synchronous speed is $N = \frac{120\ x\ 60}{2} = 3{,}600\ rpm.$

6. AC SYNCHRONOUS MOTORS



Synchronous Motor

The stator of a synchronous motor is identical to that of a squirrel-cage motor, and in fact, the synchronous motor is started like a squirrel-cage motor. It is in the rotor that differences occur. The synchronous motor rotor is wound and connected to a DC excitation source so as to produce magnetic poles that are attracted to and lock into step with the rotating magnetic field poles.

In the mining industry synchronous motors may be used to advantage on very high horsepower motors that are operated quite continuously. Therefore, some of their possible applications are for motor-generator sets, main mine fans, large pumps, and crushers.

B. MOTOR STARTERS

The majority of electrical loads in mining are electric motors. There are various methods to start and control the speed and torque output of electric motors. The following is a brief overview of the most common methods utilized on AC motors.

1. ACROSS THE LINE

These are typically referred to as Full Voltage Non-Reversing (FVNR) starters. It is simply a contactor (often with the integral required motor protection such as the over-loads) that is energized by low voltage control elements (switches, floats, etc.) that when operated applies full voltage to the motor. The advantages are simplicity, cost and reliability. The disadvantages are the inrush or starting current is limited only by the windings impedance and will approximately 500 to 700% of the motors full load amps (FLA) until it starts to rotate and assumes load. This can cause voltage drop issues on the power distribution system for large motors and impose undue stress on the motor insulation. As will be shown other methods have been developed to mitigate or minimize these negative effects.

2. REDUCED VOLTAGE STARTER

This technology was prevalent before the advent and refinement of solid state (or static) motor starters or controllers.

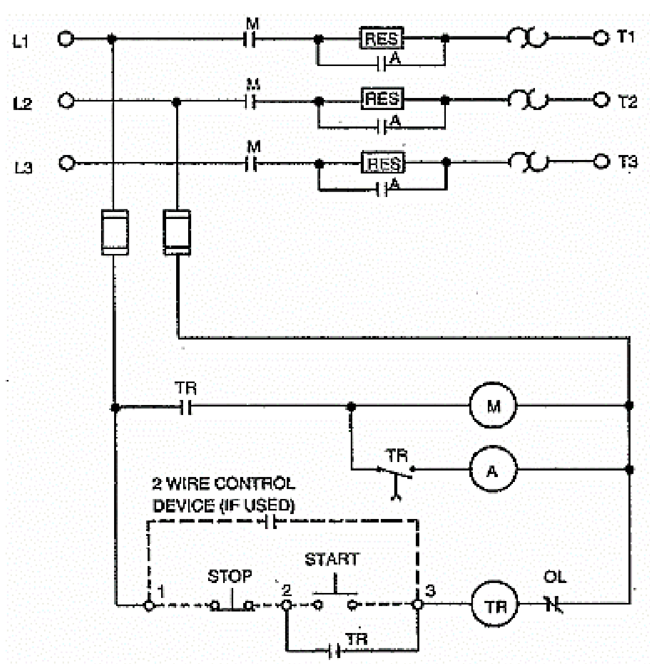
3. AUTO-TRANSFORMER (AT) STARTER

A reduced voltage tap from the AT is used as the power source for motor starting. Since the voltage is less than the motor rating and the impedance of the motor is a constant, the inrush current will be reduced proportionally by the factor $\frac{reduced\ tap\ voltage}{rated\ voltage}$. After a short time delay a contactor(s) will transfer the motor leads from the reduced voltage tap to full or line voltage. For example, if the reduced tap is 240V AC for a 480V AC rated motor inrush current will be halved to the order of 250 to 350% of the FLA.

4. WYE/DELTA TRANSITION STARTER

The principal is the same as the AT starter: use a reduced voltage to get the motor running through the inrush period then transfer to full or line voltage. The implementation is different. The motor is initially connected to the line in a Wye fashion with each motor coil/winding receiving line voltage/(1.732) or line-neutral voltage. In the case of a 480V AC motor this would be 277V AC. After a short time delay contactors will transition the winding configuration from wye to delta therefore putting full voltage (480V AC in our example) across each motor coil. This will reduce the inrush current to 58% (277/480V AC) of the FVNR method. There are several other methods which operate on the same basic principle: during initial starting then transition to full voltage.

Resistance Starting: Motor is started thru a resistor (and therefore reduced voltage) and transitioned to full voltage.



Example of a Resistance Type Starter

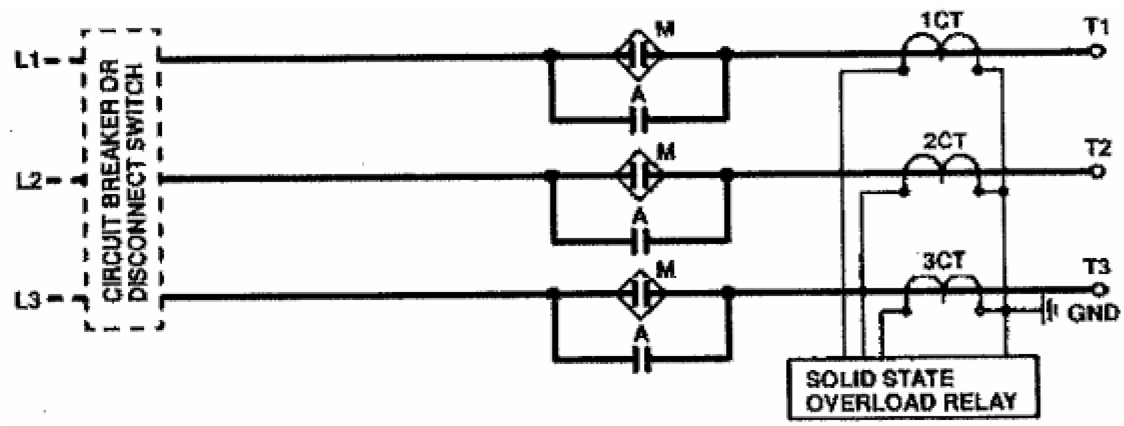
Part winding starting: The Motor is constructed with 2 windings designed to operate in parallel. Voltage is applied to one winding (which only draws ½ the current) and the second winding is connected after a short time delay. With the advancement of solid state electronics technology improved methods have been developed.

5. VARIABLE FREQUENCY DRIVES AND SOFT STARTERS (VFD'S OR SS'S)

With the advent of reliable power semiconductors other methods became available; the two predominant methods are Soft Starters and VFD's. The primary solid state devices used in these starters are silicone controlled rectifiers (SCR) or integrated gate bi-polar transistors (IGBT). These can be thought of as solid state switches.

6. SOFT STARTERS (SS)

The solid state device (SCR or IGBT) is switched on/off using a pulse width modulation scheme to limit voltage and/or current, the product of which is power or the torque. These are initially started out low and gradually ramped up to full voltage/power. It is important to note these typically do not change the line frequency. Once up to speed a by-pass contactor is typically used to shunt the SS out of the circuit.



Elementary Diagram of a Soft Starter arrangement

7. VARIABLE FREQUENCY DRIVE (VFD)

These devices are sometimes referred to as Variable Speed Drives or Inverters. An AC motor's operating speed is proportional to frequency as defined by:

$$Speed\ n\ = \frac{120\ x\ frequency}{Poles\ (in\ RPM)}$$

As can be seen for a fixed number of motor poles the speed in constant if the frequency is constant. For example, a 4 pole motor with 60 Hz will run 1800 RPM. If we reduce the frequency to 30 the speed drops to 900 RPM. Prior to the development of solid state devices, it was difficult and expensive to vary line frequency. That is why the methods described above were used to start motors. This also illustrates why DC motors were so widely used. To change the speed of a DC motor is a relatively simple affair. By varying the field strength the speed and/or torque can be adjusted. By varying the armature current the torque can be adjusted. Manipulating these variables is a simple matter with DC requiring a variable resistance to change or ramp the voltage or to modulate the current. VFD technology changed all this and rendered the above technologies and the use of DC almost obsolete.

8. VFD BASIC OPERATING PRINCIPLES

A VFD is comprised of 4 major subsystems:

1. A rectifier (converts AC to DC)
2. Positive and negative DC buses fed from the rectifier
3. Inverter (converts DC to AC) fed from the DC buses
4. A digital control system

The AC input power rectifier output is connected to the +/- DC bus(es). The solid state devices (usually IGBT's) are connected to the DC bus and to the motor. There is a set of IGBT's on the +DC bus and on the –DC.

Again, it is helpful to visualize the IGBT as an electronic NO (normally open) power switch. The control system 'triggers' this switch ON (current flows), removal of this signal defaults the switch to OFF (no current flow). The switch can be controlled in two ways: The rate at which the switch is operated and the duration the switch is allowed to be closed or conduct current. The rate determines the frequency and the duration the magnitude of the voltage or current, and hence the power or torque.

The control system IGBT firing rate will produce a desired frequency and the duration a power or torque output as determined by the control system. The positive switch (IGBT) will fire the +DC bus to generate the positive waveform and the negative IGBT to make the inverse waveform. The result is the frequency (motor speed) and power output can be precisely controlled.

For example, the VFD upon start-up can be set-up to put out 300% torque (300% FLA which is lower than the line starting value of say 600%) at 5 Hz. It can be set-up to ramp the frequency up to 60 Hz in 20 seconds as it ramps the power down to 100% FLA. A nice smooth start under load is accomplished.

In addition, the VFD drive can also be used to perform speed control functions. For example, a throttle potentiometer can be used as the control input to a VFD. The more the throttle is depressed the higher VFD frequency output, and hence the higher the motor speed.

9. REQUIRED VFD PROTECTION SCHEME

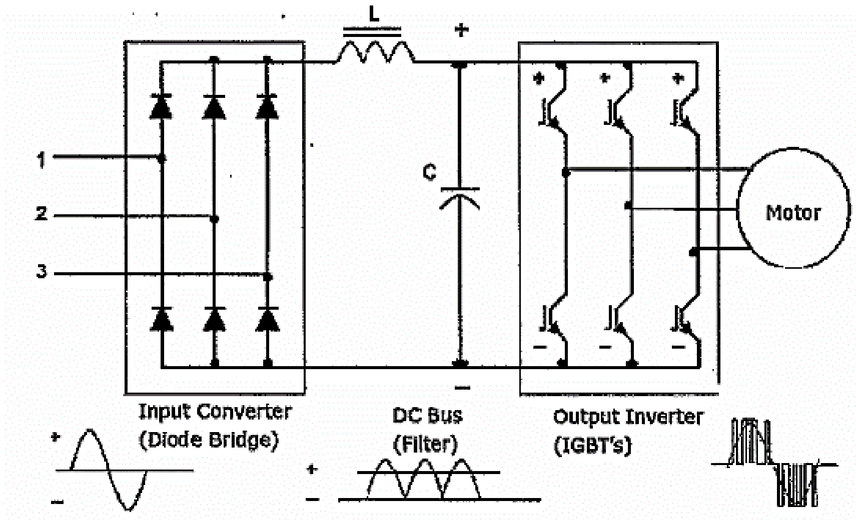
Due to their nature (DC power combined with variable frequency AC power) special electrical protections are required by law. Basically: 3 points of ground fault sensing are required to trip 2 points of interruption. The GF relays must be rated for AC/DC operation and a frequency range of 0-60 Hz. The GF relay must be set less than 50% of the lowest minimum available fault current from the 3 power sources: VFD line input, +/- DC bus(es) or VFD output. All 3 relays must sense a fault at any of the 3 power sources: VFD line input, +/- DC bus(es) or VFD output.

The points of interruption may be:

Main Machine circuit breaker (CB)

Motor or VFD supply contactor or supply CB

It is preferable to have all protection located on the same frame as the VFD.



Typical VFD Elementary Schematic

C. MINE ELECTRIC MOTORS
QUESTIONS AND ANSWERS

MES-6 Q-1: What are mine electric motors employed for?

MES-6 A-1: Electrical motors are preeminent as the means for converting electrical energy to mechanical energy. This is accomplished by utilizing the magnetic principles of attraction and repulsion to cause an interaction between a magnetically charged stationary winding called a stator, and another magnetized rotating element called a rotor.

MES-6 Q-2: What types of motors are commonly used in Underground mines?

MES-6 A-2: For the use of mine electrical power systems, there are three different types of motors:

A. DC motors
   1. Shunt motor
   2. Series motor
   3. Compound motor
B. AC motors
   1. Squirrel-cage motor
   2. Wound-rotor motor
   3. Synchronous motors

MES-6 Q-3: What are the difference between AC motors and DC motors?

MES-6 A-3: The basic differences among motors are the methods employed to produce proper relative polarity between the rotors and stators.

MES-6 Q-4: Name some applications of shunt motors.

MES-6 A-4: Shunt motors are used where a substantially constant speed is required, for example, as a conveyor, pump, and blower drives. They are better adapted than any other type of motor to speed control and speed adjustment. They are not very popular in the mining industry.

MES-6 Q-5: Name some applications of squirrel-cage motors.

MES-6 A-5: Typical mining applications for squirrel cage induction motors:

A. Conveyor belt
B. Pump motor and cutting motors in the continuous miner
C. Ventilation fan motor

MES-6 Q-6: Name some applications for synchronous motors.

MES-6 A-6: In the mining industry synchronous motors may be used to advantage on very high horsepower motors that are operated quite continuously. Therefore, some of their possible applications are for motor-generator sets, main mine fans, large pumps, and crushers.

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 7

SECTION 7

A. MINE POWER DISTRIBUTION

The following provisions shall apply to alternating current electrical systems serving portable face equipment in bituminous coal mines. The fundamental components of such a system are:

1. SUBSTATION
2. SWITCH HOUSES
3. HIGH VOLTAGE UNDERGROUND TRANSMISSION SYSTEM
4. UNDERGROUND MINE POWER and DISTRIBUTION CENTERS

The basic system for both transmission and distribution of alternating current power for face equipment shall be a three-phase four-wire system, with a ground fault current limiting resistor in the neutral circuit and the inby or load end of the neutral resistor solidly grounded. The ground end of the neutral resistor shall be connected to equipment frames through the cable ground conductor to prevent dangerous differences of potential between frame and ground under fault conditions. Figure MES 7-1 presents a block diagram of a basic mine power system. The functions of the various blocks are described below.

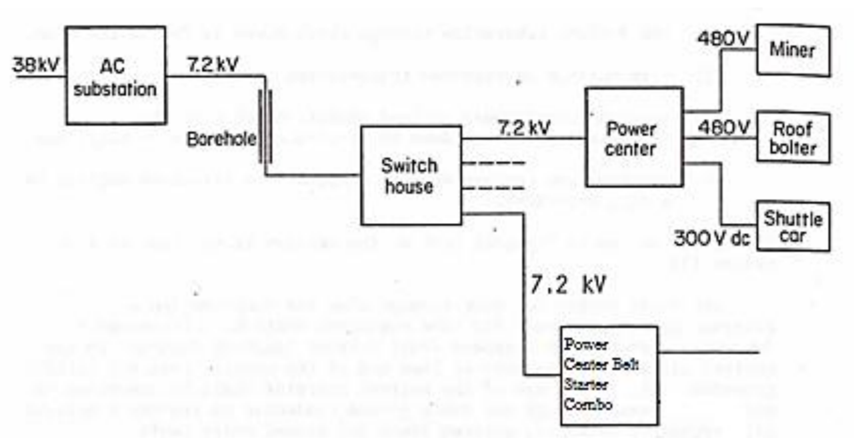


Figure MES 7-1 presents an example of a continuous mining section with the equipment cables connected to a distribution box.

1. SUBSTATIONS

Mine Power Substation – The system that converts surface utility power for use by the underground mine power system.

A. Access shall be restricted by a protective fence or building.
   1. Enclosed by a fence (minimum ht. 7') bonded to the station bed.
   2. The station ground bed shall extend at least three feet beyond the fence and gate swing.
   3. The incoming power static/ground wire shall be connected to the station ground grid.
   4. Total enclosed and locked pad-mounted equipment need not be enclosed by a fence.

B. Primary lightning arrestors are required.

C. Disconnected means on the primary with circuit breaker or fused protection.

D. Two winding transformer to transform primary voltage to transmission voltage.
   1. Underground voltage shall not exceed fifteen thousand volts, nominal, phase-to-phase.
   2. Connections to provide mine transmission voltage.
      a) Delta-Wye, Wye-Delta and Delta-Delta are permitted connections.
      b) Wye-Wye connections or autotransformers are not allowed.
      c) Surface power transformers must be installed within the substation ground grid.
      d) "Zigzag" or grounding transformers may be utilized on Delta-connected transformers.

E. Secondary lightning arrestors are required.

F. Ground fault current limiting resistor (or NRG) shall be:
   1. Limiting ground fault current to 25 amperes or less.
   2. Rated for phase-to-phase voltage.
   3. Protected by grounded fence or mounted at least eight feet above ground.

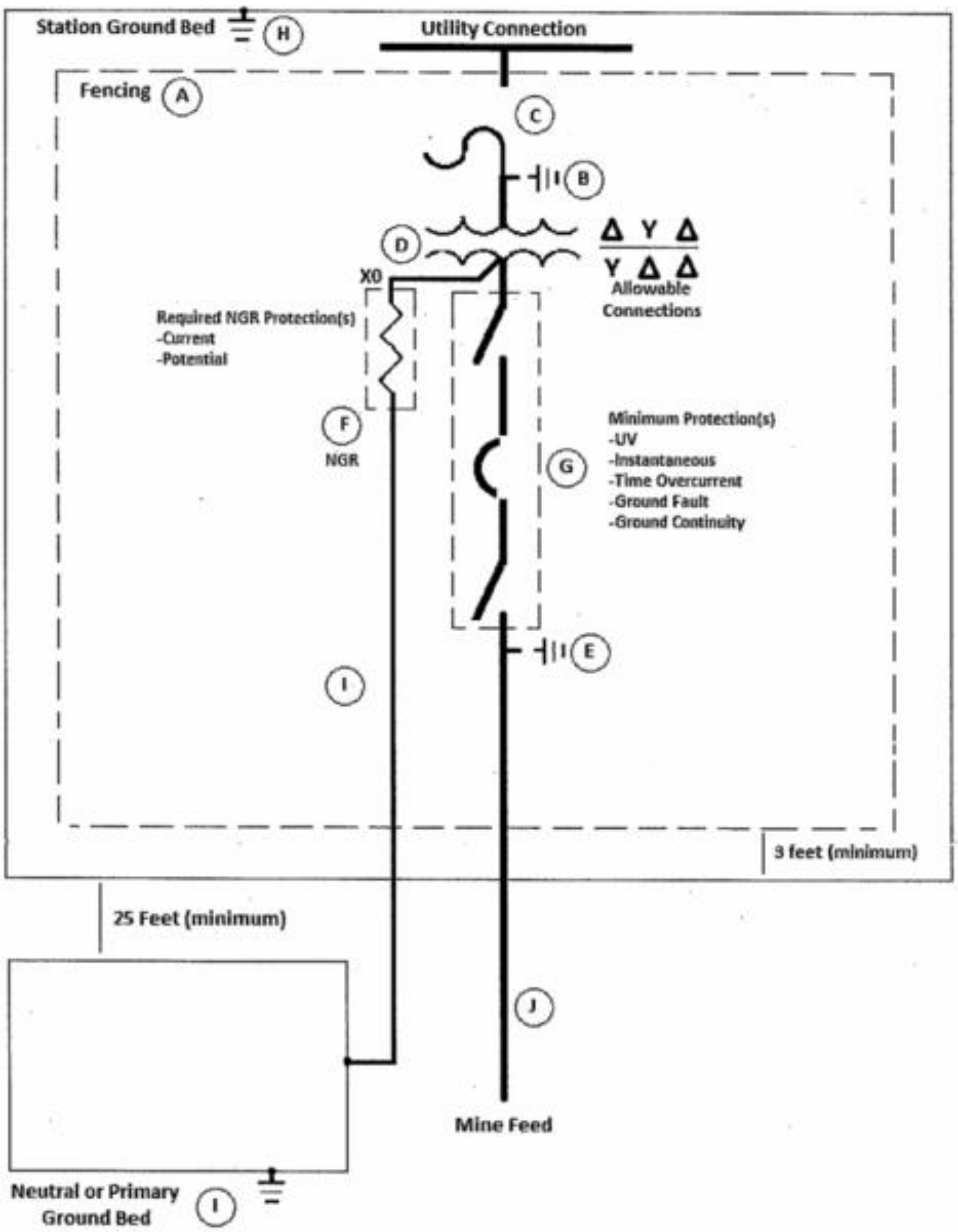
G. Secondary or mine feeder circuit breaker.
   1. Disconnects shall be provided on the input and output sides of the breaker.
   2. Use of automatic reclosing circuit breakers is prohibited.
   3. Circuit Breaker trip inputs.

        a) Under-voltage set to drop out no lower than 40% of control voltage.
        b) 3 Phase Instantaneous overcurrent (phase-phase short-circuit).
        c) 3 Phase Time Overcurrent relays on all three phases.
        d) Ground fault (<50% of NGR resistor rating).
        e) Ground-continuity check.
        f) NGR resistor protection/monitoring by current and potential.
    4. 3 phase ammeter and voltmeter.
    5. Circuit breaker status indication.

H. Station ground bed
    1. Connect lightning arrestors, equipment frames, fence, enclosures, etc.
    2. Isolated from Neutral or Primary ground bed.
    3. Less than or equal to 4 ohm.

I. Neutral or primary ground bed maintained at four ohms or less.
    1. Located at least twenty-five feet away from any station ground. This connection shall be made by an insulated conductor.
    2. The only connection shall be the inby (load) side of the NGR by an insulated conductor.

J. High voltage conductors or cables leading underground from the mine power substation shall be flame resistant type and adequate for the intended current and voltage.



The function of the substation is to receive electrical energy at voltages too high for mining purposes and deliver this energy at other voltages to the various mining load centers.

The substation may receive power at 38,000 volts, for example, and transform it to 7,200 volts for general distribution. Two or more distribution lines may leave the station at 7,200 volts; one for surface plant distribution, the others for underground power distribution. Power loads and lighting loads are usually kept separate to avoid the flickering of lamps when the motor loads are thrown on and off the line. Transformers are used to step down the voltage for lighting purposes and low-voltage rated motors.

2. SWITCH HOUSES

Portable equipment that protects the underground high voltage transmission circuit. Their internal components are chiefly protection devices, with circuit de-energization performed by disconnect switches, oil circuit breakers, or vacuum circuit breakers. Control relays typically provide minimum automatic tripping of the circuit breaker by under voltage, instantaneous and inverse time limit phase overcurrent, ground fault current not to exceed fifteen amperes and ground-continuity check not exceeding seven amperes.

The secondary voltage from the surface substation of a mine ranges up to 15 KV. Power in this voltage range is distributed by cables through boreholes or through mine entrances to one or several portable switch houses which are skid-mounted, metal-encased boxes.

In a mine sectionalized by portable switch houses, the main function of the switch houses is to protect the various sections from any fault or abnormal condition. By sectionalizing a mine with several of these switch houses, production is maintained in the other sections when a fault occurs in any one section. To maintain and put distribution power back into service when a fault occurs, the portable switch house is equipped with the electrical equipment necessary to accomplish this and with the least possible delay. Thus the circuitry of the switch house can be expected to have most of the protective components of the substation.

A portable switch house usually includes the following components: (1) incoming line, gang-operated disconnect switch; (2) oil circuit breaker and vacuum breaker; (3) protective relaying; and (4) cable couplers.

3. HIGH VOLTAGE UNDERGROUND TRANSMISSION SYSTEM

In transmitting or distributing power, four factors must be considered in determining the size and material of the conductor:

1. The wires must be able to carry the required current without overheating.
2. The voltage drop to the load must be kept within reasonable limits.
3. The wires must be of sufficient mechanical strength.This is important when wires are supported on hangers.
4. The economics of the problem must be considered, i.e., the size of the conductor should be selected which minimize energy loss and maximizes the interest on the investment.

Excerpt from the Law

Section 331. High-voltage underground transmission system.

(a) Underground.--High-voltage cables leading underground and extending underground shall be of the multiple conductor flame-resistant type with a rubber, plastic or armor sheath meeting the requirements of the department for flame resistance. They shall be equipped with metallic shields around each power conductor. One or more ground conductors shall be provided of a total size either:

(1) not less than one-half the power conductor size; or

(2) capable of carrying two times the maximum ground fault current. There shall also be provided an insulated conductor not smaller than No. 10 AWG for the ground-continuity check circuit. Cables shall be adequate for the intended current and voltage. Splices made in the cable shall provide continuity of all components and shall be made in accordance with the cable manufacturers' recommendations. A competent individual designated by the mine electrician shall supervise the making of splices.

(b) Subject to flexing. --High-voltage cables subject to repeated flexing shall be similar in construction to type SH-D in accordance with Insulated Power Cable Engineers Association standard S-19-81.

(c) Couplers. --If couplers are used, they shall be of the three-phase type with a full metallic shell and shall be adequate for the voltage and current expected. All exposed metal on the couplers shall be grounded to the ground conductor in the cable. The coupler shall be constructed so that the ground continuity conductor shall be broken first and the ground conductor shall be broken last when the coupler is being uncoupled.

(d) Equipment passing over or under cable. --At locations where cables cross haulageways or travelways or where equipment must pass over or under the cable, the cables shall be either installed in a trench in the roof, protected by some mechanical means or buried at least 12 inches below combustible material and adequately protected from crushing by the weight of equipment passing over it.

(e) Location of installation. --High-voltage cables shall be installed only in intake airways. They may be installed on intake haulageways only with the approval of the department. The cable may be installed by hanging on suitable hooks or clamps, supported by a suitable messenger cable, burying or installing in metal conduit. When suspended, the distance between supports shall not exceed 20 feet, and they shall be so placed that they do not damage the cable jacket. When hung in a haulage entry containing a trolley wire, the cable shall be installed at least 12 inches from the trolley wire or feeder wires and away from the track.

(f) Excess cable. --Any excess cable which is connected and supplying a load shall be coiled, stored on a reel or otherwise stored at a place near the load where it can be protected by dangering off the storage area. The cable shall not exceed 1,000 feet in length.

(g) Frames and enclosures. --Frames and enclosures of high- voltage switch units, transformers, metallic cable couplers and splice boxes shall be grounded to the common or primary ground for the system in the high-voltage cable.

(h) Taps or branch circuits. --Taps or branch circuits from the high-voltage feeder shall be made through circuit breakers or suitable load break switches.

(i) Non-load breaking disconnect switches.--When non-load breaking disconnect switches are used for sectionalizing high-voltage circuits, they shall be fully metal clad, equipped with a door interlock to break the ground-continuity check circuit, thus tripping the feeding breaker when the door is open, and a voltmeter or indicating lights to verify that the circuit is de-energized before the disconnected switches are opened.

(j) Applicability. --For the purpose of interpretation and compliance with subsection (h) and section 313(h), the following apply:

(1) A branch circuit is a sub portion of the high-voltage system, serving one or more loads. The branch circuit begins at the junction or splitting of the high-voltage system. The junction consists of the following distinct elements:

(a) Input feeder, which delivers power from the source.

(b) Output feeder, which may extend the feeder to other parts of the high-voltage system.

(c) Branch circuit.

The output feeder is not considered as a branch circuit and is not required to have electrical protection at the junction, but receives electrical protection either at the source substation or at some place between the source substation and the junction. The branch circuit is required to have protection at the junction.

(2) A tap supplies power to the high-voltage loads located entirely within the enclosure where the connection is made. Where no splitting of the feeder cable occurs, neither a tap nor branch is created.

(3) A suitable load-break switch, which may be used in lieu of a circuit breaker, is a gang-operated switch with a voltage rating not less than the system voltage, capable of interrupting a current equal to its continuous full load rating and to be used in conjunction with fuses to provide overload and short circuit protection for the load being served.

4. UNDERGROUND MINE POWER and DISTRIBUTION CENTERS

The heart of the AC mine power system is the power load center. Here the incoming substation voltage is transformed to utilization voltage. Usually the incoming power, whatever the voltage, is transformed to a utilization voltage of between 480 and l000V. From this secondary connection of the mine power center transformer, either a cable extends to a distribution box or cables for the various portable machines are connected.

Mine Power Load Centers and Distribution Centers

A. Input will be through a coupler

B. A positive visible disconnect (air load break switch (LBS), vacuum (LBS), CB w/cut-out):

C. If a LBS: disconnect power on the outby feeding circuit breaker or open the load center's main secondary breaker.

D. If a vacuum LBS: Interlock to break load before opening

E. Protection:
   1. Circuit breaker: instantaneous, overcurrent and under-voltage relaying
   2. Suitably sized fusing.

F. A maintained emergency stop switch.

G. Barriers and electrical interlocks to prevent inadvertent access for input and output sections greater than 1,000 volts nominal. Barriers and interlocks must be used to isolate circuits of greater than 1,000 volts from utilization circuits.

H. There will be no wye-wye connections unless a 2 winding secondary (which of at least one must be a delta connection) if utilized.

I. The load center shall be fireproof construction

J. Ground fault limiting resistor (or NGR) of no greater than 15 A with requirements similar to the substation standards.

K. Main breaker tripping devices, which must include:
   1. Short circuit
   2. Overload/Time over-current
   3. Ground fault (current)
   4. Ground fault (potential) or continuous NGR monitoring
   5. Under-Voltage
   6. Time delay tripping for coordination is allowed

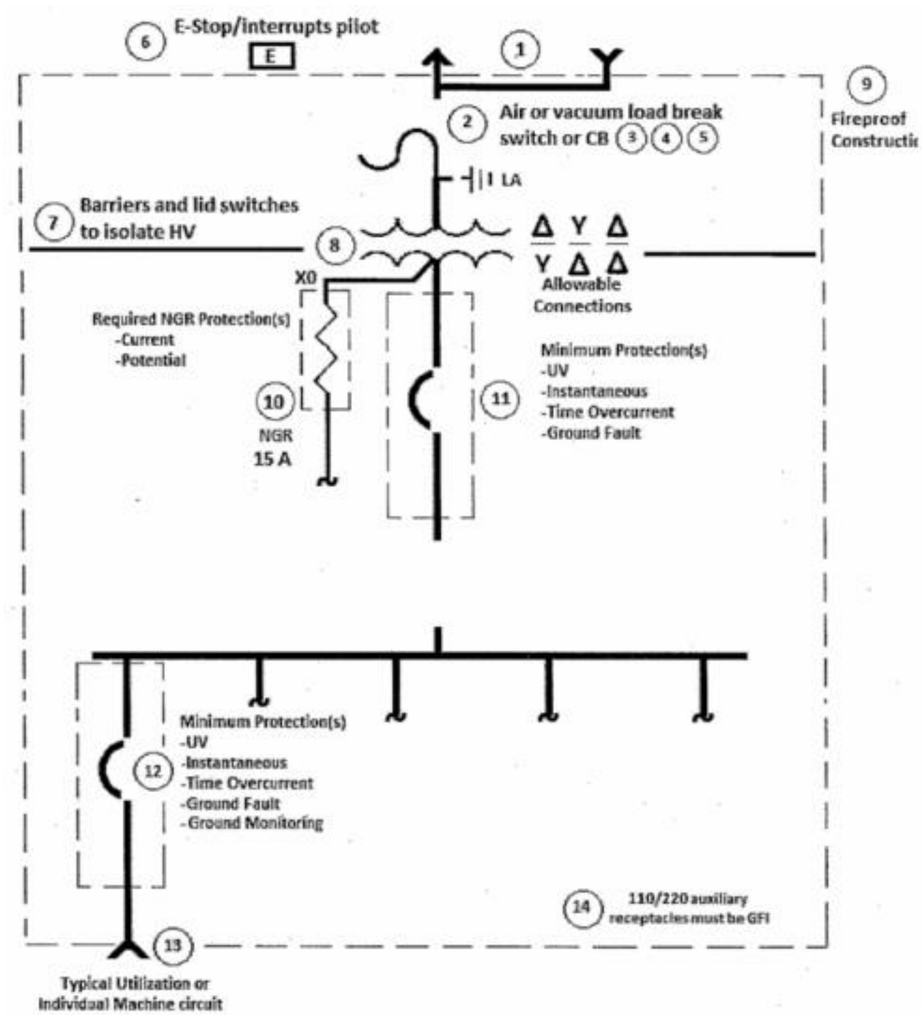
L. Individual machine breakers (or utilization circuits)
   1. Each output circuit shall have the following:
      a) Phase fault (with lockout)
      b) Overload (with lockout)
      c) Ground fault (with lockout). Set <50% of the NGR value
      d) Under-Voltage

e) Ground monitor

M. Individual machine breakers (or utilization circuit) Receptacles/Plugs

1. Grounding/Bonded
2. Ground connection to each output receptacle will be solidly connected to the NGR (the power center frame cannot be used as the conductor)

N. Ground fault interrupter (GFI) on all 110 and 220 volt receptacles.



Mine Power Load Center (General Arrangement)

The following sections describe in detail some of the important components of the various systems

5. LIGHTENING ARRESTORS

Substations are also protected by lightning arresters that carry off to ground the high voltage from electrical storms. The lightning arrester provides a path over which the spike or transient can pass to the ground before it has a chance to damage the transformer or other equipment or be transmitted underground.

The lightning arresters connected into the substation ground system may be any of several types. The simplest type consists of an air gap with a resistive element in series. The arc caused by the power surge jumps the air gap and follows a path through the resistive element to the ground. There is no chance for the line current to pass to ground because a surge of power is necessary to cause the arc to jump the air gap.

6. CIRCUIT BREAKER, SWITCH, AND FUSE

A circuit-breaker or switch capable of breaking the circuit under load or fault is required at the origin of each main circuit and at the origin of each branch circuit. Provisions must also be made in each separate circuit for cutting off all fault current automatically from the supply line to prevent danger in the event of a fault. To meet this requirement fuses or automatic circuit-breakers are required in each main circuit, each branch circuit, and all apparatus connected to the circuit.

A fuse is a device designed to open an electric circuit whenever the flow of current becomes too great, either from a "short circuit" or an over-supply of current. The resistance of the fuse is so calculated as to develop enough heat to melt when the current exceeds the normal value. In order to protect the electrical equipment and circuit wiring, a fuse must melt at a lower temperature than that needed to destroy the insulation of the other components of the circuit or that is needed to ignite the most combustible material in or near the circuit. The vapor formed must also be a poor conductor so as not to produce an electric arc between the fuse terminals.

Fuses are usually limited to circuits where short circuits or heavy overloads are infrequent, but protection must be positive against such possible faults. The low cost, infrequent maintenance, ease of replacement and no mechanical difficulties are a few advantages of their use. The chief disadvantage has been the ease with which a fuse can be replaced with one of a larger current rating. However, this is restricted to a certain extent by varying the designs of the fuses and fuse receptacles according to the current rating.

Common causes for a "blown fuse" are: (1) starting conditions develop a high surge of current; (2) wrong fuse rating; (3) poor ventilation preventing dissipation of normal heat developed in service; (4) short circuits; and (5) poor contacts.

All of these causes can normally be eliminated by careful installation. The time-lag type of fuse should be used to eliminate the blowing of fuses as a result of momentary abnormal current loads during the starting of some motors.

Circuit breakers are in many ways superior to fuses in protecting electrical circuits and apparatus from overloads and short circuits. The circuit breaker is more flexible, may have time element and under voltage trips added, may be opened and closed by remote control, and may have a reclosing action and trip in accordance with the rate of current rise.

Circuit breakers cost more than fuses, but they are preferable for use on circuits subject to frequent overloads, in circuits of large capacity, and on motor feeder circuits.

7. CAPACITOR PACK

The fluctuating nature of the mine loads at the face area result in adverse power factors. However, since severe penalties are exacted by the utility company for adverse power factors, power factor correction must be practiced through the use of capacitor packs.

Capacitors should be placed as close to the motor load as possible. But since their weight and bulk make it impractical to place them on the mining machines themselves, they are usually skid-mounted to facilitate moving them and are connected to the power center outby, no correction being provided on the trailing cables.

8. DEVICE SETTINGS (relocated from above)

Coordination of a mine power system entails complete organization of time settings and/or current settings for all protective devices from the loads to the sources. This necessitates a comprehensive coordination study of the entire system to determine the range of correct values for all instrument transformers, pickup and time settings, fuse ratings, and circuit breaker trip ratings, which will provide effective coordination and selectivity and ensure that the minimum of unfaulted load is disturbed when protective devices isolate a fault. The prime concern is overcurrent, since the circuitry must provide simultaneous overload, short-circuit and ground-fault protection without causing nuisance tripping. The application of this "art" is perhaps the most perplexing problem facing practicing engineers. A system fault analysis is vital input for any comprehensive coordination study and should address not only maximum values but also minimum values, together with the normal operation and maximum allowable currents.

The balance of this section is broken into the major aspects of the coordination study: relay pickup settings, CT matching, circuit breaker trip settings, fuse characteristics, and overall coordination. Emphasis is placed on radial ac systems that are high-resistance grounded.

9. RELAY PICKUP SETTINGS

Pickup has already been defined as the minimum value of the actuating quantity that will cause a relay to operate its contacts. Whether the application is at a trailing cable, feeder cable, or overhead conductor, the requirements for overload, short-circuit, or ground-fault protection usually translate into current pickup settings. The values used in this section are generally in line with those contained in 30 CFR 75 and 77, which are in effect at this writing. The quantities that define pickup may change in the future, but the techniques presented here for establishing relay pickup settings should not. Because of their widespread use, induction-disk relays are implied in most of the following pickup applications; however, pickup techniques for other relays are basically the same. To avoid confusion, molded-case circuit breaker trip settings will be covered later.

Establishing a pickup setting for an AC relay involves selecting a CT ratio and operating-coil current. For marginal short-circuit currents, the accuracy of the combination might require verification. The following material describes pickup settings for a single zone in a mine power system, but it must be remembered that the overall goal is to obtain coordination, and the settings at any location can be affected not only by requirements and regulations but also by other upstream and downstream relaying.

10. SHORT-CIRCUIT PROTECTION

Short-circuit protection can be obtained with an instantaneous element (no intentional delay) or an inverse-time overcurrent relay using the minimum time dial setting (maximum time delay here is often restricted to no more than 0.6 s). The general requirements can be determined by selecting the lower value calculated from the following:

1. 115% of the maximum starting current or 115% of the peak load current, whichever is higher, for the equipment being protected; or

2. 60% of the smallest bolted three-phase symmetrical rms fault current for any point of the zone protected by the relay.

The first value is designed to have pickup above the normal operating current to prevent nuisance tripping. Usually, the bolted fault current value is higher than the first value.

Generally, starting current of motor loads starts at a protection level of 700% of FLA. Instantaneous setting of protection breaker can be adjusted to 1300% of FLA if applications require.

The motor full-load current can be estimated from

$$I_{fl} = \frac{746(hp)}{\sqrt{3\, V_{\eta}\, (pf)}}$$

Where hp = rated machine horsepower
V = rated line-to-line voltage of motor, V,
η = motor efficiency,
pf = full-load power factor, which can be assumed to be 0.85

Inrush currents for transformers usually range from 8 to 12 times the full-load current rating for a duration of 0.1 s; typical values for inrush currents should be available from the transformer manufacturer.

To show how short-circuit pick-up is selected, consider that a production shovel in a surface mine has 2,000-hp connected load rated at 4,160 V line to line. The shovel is powered through 1,000 ft. of 4/0 AWG cable. The shovel induction motor is 85% efficient at full load and operates at 0.8 pf. The minimum value of bolted three-phase fault current has been found to be 6,130 A. CT ratios are 400:5 A, and the instantaneous element has a pickup range from 10 to 50A. The procedure is to find the minimum current according to the above criteria.

First, for the fault current, the pickup would be 60% of the bolted value divided by the CT turns ratio:

$$Pickup = \frac{(0.6)(6{,}130)}{80} = 46\,A$$

This value must not be compared with the pickup needed to slightly exceed maximum starting current, which may be estimated using the following equations. Here, the full-load current of the shovel motor is about:

$$I_{fl} = \frac{746(2{,}000)}{\sqrt{3}\,(4{,}160)(0.85)(0.8)} = 305\ A$$

The estimated motor starting current is 700% x FLA:

$$I_S = (7)(305) = 2{,}135\ A$$

Allowing 115% to prevent nuisance tripping, the pickup setting is then

$$Pickup = \frac{(2{,}135)(1.15)}{80} = 31\ A$$

As this is less than the value for the fault current, 31 A is the selected pickup setting for short-circuit protection.

B. MINE POWER DISTRIBUTION
QUESTIONS AND ANSWERS

MES-7 Q-1: Name the fundamental components of the mine power distribution system.

MES-7 A-1: The fundamental components of the system are:

1. SUBSTATION
2. SWITCH HOUSES
3. HIGH VOLTAGE UNDERGROUND TRANSMISSION SYSTEM
4. UNDERGROUND MINE POWER and DISTRIBUTION CENTERS

MES-7 Q-2: What is the function of the substation?

MES-7 A-2: The function of the substation is to receive electrical energy at voltages too high for mining purposes and deliver this energy at other voltages to the various mining load centers.

The substation may receive power at 13,200 volts, for example, and it to 2300 volts for general distribution. Two or more distribution lines may leave the station at 2300 volts; one for surface plant distribution, the others for underground power distribution. Power loads and lighting loads are usually kept separate to avoid the flickering of lamps when the motor loads are thrown on and off the line. Transformers rated at 2300 to 230-115 volts are used to step down the voltage for lighting purposes and low-voltage rated motors. Many AC motors are operated at 440 volts by some members of the mining industry.

MES-7 Q-3: What is the purpose of a switchhouse?

MES-7 A-3: Portable equipment that protect the high voltage transmission circuit. Their internal components are chiefly protection devices with circuit de-energization performed by disconnect switches, oil circuit breakers, or vacuum circuit breakers. Control relays typically provide minimum automatic tripping of the circuit breaker by under voltage, instantaneous and inverse time limit phase overcurrent, ground fault current not to exceed fifteen amperes and ground-continuity check not exceeding seven amperes.

MES-7 Q-4: Discuss the four factors used to determine the size and material of electric conductors.

MES-7 A-4: In transmitting or distributing power, four factors must be considered in determining the size and material of the conductor:

(1) The wires must be able to carry the required current without overheating.

(2) The voltage drop to the load must be kept within reasonable limits.

(3) The wires must be of sufficient mechanical strength. This is important when wires are supported on hangers.

(4) The economics of the problem must be considered. That size of the conductor should be selected which makes the cost of the energy loss plus the interest on the investment a minimum.

MES-7 Q-5: What is a fuse and its required characteristics?

MES-7 A-5: A fuse is a device designed to open an electric circuit whenever the flow of current becomes too great, either from a "short circuit" or an over-supply of current. The resistance of the fuse is so calculated as to develop enough heat to melt when the current exceeds the normal value. In order to protect the electrical equipment and circuit wiring, a fuse must melt at a lower temperature than that needed to destroy the insulation of the other components of the circuit or that needed to ignite the most combustible material in or near the circuit. The vapor formed must also be a poor conductor so as not to produce an electric arc between the fuse terminals.

MES-7 Q-6: List the advantages and disadvantages of a fuse.

MES-7 A-6: Fuses are usually limited to circuits where short circuits or heavy overloads are infrequent, but protection must be positive against such possible faults. The ease of replacing, low cost, infrequent maintenance, and no mechanical difficulties are a few advantages of their use. The chief disadvantage has been the ease with which the fuse can be replaced with one of a large current rating. However, this is restricted to a certain extent by varying the designs of the fuses and fuse receptacles according to the current rating.

MES-7 Q-7: What are the common causes for a blown fuse?

MES-7 A-7: Common causes for a "blown fuse" are:

(1) Starting conditions develop a high surge of current;

(2) Wrong fuse rating;

(3) Poor ventilation preventing dissipation of normal heat developed in service;

(4) Short circuits; and

(5) Poor contacts.

MES-7 Q-8: Where are circuit-breakers used most effectively?

MES-7 A-8: A circuit-breaker or switch capable of breaking the circuit under load is required at the origin of each main circuit and at the origin of each branch circuit. Provisions must also be made in each separate circuit for cutting off all pressure automatically from the supply line to prevent danger in the event of a fault. To meet this requirement fuses or automatic circuit-breakers are required in each main circuit, each branch circuit, and all apparatus connected to the circuit.

Circuit breakers cost more than fuses, but they are preferable for use on circuits subject to frequent overloads, in circuits of large capacity, and on motor feeder circuits.

MES-7 Q-9: Name the essential parts of the skid-mounted mine power center unit.

MES-7 A-9: The essential parts of the skid-mounted unit are:

(1) An incoming-line, primary-voltage compartment;

(2) A dry-type, air-cooled transformer; and

(3) A low-voltage panel board of molded case air circuit breakers.

Secondary equipment includes ground relays, current transformers, a ground resistor, capacitor banks for load power factor correction, and a low-voltage transformer for auxiliary lighting. Some units also contain primary protective relays. There is no single unit built that fits all conditions. Usually, a unit is built to a mine’s power specifications, utilizing various standard components.

MES-8 Q-10: What is the function of capacitor pack?

MES-8 A-10: The fluctuating nature of the mine loads at the face area result in adverse power factors. However, since severe penalties are exacted by the utility company for adverse power factors, power factor correction must be practiced through the use of capacitor packs.

MES-8 Q-11: Where are capacitor packs placed?

MES-8 A-11: Capacitors should be placed as close to the motor load as possible. But since their weight and bulk make it impractical to place them on the mining machines themselves, they are usually skid-mounted to facilitate moving them and are connected to the power center outby, no correction being provided on the trailing cables.

TABLE OF CONTENTS

MINE ELECTRICAL SYSTEMS SECTION 8

SECTION 8
A. GROUNDING

Grounding is that procedure necessary to maintain a machine frame as near earth potential as possible. It is equally important to adequately bring the frames of the equipment housing the source of power to earth potential. Figure MES 8-1 presents a typical resistance grounded power system. The distribution/transmission supply is brought into the power center to the primary of the transformer. The neutral of the y-connected secondary is connected to the safety ground and the ground wire through the safety ground resistor. For utilization voltage systems, this resistor must limit phase-ground fault currents to 25 amps or less. The safety grounded resistor reduces the shock hazard during fault conditions. If a ground fault occurs, a ground-fault relay opens a set of contacts in series with the under voltage release (UVR) to trip the circuit breaker. The safety of this system depends upon the integrity of the ground wire between the load machine frame and the connection to the safety grounded resistor. If the ground wire is open when a phase-machine frame fault occurs, a person touching the machine frame can receive a lethal shock. This may seem an unlikely event, but since the ground wire normally carries no current, operations can continue for a considerable length of time with an undetected break in the ground wire.

The unreliability of local ground connections in a mine, such as the machine resting on the bottom or driving rods into the bottom has resulted in the establishment of the practice of running a separate ground wire to each machine frame. This wire should be connected to a continuous grounding system leading back to the frame of the source of power supply with adequate contact to earth.

All metallic frames, casings, and coverings of motors, generators, switchboards, and other electrical equipment that may be electrified by failure of insulation should be grounded by this method. It is accepted practice, however, to use trailing cables containing a ground conductor of not less than half the size of the power conductors.

Further protection is suggested when two or more machines are used together in the same working place. The contact between the frames of the machines, such as a continuous miner or loading machine boom touching a shuttle car, may result in electric sparks or flashes due to faulty insulation in one or both machines. An effective ground will prevent this hazard. In mobile equipment, such as a continuous miner or loading machine operating in conjunction with a shuttle car, a difference of potential between the machine frames can be overcome by employing trailing cables containing a separate grounding conductor. The grounding wires from each trailing cable should be attached to the ground circuit at a separate point and nearer the power supply than the return conductors from the machines. All grounding wires for machines operating as a unit should be connected at the same spot in order to eliminate the possibility of voltage differences.

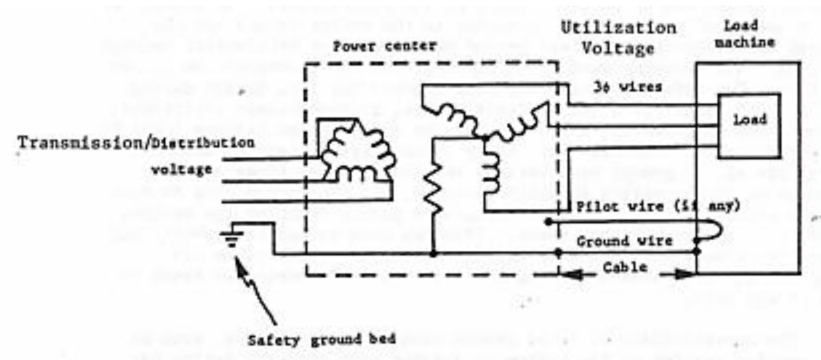


Figure MES 8-1: Typical Resistance Grounded Power System

A point to be considered in grounding is that separate ground wires may become electrified should a rail, bond, or any other part of the return circuit be broken. However, properly installed protective equipment, such as adequate circuit breakers or fuses, when placed in both sides of the branch circuits will remedy this situation. The following are commonly known grounding methods for AC systems:

A. Solidly grounding. Solidly grounded systems have the power transformer or generator neutral connected directly to earth ground. This type of grounding is used on the power systems on the surface.

B. High Resistance Grounding. When high-resistance grounding is employed, the system neutral is connected through a resistance element whose ohmic value would allow a ground fault current greater than system charging current (usually less than 2% of the 3-phase fault current). Advantages include low fault current, low transient overvoltage, and the ability to easily locate the faulted system component. Disadvantages include high initial cost of grounding equipment and necessity of using 100% rated lightning arresters. Mine power systems are high resistance grounded, because of the poor grounding conditions.

C. Derived Neutral from a Zig-Zag Transformer. Wye-delta and delta-delta transformers offer very high impedance to zero sequence (ground fault) currents. In order to detect ground faults in the above cases, a derived neutral is obtained through the use of a zig-zag transformer. This transformer connection is very sensitive to ground fault currents and is insensitive to normal line currents.

D. Diode Grounding. In mixed (AC and DC) systems such as underground mine power system a diode is used in the DC face equipment grounding. When a diode is used, the center-tapped transformer is removed because the frame is grounded through the diode to the negative conductor. The diode circuit also includes a ground protective device which will interrupt the power if a positive-power-conductor-to- equipment-frame current flow occurs. Diode grounding should ensure good ground continuity since the same wire acts as both DC negative conductor and ground wire. However, a grounding diode protects the DC subsystem against ground faults within the equipment frame only. Faults within trailing cables can still present hazards.

E. Ungrounded Power System. Most commonly found at surface facilities, in a three phase ungrounded power systems there is no intentional connection to ground. In order to provide a fault path between different faulted phases equipment is bonded together. If a phase faults to a frame on one piece of equipment and a different phase faults to a different piece of equipment the bonding will cause a phase-phase fault and clearing by the circuit breaker. If a single phase faults to frame there is no hazard as the frame potential is zero and there is no ground return path.

In order to indicate ground fault, detection and indication is required. This may take the form of phase lights or relaying.

An advantage of an ungrounded system is that a single phase-ground fault will allow the system to operate since all 3 phase-phase voltages will remain the same and the ungrounded neutral reference point will shift with the faulted phase going to zero.

1. GROUND-CHECK MONITORS

Low- and medium-voltage resistance-grounded systems are required to have a fail-safe ground-check circuit to continuously monitor the continuity of the grounding conductor. The monitor must cause its associated circuit breaker to trip if the grounding conductor or a pilot wire is broken. An indicator lamp on the monitor should indicate a tripped condition. The monitors are usually enclosed in a dead-front package and mounted near the associated circuit breaker. Monitors in common use in mining can be divided into two general classifications, impedance types and continuity types.

Impedance-type monitors require the trailing cable to have a pilot conductor. The monitor is calibrated to the impedance of the loop formed by the pilot and grounding conductors. The device then monitors the change of impedance from the initial calibration. If the impedance of the loop increases beyond a preset value, the monitor must trip its associated circuit breaker by opening a set of contacts in series with the under-voltage release.

The maximum allowable increase in impedance is dependent upon the maximum ground-fault current permitted by the system. For a 15-A neutral-grounding resistor, the monitor should cause tripping is the impedance increases by 2.7Ω . This value is based upon the maximum allowable frame-to-ground potential of 40 V, as follows:

$$Z = \frac{V_{max}}{I_f} = \frac{40}{15} = 2.7\Omega$$

From the above, it is apparent that a ground-check monitor does not ensure that the frame potential of a piece of equipment will not rise above 40 V, since the device only monitors the change in impedance of the pilot and grounding-conductor loop and not the actual impedance of the grounding conductor.

A schematic of a common impedance-type monitor is shown in Figure MES 8-3. This specific monitor is powered from a 24-V to 32-V source (others do not use 120-V power). Out-of-phase induced currents during motor starting can result in canceling out the monitoring current, which in turn can cause nuisance tripping. As a result, a polarity-reversal switch should be provided to change the phase relationship of the pilot current with respect to the induced current. Some manufacturers use impedance-matching transformers to amplify the change in impedance for easy detection. The monitor should also provide a test button. With the button depressed, the appropriately sized resistor is inserted into the pilot circuit and should result in tripping the circuit breaker.

A disadvantage of the impedance-type monitor is that it cannot detect a pilot-to-ground fault. This type of monitor is also susceptible to problems with parallel paths.

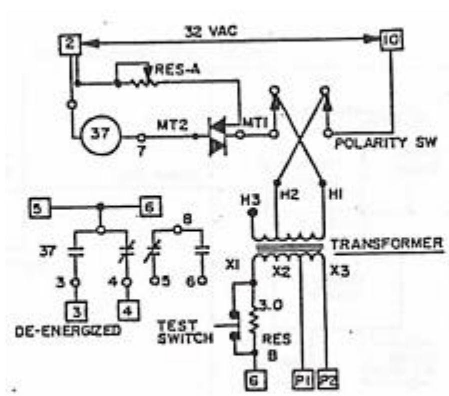


Figure MES 8-2: A Typical Impedance-Monitor Circuit

Continuity monitors do not monitor impedance change, but only the grounding-conductor continuity. However, they must be adequately immune to parallel paths. Continuity types (also termed "pilotless " or "wireless") are of the audio type and do not require a pilot conductor for operation. Figure MES 8-3A contains a block diagram of common unit. Most makes can also be wired for pilot operation (see Figure MES 8-3B), but only the operation of the pilotless configuration will be discussed.

The monitor generates an audio frequency which is coupled to the grounding conductor by means of the transmitting coil. The pilot wire is eliminated by using the phase conductor as a return path. A filter at the monitoring location and within the monitored machine is necessary for coupling and uncoupling the audio signal from the phase conductors. If the grounding conductor is intact, the receiver coil picks up the audio signal. If the grounding conductor is open, the receiver coil will pick up no signal, and the monitor will cause a set of contacts to open in the under voltage circuit of its associated circuit breaker. A continuity monitor operates in a somewhat different fashion; the signal is impressed and removed from the phase conductors with the grounding conductor used for the return path.

The major advantage of the continuity monitor is that it is immune to parallel paths and stray currents. It is also immune to pilot-to- ground faults, if a pilot conductor is not used. However, the continuity monitor is expensive and complex when compared to an impedance monitor. A pilotless monitor must still be wired into the pilot and grounding contacts of the coupler in order to trip the circuit breakers when disconnecting the coupler. Another problem is that the grounding conductor must be isolated from the coupler shell or the intended monitoring is bypassed. (For metallic shells, a separate grounding conductor must be supplied through a spare coupler contact.)

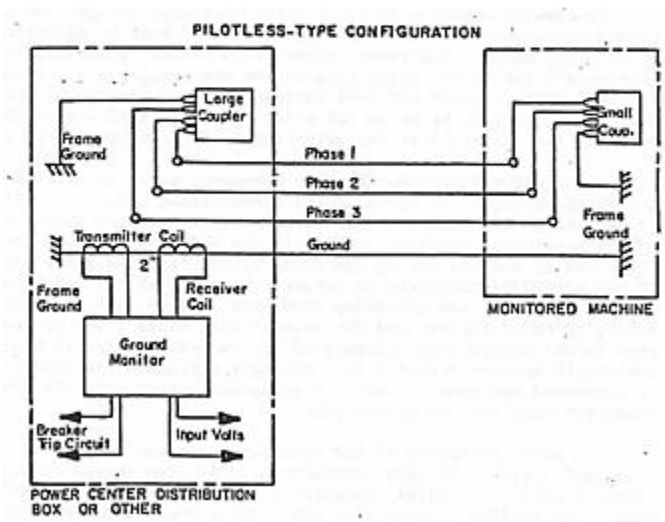


Figure MES 8-3A

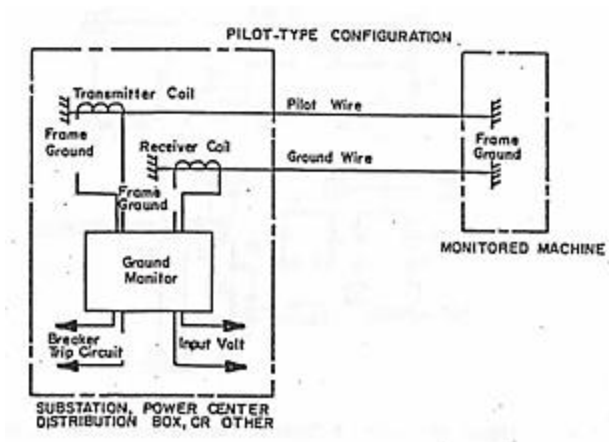


Figure MES 8-3B

2. GROUND-BED RESISTANCE MEASUREMENT

Measurement Method

The accepted technique for determining the resistance to infinite earth of a grounding resistance is called the "fall-of-potential method". Figure MES 8-4A shows a drawing of this arrangement. Three terminals are required: the ground under examination, a potential electrode, and a current electrode. The current electrode is spaced far from the ground system being tested, and the potential electrode is placed at some point on a straight line between the two. The resistance-measuring equipment is operated, and a reading is taken. Here, a known current is passed through the current electrode, the voltage between the potential electrode and ground is measured and the resistance is the ratio V/I. This process is repeated as the potential electrode is moved farther and farther from the grounding electrode, toward the current electrode. A graph is then drawn in which the ground resistance is the ordinate and the distance between the ground and potential electrodes is the abscissa. Figure MES 8-4B shows two typical plots that may result. Curve A was taken with the current electrode at a greater distance than in curve B. The flat portion of curve A is an indication that the current electrode is now far enough away from the grounding system that the mutual effect no longer exists. This is illustrated in Figure MES 8-5 by the "hemispheres of influence" surrounding the ground and current electrodes.

The proper spacing for the measurement probes is based upon hemispherical electrodes, so any actual ground system must first be converted to an equivalent hemisphere before the needed spacing can be determined. This may be approximated by assuring that the equivalent radius is equal to one-half the length of the longest diagonal that can be placed inside the perimeter of the system (that is, 50% of the maximum bed dimension). Figure MES 8-7 shows the proper spacing for both current and potential electrodes for a given equivalent radius of the grounding system. The potential-electrode spacing that yields the true value of ground resistance is equal to about 61.8% of the current-electrode spacing. For large ground systems, it may be impossible to attain the necessary spacing for potential and current electrodes resulting from this technique. In this case, the procedure outlined earlier may still be followed, that is, varying the potential electrode spacing while keeping the current electrode at some fixed spacing as far as possible from the grounding system. The true resistance may then be derived from the resulting graph using one of several available methods.

3. GROUND TEST INSTRUMENTS

Certain precautions should be observed when a ground testing instrument is chosen. A machine that uses de should be avoided because of problems with polarization and electro-osmosis. AC is satisfactory, but a frequency slightly removed from the actual power frequency is preferable so the effects of stray currents can be avoided. On the other hand, if the frequency used is too far removed from the power frequency, erroneous results may occur since ground resistance (impedance) varies with frequency. The leads from the instrument to the electrodes should be spaced as far apart as possible to minimize the effects of mutual inductance and capacitance. In a good instrument, the resistance of current and potential probes is not critical, but inferior equipment will give readings that vary widely depending upon the probe resistance. Great accuracy in measuring earth-ground resistance is not critical because the earth resistance measurement techniques themselves can never be precise or accurate.

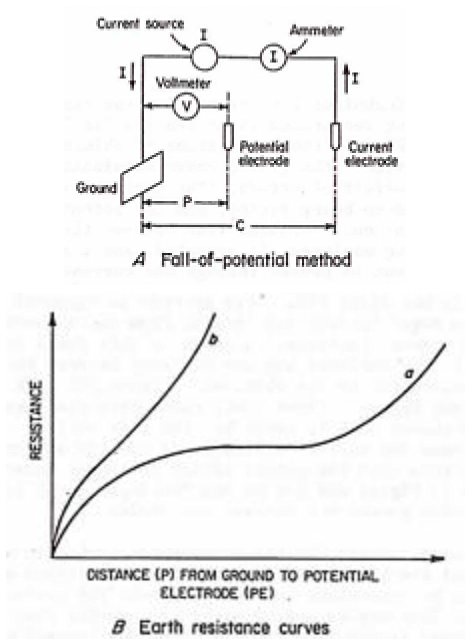


Figure MES 8-4A & B: Measuring Resistance of Grounding Systems

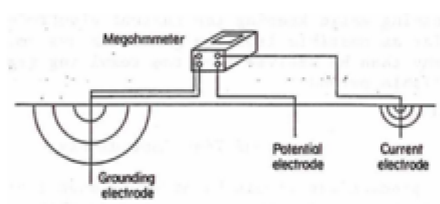


Figure MES 8-5:
Concentric Earth Shells Around Ground Connection Being Tested and Around Current Electrode

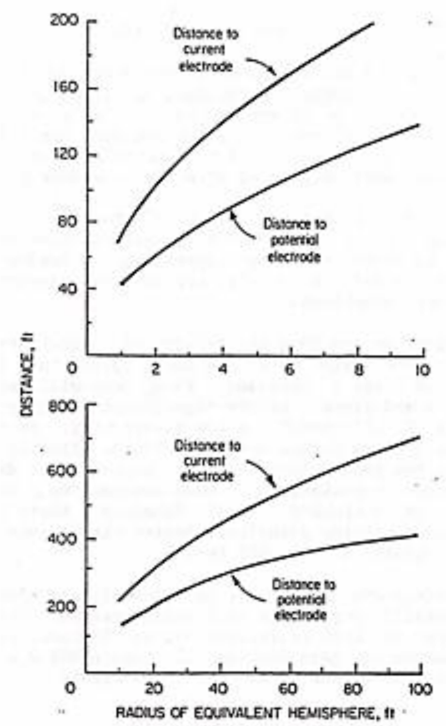


Figure MES 8-6: Correct Spacing of Auxiliary Electrodes to Give True Resistance Within 2.0%

4. GROUND-BED RESISTIVITY

In the discussions on resistance it was pointed out that soil resistivity, p, is an important parameter; specifically, ground-bed resistance is directly proportional to soil resistivity. The resistivity of a material was defined as the resistance in ohms between the opposite faces of a unit cube of that material. The value of resistivity varies widely depending upon the substance being measured:

5. FACTORS AFFECTING RESISTIVITY

Several factors can affect resistivity, and these are generally considered to include:

--Moisture content,
--Dissolved salts,
--Temperature,
--Soil type,
--Grain size and distribution, and
--Location

B GROUNDING
QUESTIONS AND ANSWERS

MES-8 Q-1: Distinguish between a return conductor and a ground conductor.

MES-8 A-1: The unreliability of local ground connections in a mine, such as the machine resting on the bottom or driving rods into the bottom has resulted in the establishment of the practice of running a separate ground wire to each machine frame in addition to the regular return conductor. This ground wire should be connected to a continuous grounding system leading back to the frame of the source of power supply with adequate contact to earth. The ground wire should, also, be brought to earth potential at as many places as possible throughout its length to minimize the hazards of a parallel path being better through a human making contact with the ground wire.

All metallic frames, casings, and coverings of motors, generators, switchboards, and other electrical equipment that may be electrified by failure or insulation should be grounded by this method. A grounding conductor in either a direct current or alternating current system should not be less in current carrying capacity than that of the largest power conductor in the system.

MES-8 Q-2: What is the general practice in providing a return path in coal mining?

MES-8 A-2: It is accepted practice to use trailing cables containing a ground conductor of not less than half the size of the power conductor.Ground connections should be tested frequently to determine their continuity and occasionally to determine their resistance.

MES-8 Q-3: Define solidly grounding.

MES-8 A-3: Solidly grounded systems have the power transformer or generator neutral connected directly to earth ground. This type of grounding is used on the power systems on the surface.

MES-8 Q-4: Define resistance grounding.

MES-8 A-4: Low-resistance grounded systems require that the system neutral be connected to earth ground through a resistance significantly lower than those used in high resistance grounding. The value of this resistor is such that it will allow 50A to 600A current flow.

When high-resistance grounding is employed, the system neutral is connected through a resistance element whose ohmic value would allow a ground fault current greater than system charging current (usually less than 2% of the 3-phase fault current). Advantages include low fault current, low transient overvoltage, and the ability to easily locate the faulted system component. Disadvantages include high initial cost of grounding equipment and necessity of using 100% rated lightning arresters. Mine power systems are high resistance grounded, because of the poor grounding conditions.

MES-8 Q-5: Define diode grounding.

MES-8 A-5: Diode grounding - In mixed (AC and DC) systems such as underground mine power system a diode is used in the DC face equipment grounding. When a diode is used, the center-tapped transformer is removed because the frame is grounded through the diode to the negative conductor. The diode circuit also includes a ground protective device which will interrupt the power if a positive- power-conductor-to-equipment-frame current flow occurs. Diode grounding should ensure good ground continuity since the same wire acts as both DC negative conductor and ground wire. However, a grounding diode protects the DC subsystem against ground faults within the equipment frame only. Faults within trailing cables can still present hazards.

MINE ELECTRICAL SYSTEMS
MISCELLANEOUS QUESTIONS AND ANSWERS

MES Q-1: How often must the mine plan of the electrical system be brought up to date?

MES A-1: At intervals not exceeding six months.

MES Q-2: What are distribution centers used for?

MES A-2: To distribute utilization power to portable mobile or stationary equipment.

MES Q-3: Where must instructions for restoration from electrical shock be posted?
MES A-3: In every generating, transforming, and motor room and at the entrance to the mine.

MES Q-4: Cross bonds shall be installed how far apart on main haulage roads?
MES A-4: 200 feet.

MES Q-5: Explain how the ground-continuity check-circuit operates.
MES A-5: The ground-continuity check-circuit shall continuously monitor the integrity of the neutral circuit leading underground and shall cause the breaker to open when either the ground or pilot check wire is broken.

MES Q-6: What is the minimum scale of the mine plan of the electrical system?
MES A-6: 500 feet/inch

MES Q-7: Under what two conditions shall fuses and automatic breakers be constructed as to effectively interrupt the current?
MES A-7: (1) Short circuit. (2) When the current through them exceeds a pre-determined value for a period of time (overcurrent).

MES Q-8: How wide shall the minimum space be in front of an underground switchboard?
MES A-8: Not less than 3 feet in width.

MES Q-9: What does the law say regarding connections to trolley or feeder wires?
MES A-9: All permanent connections to trolley or feeder circuits shall be made with suitable mechanical connectors. No connection, temporary or permanent, shall be wrapped or tied.

MES Q-10: To what extent shall all joints in insulated wire, after the joint is complete, be re-insulated?
MES A-10: To at least the same extent as the remainder of the wire.

MES Q-11: In what two ways shall all joints in conductors be efficient?
MES A-11: Mechanically and electrically.

MES Q-12: Which employees must be familiar with and competent to carry out the instructions for the restoration from shock?
MES A-12: All employees working in connection with the electrical apparatus.

MES Q-13: What current level shall circuit breakers used to protect feeder circuits be set to trip?

MES A-13: When the current exceeds by more than 50% of the rated capacity of the feeder.

MES Q-14: On main haulage roads, what is the maximum distance between switches or circuit breakers or de-energize trolley or feeder lines?

MES A-14: 2,000 feet.

MES Q-15: What is the maximum underground transmission voltage?

MES A-15: 15,000 volts.

MES Q-16: Explain a high voltage underground transmission system regarding high voltage cables leading and extending underground.

MES A-16: They shall be of the multiple conductor flame resistant type with either a rubber, plastic, or armor sheath meeting the requirements of the BDMS for flame resistance.

MES Q-17: A high voltage underground transmission cable shall be equipped with what?

MES A-17: Metallic shields around each power conductor.

MES Q-18: What type of couplers shall be used on an underground high voltage transmission system?

MES A-18: They shall be of the three-phase type with a full metallic shell, and shall be adequate for the voltage and current expected. All exposed metal on the couplers shall be grounded to the ground conductor in the cable. The coupler shall be constructed so that the ground continuity conductor shall be broken first and the ground conductor shall be broken last when the coupler is being uncoupled.

MES Q-19: How shall a high voltage underground transmission cables be installed at locations where they cross haulageways or travelways, or where equipment must pass over or under the cable?

MES A-19: They shall be either installed in a trench in the roof, protected by some mechanical means or buried at least 12 inches below combustible material and adequately protected from crushing by the weight of equipment passing over it.

MES Q-20: Where shall high voltage cables be installed?

MES A-20: Shall be installed only in intake airways. They may be installed on intake haulageways only with the written approval of the Director of Deep Mine Safety.

MES Q-21: What shall be done with excess cable which is connected and supplying a load?

MES A-21: It shall be coiled, stored on a reel or otherwise stored at a place near the load where it can be protected by "dangering off" the place. Such cable shall not exceed 1,000 feet in length.

MES Q-22: How shall transmission voltage be reduced to utilization voltage?

MES A-22: By a portable transformer or load center of adequate capacity for the equipment powered by it.

MES Q-23: What kind of transformer shall be used in bituminous underground coal mines?

MES A-23: Of the dry type, ventilated, non-ventilated, or sealed, substantially constructed and completely enclosed in a metal case.

MES Q-24: How can the danger of explosion from operating motors be reduced?

MES A-24: By having the current carrying parts completely enclosed in explosive proof enclosures.

MES Q-25: What is the effect of continued overload or under voltage on operating motors?

MES A-25: Heating that will destroy the insulation.

MES Q-26: Tell how temporary connections for D.C. portable or face equipment may be made to trolley systems.

MES A-26: Temporary connections may be made through fused trolley taps.

MES Q-27: On what does the speed of a series wound motor depend?

MES A-27: The counter EMF generated by the armature is a large factor in determining the speed of a series wound motor when carrying a load.

MES Q-28: What is a differential compound-wound motor?

MES A-28: A motor with the series and shunt field connected in such a manner as to oppose each other.

MES Q-29: What is residual magnetism?

MES A-29: The magnetism remaining in the pole after the current is shut off is called "residual magnetism". The residual magnetism plays and important part in the starting of the generator.

MES Q-30: What is a consequent-pole motor?

MES A-30: A motor having two field coils and four poles .

MES Q-31: When does current flow?

MES A-31: Current flow takes place whenever most of the electron movement in a material is in one direction. This movement is from a minus (-) charge to a plus (+) charge, and occurs only as long as a difference of charge exists.

MES Q-32: What controls current flow?

MES A-32: Resistance

MES Q-33: What horsepower will be required to pump 200 gallons of water a vertical distance of 33 feet in one minute? (SHOW WORK)

MES A-33: $\frac{hp = (L\ x\ W)}{33{,}000\ x\ T} = \frac{33\ x\ (200\ x\ 8-1)}{33{,}000\ x\ 1} = \frac{55{,}000}{33{,}000} = 1 - \frac{1}{2}\ hp$

MES Q-34: The demand of a motor is 300 amps when the line voltage is 250 volts. What is the motor rating in horsepower and kilowatts? (SHOW WORK)

MES A-34: $P = E\ x\ I = 250\ x\ 300 = 75{,}000\ Watts$

$$HP\ Rating = \frac{75{,}000}{746} = 100.53\ hp$$

$$KW\ Rating = \frac{75{,}000}{1{,}000} = 75\ KW$$

MES Q-35: List the requirements of the Bituminous Coal Mining Laws of Pennsylvania, Act 339, relative to the installation of disconnect switches, cut-out switches, and branch circuits of direct-current system, such as trolley and feed lines.

MES A-35: Disconnecting switches shall be installed underground in all main direct current power circuits within 500 feet of the bottom of shafts, boreholes or at other places where main power circuits enter the mine. All branch trolley and feeder lines shall be fitted with either a switch, circuit breaker, or section insulator and line switch or some other device that will allow the current to be shut off from such branch headings. Switches or circuit breakers shall be provided on haulage roads to de-energize all trolley and feeder lines at intervals not to exceed 2,000 feet.

MES Q-36: State the Mining Law of Pennsylvania relative to electrically operated hand-held tools.

MES A-36: Electric drills and other electrically operated rotating tools intended to be held in the hands shall be equipped with an integrally mounted electric switch designed to break the circuit when the hand releases the switch.

MES Q-37: State the Mining Law of PA relative to grounding in underground transformer and substation rooms.

MES A-37: All metallic coverings, metal armoring of cables, and the frames and bedplates of generators, transformers, and motors shall be effectively grounded.

MES Q-38: State the Mining Law of PA relative to voltage restrictions on electrical face equipment.

MES A-38: Motors of electrical face equipment shall not be operated at higher than medium voltage, except as approved by the Director of Deep Mine Safety under Section 334 and except those on hand-held tools which shall be restricted to low voltage.

MES Q-39: All electrical face equipment shall be inspected how often by the person in charge of electrical equipment?

MES A-39: All electrical face equipment shall be inspected by the mine electrician or person designated by him at least weekly, and, where necessary, shall then be cleaned and repaired.

MES Q-40: Explain explosion-proof casings or enclosures as defined in the Mining Law of PA.

MES A-40: Those which, when completely filled with a mixture of methane and air, and the same explode, are capable of either entirely confining the products of such explosion within the casing or of so discharging them from the casing that they cannot ignite a mixture of methane and air, combined in proportions most sensitive to ignition and entirely surrounding the points of discharge, and in most intimate proximity therewith.

MES Q-41: Define a mine power center.

MES A-41: A mine power center is a combined transformer and distribution unit and may include a rectifier complete within a metal enclosure from which one or more low voltage or medium voltage power circuits are taken.

MES Q-42: What are the requirements of the law relative to the location of a combined alternating and direct current distribution or load center utilizing a high voltage power supply?

MES A-42: A combined alternating and direct current distribution or load center utilizing a high voltage power supply shall be located on intake air only, shall not be located beyond the last open crosscut and shall not be located closer than 250 feet along the air route to pillar workings .

MES Q-43: Define phase.

MES A-43: Phase is defined as the difference in time between any point on a cycle and the beginning of that cycle.

MES Q-44: Define inductance.

MES A-44: Inductance is that property of an electrical circuit which tends to prevent a change of current.

MES Q-45: Define capacitance.

MES A-45: Capacitance is that property of an electrical circuit which tends to prevent a change of voltage.

MES Q-46: Define impedance.

MES A-46: Impedance is the opposition to the flow of current in an AC circuit and is affected by the resistance, inductance and capacitance of the circuit and the frequency of the applied voltage.

MES Q-47: What are the legal requirements relative to storage battery equipment?

MES A-47: (a) All storage battery equipment and charging stations shall be designed, operated and ventilated so that gas from the batteries will be safely diluted. (b) Storage battery charging stations shall be on a separate split of air. (c) Smoking or the presence of flammable materials is not permitted in any storage battery room or charging stations. Signs to this effect shall be posted in all battery rooms or charging stations. (d) Storage battery operated equipment may be used in face areas of gassy mines when all electrical parts that it is practicable to enclose are enclosed in explosion-proof casings and the batteries are adequately ventilated.

MES Q-48: What is the law relative to Braid Covered Cable?

MES A-48: No power wires or cables having what is commonly termed as weatherproof insulation or insulation consisting of braided covering, which is susceptible to moisture absorption from the outer surface to the conductor shall be installed in any mine.

MES Q-49: What is the law relative to ground detectors?

MES A-49: All underground systems of distribution that are completely insulated from earth shall be equipped with properly installed ground detectors of suitable design, maintained in working condition. The condition of such a system as indicated by the ground detector shall be noted each day by the person in charge of the underground electrical system, or by another competent person, who shall immediately report to the mine foreman the occurrence of a ground.

MES Q-50: If a 50 hp. motor is rated at 500 volts D.C., what is the full load current of this motor?

MES A-50: $Amperes = \frac{hp. x\ 746}{V} = \frac{50\ x\ 746}{500} = 70.46\ Amperes$

MES Q-51: What would be the proper size fuse for this motor?

MES A-51: 1.25 x 70.46 = 88.075 or 90 amp fuse

MES Q-52: If the voltage of a power system is 500, the power transmitted is 500 kw and the allowable voltage drop is 10%, what would be the number of amperes flowing in the circuit and what would be the number of volts lost?

MES A-52: $Watts = kW\ x\ 1{,}000 = 500\ x\ 1{,}000 = 500{,}000$

$Amperes = \frac{Watts}{V} = \frac{500{,}000}{500} = 1{,}000\ Amperes$

$500\ x\ 0.10 = 50\ volts\ lost$

MES Q-53: How is electricity produced from cells and batteries?
MES A-53: By chemical action

MES Q-54: What is an electric cell?
MES A-54: It is the basic source of electricity produced by chemical action.

MES Q-55: What is a battery?
MES A-55: When two or more cells are combined, they form a battery.

MES Q-56: What are the inner parts of a cell or a battery?
MES A-56: Case or container, plates and liquid.

MES Q-57: What is electrolyte?
MES A-57: The liquid in a cell or battery.

MES Q-58: Explain the action of electrolyte in a battery.
MES A-58: The action of the electrolyte in carrying electrons from one plate to the other is actually a chemical reaction between the electrolyte and the two plates. This action changes chemical energy into electrical charges on the plates and terminal.

MES Q-59: What is used for electrolyte?
MES A-59: Acids or salt compounds.

MES Q-60: What causes gas to be omitted from batteries?
MES A-60: As the electrolyte carries electrons, the negative plate is used up and bubbles of gas would be present at the positive terminal caused by chemical action.

MES Q-61: What kind of acid is used in storage batteries?
MES A-61: Sulfuric Acid.

MES Q-62: What means is provided to let gas escape from batteries?
MES A-62: The cap has a vent to allow gas to escape since the cell, in operation, forms gas at the positive plate.

MES Q-63: Why are connectors and terminals of batteries made of lead bars?
MES A-63: Other metals would corrode rapidly due to the acid electrolyte.

MES Q-64: What is the main feature of a storage battery?

MES A-64: A storage battery can furnish large amount of power for a short time and can be recharged.

MES Q-65: How does the maintenance of a battery compare with the maintenance of other electrical equipment you will work with in the mines?

MES A-65: Batteries require more care and maintenance than most of the equipment on which you will work.

MES Q-66: What gas escapes from batteries?

MES A-66: Hydrogen.

MES Q-67: What are the requirements of the law relative to Ground Bed Testing?

MES A-67: Ground bed resistance shall be measured at least every six months and appropriate action taken to assure the maintenance of the lowest possible value of ground resistance. A record of the resistance measurements shall be kept in a book provided for that purpose.

MES Q-68: What are the legal requirements when the potential of the branch circuit exceeds the limits of medium voltage?

MES A-68: When the potential of a branch circuit exceeds the limits of medium voltage, it shall be protected by a circuit breaker, except as otherwise permitted, which would be a suitable load break switch with the ground continuity check circuit so wired that the ground wire or ground continuity conductor or any connection on either wire cannot be broken without interrupting the check circuit unless such break occurs on a branch circuit which has been disconnected.

MES Q-69: Name five good conductors.

MES A-69:

1. Gold
2. Silver
3. Copper
4. Iron
5. Aluminum
6. Carbon

MES Q-70: Name five good insulators.

MES A-70:
1. Air
2. Oil
3. Glass
4. Plastic
5. Wood
6. Rubber
7. Cotton

MES Q-71: What is the requirement of the law relative to utilization voltage cables fitted with couplers?

MES A-71: Utilization voltage cables shall be fitted with plug couplers and provisions made so that cables cannot be uncoupled under load. All plugs and sockets shall be substantially constructed and any exposed metal portions shall be grounded. Couplers shall be constructed so that the ground conductor connection is broken last during uncoupling.

MES Q-72: In what manner shall cables underground be installed?

MES A-72: (1) Where the cables or feed wires, other than trolley wires, in main haulage roads, cannot be kept at least twelve inches from any part of the mine car or locomotive, they shall be specially protected by proper guards. (2) Cables and wires, except trailing or portable cables or bare return cables shall be installed on roof, ribs, walls, or timbers by means of efficient insulators or suitable supports. In no instance shall the method of support damage the cable jacket or armor

MES Q-73: How and where shall fireproof rectifiers be installed?

MES A-73: Section 313 - Fireproof rectifiers and transformer.

(j) A portable rectifier with dry type transformer, except those using pumped tubes or glass bulb mercury arc tubes, or dry type transformer designed for underground use with adequate automatic electrical protection and substantially of fireproof construction, fully metal-clad, which will not be in the same location in excess of one year, may be installed in any intake air current, not beyond the last open crosscut and not closer than two hundred and fifty feet along the air route to pillar workings. The location where such fireproof rectifier or transformer is installed need not be made fireproof with masonry or steel, but shall be equipped with doors, grill work or otherwise to prevent entry or access by unauthorized persons.

MES Q-74: Define zig-zag transformer.

MES A-74: A zig-zag transformer is a three-phase transformer used to provide a neutral point on "delta" systems and is capable of carrying continuously the maximum ground fault current of the system.

MES Q-75: Define wye-connected power system.

MES A-75: A wye-connected power system is a system in which one end of each phase winding of the transformer or AC generators are connected together to form a neutral point and the end of the windings are connected to the phase conductors.

MES Q-76: Define portable trailing cable.

MES A-76: A portable trailing cable is a flexible cable or cord used for connecting mobile, portable or stationary equipment in mines to a trolley system or other external source of electrical energy where permanent mine wiring is prohibited or is impractical.

MES Q-77: Define portable flame-resistant cable.

MES A-77: A portable flame-resistant cable is a portable cable that has met the Bureau of Deep Mine Safety requirements for flame resistance and has been assigned an approval number, a "P" number embossed or indented on the jacket at intervals not to exceed twelve feet.

MES Q-78: What is the purpose of a transformer?

MES A-78: The transformer serves numerous purposes:
(1)To raise the voltage
(2) To lower the voltage
(3)Isolation
(4) Establish a neutral

MES Q-79: Name the 3 principle parts of the transformer.

MES A-79: The 3 principle parts of the transformer are:
(1) An iron core (or in case of small transformer - air)
(2) The primary winding
(3) The secondary winding

MES Q-80: What physical connection is there between the primary and secondary coils of a transformer?
MES A-80: The primary and secondary coils of a transformer are not connected to each other in any way. They are electrically independent but are magnetically interconnected by alternating flux.

MES Q-81: What is an isolation transformer?
MES A-81: Any two-winding transformer is known as an isolation transformer because there are no electrical connections between the two windings. It provides isolation between the two circuits.

MES Q-82: According to law, what electrical equipment is required to be examined daily by the mine electrician?
MES A-82: All electrical equipment and other machinery under his jurisdiction shall be examined daily.

MES Q-83: By whom shall this examination be made?
MES A-83: It shall be the duty of the mine foreman or assistant mine foreman or any authorized person designated by the mine foreman.

MES Q-84: In what book must the daily electrical examination report be entered?
MES A-84: In the assistant mine foreman's book.

MES Q-85: How often must the daily electrical examination report be made?
MES A-85: Daily

MES Q-86: How often does the mine electrician make and sign a report in a book provided for that purpose, stating the condition of electrical equipment and other machinery in the mine?
MES A-86: Weekly

MES Q-87: Who is required to countersign the above report?
MES A-87: Mine foreman.

MES Q-88: What is the distance allowed between trolley hangers?
MES A-88: They shall not be more than twenty-four (24) feet apart.

MES Q-89: How shall branch circuits be de-energized?
MES A-89: By trolley switches, circuit breakers, section insulators, and line switches or some other device that will allow the current to be shut off.

MES Q-90: Where is trolley guarding required?
MES A-90: At all landings and partings or other places where men are required to regularly work or pass under trolley or other bare wires which are placed less than six and one-half (6-1/2)feet above the top of the rail.

MES Q-91: How far below trolley wire shall guard boards be extended?
MES A-91: Not less than two (2) inches.

MES Q-92: Where shall haulage locomotives be used?
MES A-92: On intake air only.

MES Q-93: How far shall trolley and feeder wires be kept from pillar workings?
MES A-93: At least one hundred and fifty (150) feet.

MES Q-94: What protection shall be used at the outby end of trailing cables?
MES A-94: Circuit breakers equipped with tripping devices which will function on overload, phase fault, and ground fault shall be used at the outby end of trailing cables.

MES Q-95: If an incombustible structure used to house electrical equipment in a mine is constructed of steel plates, what is the allowable minimum thickness of the plates?
MES A-95: 1/8 inch

MES Q-96: Define the term "Electrical Inspector."
MES A-96: A qualified and certified employee of the Commonwealth who performs electrical inspections in and about the bituminous coal mines.

MES Q-97: What is the effect of distance upon direct current voltage?
MES A-97: The voltage is decreased by line drop due to the resistance of the conductors.

MES Q-98: What is the effect upon motors from a drop in voltage?
MES A-98: Inefficient operation, abnormal heating, possible burnouts, and decreased speed.

MES Q-99: How can excessive line voltage loss be avoided?
MES A-99: By ample current-carrying capacity in the conductors, by adequate bonding and by locating converting equipment near the point of operation.

MES Q-100: What are the requirements of the Law relative to joints in conductors?

MES A-100: All joints in conductors shall be mechanically and electrically efficient. Suitable connectors or screw clamps shall be used. All joints in insulated wire shall, after the joint is complete, be reinsulated to at least the same extent as the remainder of the wire.

MES Q-101: What are the requirements of the Law relative to cables entering fittings?

MES A-101: The exposed ends of cables, where they enter fittings of any description, shall be protected and finished off so that moisture cannot enter the cable or the insulating material. Where unarmored cables or wires pass through metal frames or into boxes or motor casings, the holes shall be substantially bushed with insulating bushings, and where necessary or required with gas-tight bushings which cannot readily become displaced.

MES Q-102: What are the requirements of the law relative to electric wires and cables in complete portions of conveyor entries?

MES A-102: All electric wires or electric cables in completed portions of conveyor entries shall be carried on insulators.

MES Q-103: What are the requirements of the law relative to cables in rooms, pillars, or other working places?

MES A-103: Electric cables constantly kept in rooms or pillars or other working places shall be carried on suitable supports to within seventy feet of the face of each working place.

MES Q-104: State the legal requirements for electrical installation in dusty locations in tipples and cleaning plants.

MES A-104: In dusty locations, electric motors, switches and controls shall be of dust-tight housings or enclosures. Open-type motors, switches or controls in use at the effective date of this act in tipples and cleaning plants in dusty locations may be continued in use until such dust-tight equipment can be procured, or until they can be provided with reasonably dust-tight housings or enclosures.

MES Q-105: The demand of a motor is 300 amps when the line voltage is 250 volts. What is the motor rating in horsepower and kilowatts?

MES A-105:

$$P = E\,x\,I = 250\,x\,300 = 75{,}000\ watts$$

$$hp\ rating = 75{,}000/746 = 100.53\ hp$$

$$kW\ rating = 75{,}000/1{,}000 = 75\ kW$$

MES Q-106: A mining machine undercutting in bituminous coal takes 160 amperes at 230 volts. If the place is 20 feet wide and the undercut is 10 feet deep, find (a) the watt hours consumed per place and (b) the watt hours consumed per square foot of undercut if the machine requires 10 minutes to cut the place.

MES A-106:

$$W = E_1 = 230 \; x \; 160 = 36{,}800 \; watts$$

$$\frac{36{,}800}{\frac{1}{6} hours \; or \; 10 \; minutes} = 6133.3 \; watt \; hours/place$$

$$20 \; x \; 10 = 200 \; square \; feet \; cut$$

$$\frac{6133.3}{200} = 30.7 \; watt - hours \; per \; square \; foot$$

MES Q-107: How shall storage-battery-charging stations be ventilated?
MES A-107: By separate split of air.

MES Q-108: What signs are to be posted in all battery rooms and charging stations?
MES A-108: No Smoking and No Inflammable Materials Permitted.

MES Q-109: Telephone service or equivalent two-way communication facilities shall be provided if the working sections are more than how far from the main portal?
MES A-109: One thousand five hundred (1,500) feet.

MES Q-110: Where and how shall telephone lines other than cables be installed?
MES A-110: Installed on opposite side from power or trolley wires, and shall be carried on insulators. Where they cross power or trolley wires, they shall be adequately insulated.

MES Q-111: In the face areas of a gassy mine the potential used for signal purpose shall not exceed how many volts?
MES A-111: 24 volts

MES Q-112: What is the maximum potential for signal circuit not including haulage block signal system?
MES A-112: 125 volts

MES Q-113: What is the voltage restriction on electrical face equipment?
MES A-113: Medium voltage except by 334 commission.

MES Q-114: What is the maximum number of temporary splices that can be made in a trailing cable?
MES A-114: 5

MES Q-115: How far shall splices in trailing cables be kept away from machines that do not have cable reels?
MES A-115: 50 feet from the machine

MES Q-116: What type of trailing cable shall be used?
MES A-116: Approved flame-resistant

MES Q-117: Each trailing cable in use shall be examined how often by the machine operator?
MES A-117: Beginning of each shift

MES Q-118: In a gassy mine how long can a piece of electrical equipment be operated at the face before it is stopped and tests made for explosive gas and other hazards?
MES A-118: 1/3 hour

MES Q-119: Open type trolley locomotives can be operated in gassy mines under what conditions?
MES A-119: Those ventilated by fresh intake air.

MES Q-120: How shall a substation in a mine be ventilated?
MES A-120: By a separate split of air.

MES Q-121: At what mines must a mine electrician be employed?
MES A-121: At every mine where electricity is used underground.

MES Q-122: Whose duty is it to see that the electrical requirements of the mining law are observed?
MES A-122: Mine foreman and superintendent.

MES Q-123: How often shall the electrical equipment used in gassy mines be inspected by the Electrical Inspector?
MES A-123: At least twice a year.

MES Q-124: The Electrical Inspector's Report of Inspection shall remain posted at the mine for how long a period?
MES A-124: One year

MES Q-125: What is a "megger"?
MES A-125: A device to measure electrical resistance.

MES Q-126: What instrument is used to measure current?
MES A-126: Ammeter

MES Q-127: What instrument is used to measure voltage?
MES A-127: Voltmeter

MES Q-128: What is the law of resistance as to length?
MES A-128: The resistance of a conductor is directly proportional to its length.

MES Q-129: What is the law of resistance as to area?
MES A-129: The resistance of a conductor is inversely proportional to its cross-sectional area or to the square of its diameter.

MES Q-130: An additional lightning arrestor shall be installed if the distance from the generator station to the point where the line enters the mine exceeds how many feet?
MES A-130: 500 feet

MES Q-131: In all underground systems of distribution that are completely insulated from each, how often shall the condition of a ground detector system be noted by the person in charge of the underground electrical system?
MES A-131: Each day

MES Q-132: Each power circuit leading underground shall be provided with a suitable ammeter if it is in excess of how many kilowatts?
MES A-132: 50 kilowatts

MES Q-133: Cable and feed wires other than trolley wires on main haulage roads shall be protected by proper guards if they cannot be kept how far from mine cars or locomotives?
MES A-133: 12 inches

MES Q-134: A branch circuit shall be protected by a circuit breaker if its potential exceeds what voltage?
MES A-134: 1000 volts

MES Q-135: Electrical equipment containing flammable material shall be kept how far from the door in an underground substation?
MES A-135: 8 feet

MES Q-136: What is the maximum amount of flammable liquid that may be contained in a transformer, circuit breaker, or other device in an underground substation?
MES A-136: 20 gallons

MES Q-137: What shall be kept at all electrical stations or transformer rooms for fire protection?
MES A-137: Rock dust or fire extinguishers.

MES Q-138: If there are any high voltage connections at the back of an underground switchboard, the space in back of the switchboard shall be how wide?
MES A-138: Three (3) feet.

MES Q-139: Where the supply is at a voltage exceeding the limits of medium voltage, there shall be no live metal work on the front of the main switchboard within how close to the floor?
MES A-139: Seven (7) feet.

TABLE OF CONTENTS

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 1

ENGINE FUNDAMENTALS

Diesel Section 1

Diesel Engine Fundamentals

Perhaps the best place to start when talking about diesel engine fundamentals would be with the invention of the compression ignition engine. The invention of the engine is accredited to Dr. Rudolf Diesel. Although it is really a compression ignition engine it is referred to as a diesel engine. Below are some interesting facts about the inventor and the engine:

A. INVENTION OF THE DIESEL ENGINE

- The first engine ran on its own power in 1893 and Dr. Diesel received a patent for the engine in 1895.
- The engine produced a mechanical theoretical efficiency of 75 %. At that time the steam engine only gave a 10% efficiency return on fuel.
- Dr. Diesel's first attempted to operate the engine by injecting powdered coal into the combustion chamber but the resulting explosion almost killed him; he next attempted bio-mass fuels and finally settled on peanut oil. Looking back, Dr. Diesel was quite a visionary. Today we can use powdered coal to operate these engines. The process is called CTL or coal to liquids. The powdered coal is converted in to a liquid and is a good diesel fuel. However it is more expensive than regular petroleum based diesel fuel. Thus it is not used as much. Also the peanut oil that Dr. Diesel did settle on is known today as Bio diesel. Today we can make diesel fuel from many organic substances, but we label all of these as Bio diesel fuels. Interestingly the fuels he tried over 100 years ago are being refined and used in growing quantities today.

Source: John Stafford - webmaster of www.diesel-generator-central.com
Wikipedia - www.wikipedia.org , Diesel / Brayton Cycles ASEN 3113. Diesel Cycle Invented by Rudolf Christian Karl Diesel in 1893.

B. SPARK IGNITED ENGINES VS. COMPRESSION IGNITION ENGINES

The compression ignition (diesel) engine is unlike the spark ignited (SI) engine in many ways. First the SI engine mixes the fuel and air in the intake manifold. This is known as a homogenous mixture, which is then injected then into the cylinder. The mixture is then compressed and ignited by a spark plug which results in an explosion in the combustion chamber. This is why the SI engines are capable of operating at much higher RPM's than a diesel engine. However it is also why the SI engine is much less efficient than diesel engines. How does the diesel engine differ? First, on the intake stroke of a diesel engine, only air is drawn into the cylinder. Fuel is not added at this time. The air is then compressed, which in turn heats the air above the auto ignition temperature of the fuel. It is at this point that the fuel is injected, which starts what is known as a slow burn instead of an explosion. This slow burn is why the diesel engine is much more efficient that the SI engine. It is also why most diesel engines operate between 2000 and 3000 RPM. Because the fuel is not injected into the cylinder of a diesel engine until the air has been compressed and heated above the auto ignition temperature of the fuel, air intake system maintenance is critical on a diesel engine. If you restrict the air going into the cylinders in any way you will alter the fuel air ratio, this will result in an over fueled condition and the engine's carbon monoxide (CO) emissions will rise.

The compression ratios on the diesel engine and the SI engine differ greatly. First, the SI engine in use today will use compression ration ranging between 7- 10: 1 while the diesel engine compression ratio is normally 17-22: 1. Because of these high ratios the diesel engine must be built stronger than the SI engine. It is not unusual to have diesel engines running with 20,000 to 25,000 hours of run time and still going strong. This is also why a diesel engine will carry more oil in the crankcase than the SI engine. Most SI engines will require about 5 to 6 quarts of engine oil, while a diesel engine will usually require between 3 to 5 gallons of engine oil. It must be noted because of these high ratios, the diesel engine will have more engine oil blow by, which means that some engine oil will slip past the piston rings and be burned in the combustion chamber. This is another way that carbon monoxide emissions can raise in a given engine. Even though the diesel engine is designed to burn any number of oils, engine oil will not burn as efficiently as diesel fuel. If you have to add oil at an increasing rate your engine CO emissions will raise. As the engine continues to wear it will eventually have to be taken out of service for high CO emissions.

Let's next look at "FLASHPOINT". Flashpoint is the temperature at which a liquid will give off enough vapor that it can be easily ignited by a spark or a flame. Any fuel that has a flashpoint of less than 100° F is designated as a flammable liquid and cannot be used in underground bituminous coal mines. Liquids which have a flashpoint of greater than 100° F are classed as combustibles. This is the class that diesel fuel falls into. Diesel fuel has a flashpoint of about 125° F which means that it is not a flammable liquid and thus can be used in underground coal mines. However, if you were to spill diesel fuel on hot exhaust components it would exceed its flashpoint and there would be a chance that the vapors could be ignited by a spark or flame. Remember if you exceed the flashpoint of a liquid this does not mean that the liquid will ignite. Flashpoint is not the auto ignition temperature of a liquid. It simply means that you can now ignite the vapors. Prior to achieving the flashpoint, the liquid will not burn.

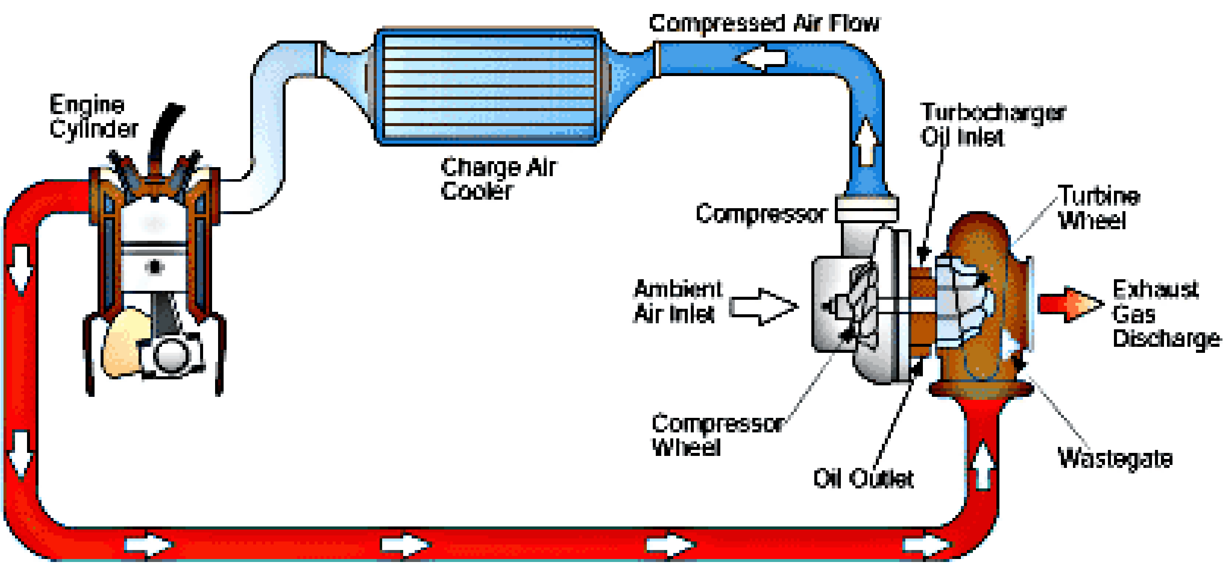
The diesel engine is a lean burn engine (which means it cannot use all of the oxygen that is supplied to the engine). This means that there is always unused oxygen is the exhaust stream of the diesel engine. If you were to measure the oxygen content in the exhaust stream of a diesel engine in our region, while idling with no load on the engine, you would find the oxygen content to be about 16.5 to 18%. Even at full throttle and maximum load there would still be about 8 to 10 % oxygen in the exhaust stream. This lean burn characteristic is the reason that the oxidation catalyst works very well on diesel engines to control diesel emissions.

Diesel engines also differ from the SI engine in that they will not over-rev as SI engines do. Let's take a look at your car engine. If you were to start the engine and put the accelerator to the floor you would find that the engine RPM's would climb to about 4000. The rpm's would then start to drop off and pick back up. The engine would continue to do this until you release the accelerator. This is caused by what is known as a throttle limiter. If it were not for this limiter, the engine would continue to gain RPM's until it destroyed itself. The diesel engine is not like that. You could start a diesel engine and hold the accelerator to the floor and the engine would not continue to produce rpm's. This is a function of the fuel governor which governs the amount of fuel that is metered into the engine. So if you were to hold the accelerator to the floor all day long the engine would only run at its governed limit. By the way a diesel engine should be operated at high rpm's for most efficiency.

C. MAJOR ENGINE COMPONENTS

1. TURBOCHARGER

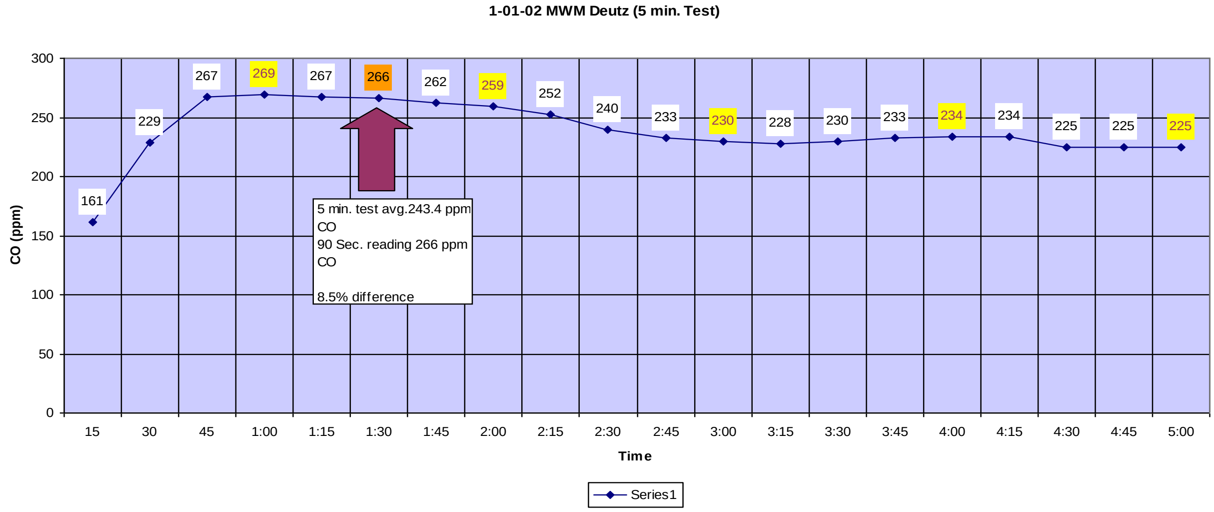
Simply stated, the turbocharger is a dual vane pump. The heat of the exhaust will cause the exhaust vane of the pump to spin. Once this occurs the cold side or intake side of the pump will begin to compress air and feed it into the engine. However, the turbo does not fully function instantly. There is a short delay from the time the throttle is depressed until the turbocharger is fully functioning. This time delay is called turbo lag or turbo response time. Here is the definition "Turbo Lag" is the time delay between injecting fuel to accelerate and delivering air to the intake manifold by the turbocharger. This phenomenon may cause black smoke emissions in some turbocharged diesel engines during acceleration.



Source: http://auto.howstuffworks.com/turbo2.htm.

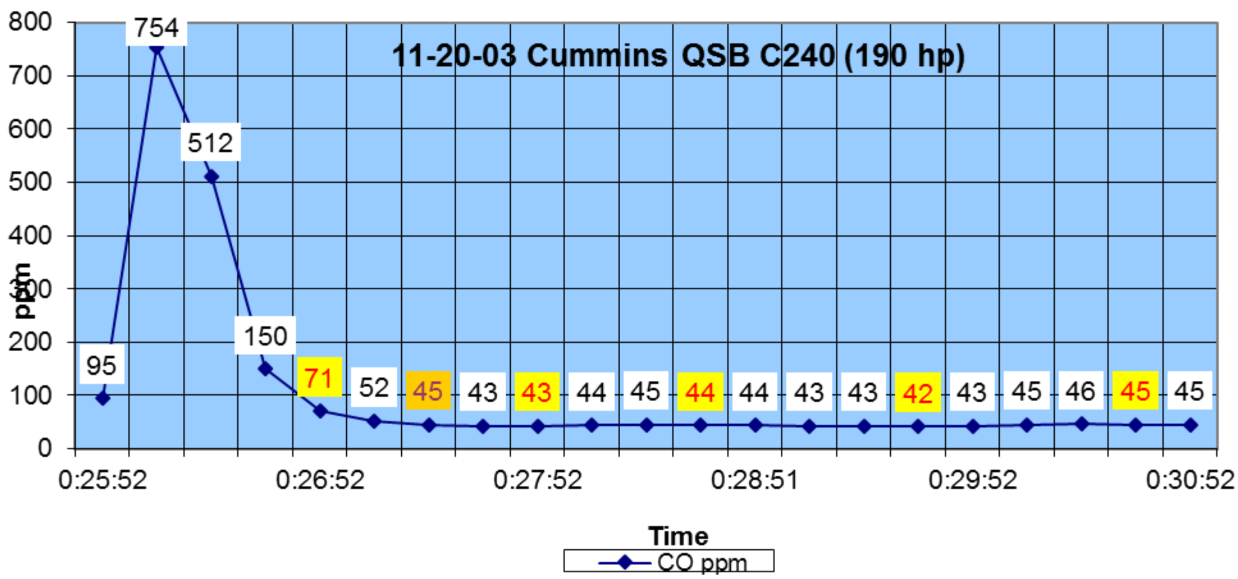
Turbocharger failure is one of the mining industry's most frequent problems, especially in Pennsylvania. This is caused by the fact that our underground diesel law requires surface temperature control of all exhaust components. This control is usually achieved by a polyimide coating or a pipe wrap over the turbocharger. These coatings will maintain the surface temperature of the turbocharger below the required $302° F$, however because of these coating the turbocharger never gets to cool down properly, which causes coking and eventually ruins the turbocharger. Let's look at the function of the turbocharger.

The charts below show the carbon monoxide (CO) emissions of a Naturally Aspirated (NA) engine vs. a turbo aspirated engine.



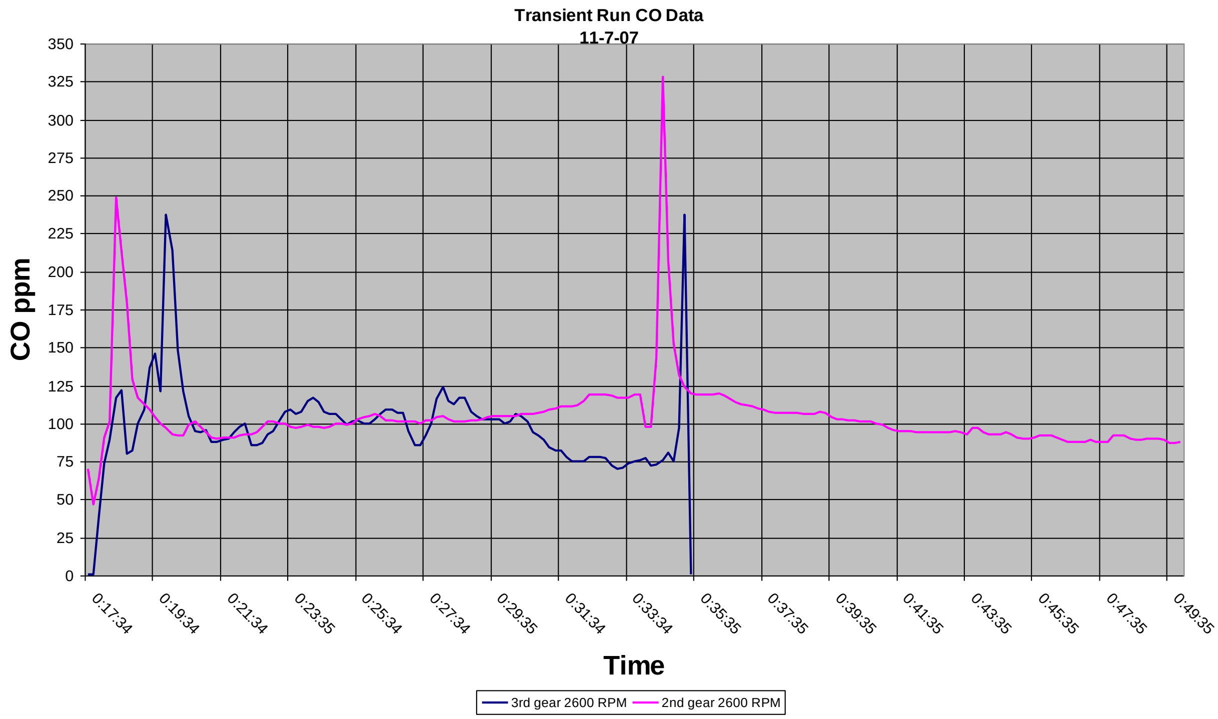
Source: Davis Training & Consulting

In the chart above, you will see a typical CO emissions trend for an NA engine. The engine was operated for 5 minutes. The blue line represents the CO emissions of the engine. The nodules on the blue line are at 10 second intervals. If you look at the chart you will notice that the CO emission for this engine was 191 ppm at low idle. Once the engine was put into load and the throttle fully depressed the CO emissions increased to 269 ppm in about forty seconds. You will then notice that the CO slowly decreased and eventually starts to level out at about 230 to 235 ppm. As stated, this type of a CO trend is typical of a Non- turbocharged engine.



Source: Davis Training & Consulting

This chart illustrates what turbo lag is. The engine was operated for 5 minutes. The blue line represents the CO emissions of the engine. The nodules on the blue line are at 10 second intervals. You will notice that the engine CO emissions output at low idle are 95 ppm. But once the engine is put into load and the throttle is fully depressed, the CO output increased to 754 ppm within 10 seconds. This is the turbo lag or turbo response time. This is the time that you have increased the fuel going into the engine but the engine does not have enough air to burn all of the fuel efficiently. This is turbo lag. You will then notice what happens to the CO emissions after the turbo starts to produce air and deliver it to the engine. After about 45 to 50 seconds, the entire turbo lag has been cleared and the fuel air ratio of the engine is being met, which produces a very lean burn. You, as an underground diesel operator, should remember this phenomenon as you are traveling throughout the mine. Remember, any time you decrease the throttle on a diesel engine you remove some heat from the turbo. Once you increase the throttle again, you will have to go through some turbo lag. It may not be as pronounced as the chart which was total stop to full throttle but there will be some lag every time you are on the throttle and off the throttle. If you want the engine to run more efficiently, keep the equipment in a lower gear and operate the engine in a higher rpm mode. This will minimize the turbo lag. This is the way you should always operate a diesel engine. The following chart will further explain this.



Source: Davis Training & Consulting

The above chart was captured in an underground mine that used rail mounted diesel equipment. The two lines on the chart represent the CO emissions from the engine. The equipment was operated over the same section of rail for both lines. The blue line is showing the equipment being operated in high gear. The pink line shows the equipment being operated in a lower gear. The very high peaks for both the blue and pink lines are turbo lag from dead stop to full throttle. Looking at the blue line first, you will notice that the line is jagged with a lot of peaks and valleys. This is being caused by turbo lag as the equipment is being operated. Every time you see a down trend this is when the operator decreased the throttle. The up-trend is when the operator increased throttle. Each one of the peaks represents the lag of the turbo. With the pink line, you will notice that the trend is more even with less peaks and valleys. This is a result of having the equipment in a lower gear and not having to drop off the throttle as much during equipment operation. If you operate the engine in a lower gear with higher rpm, the emissions output of the engine will be reduced.

Turbochargers tend to fail at a higher rate in the mining industry than in other industries. This is due to the coking of the turbocharger. Coking happens when the turbocharger is not allowed to cool down properly. Turbochargers are cooled by engine oil, and in many cases, engine coolant as well. Turbochargers get very hot when making boost. When you shut the engine down the oil and coolant stop flowing. If you shut the engine down when the turbo is hot, the oil can burn and build up in the unit, this is known as coking. This will eventually cause it to leak oil (this is the most common turbocharger problem). It is a good idea to let the engine idle for at least 2 minutes after any time you ran under boost. This will cool the turbo down and help prevent coking. The law in Pennsylvania requires no unnecessary idling of the equipment but, this idle time is necessary. In coking, it is not the bearings that get coked, but it is the oil in the bearings that gets coked. Oil, be it synthetic or not, has a temperature where it starts coming apart. For example, when you put cooking oil in a frying pan and apply too much heat to it. It turns black, and adheres to the frying pan like cement. That is coking. Think of the same thing happening to needle bearings in a turbocharger. Coking does not happen immediately, but over time, by not allowing the turbocharger to cool down properly. Remember allow the turbocharger to cool prior to shutting off the engine.

2. FUEL DELIVERY SYSTEM

NOTE: All adjustments to the fuel delivery system must be made by an authorized representative of the engine OEM. This is a requirement of the Pennsylvania diesel law. The fuel delivery system consists of the fuel injection pump, fuel injectors, and fuel governor. You cannot make any adjustments to these parts unless you are authorized by the engine OEM.

3. AIR INTAKE SYSTEM

As we know, the diesel engine operates on a fuel air ratio (AFR).An AFR is nothing more that the mass of fuel vs. the mass of air entering the engine. As stated, if you restrict the amount of air entering a diesel engine in any way you will induce an over fueled condition in the engine. This will happen because the fuel in a diesel engine is metered by the governor. As you increase the throttle, the governor increases the amount of fuel going into the engine. However, the governor can only assume that the air flow is unrestricted and proper. There is no mechanism that will adjust the fuel to match the available air as the oxygen sensor does in a spark ignited engine. So, any restricting of the air flow will cause the engine to be over fueled. This restriction can be caused by a clogged air filter, leaking connections on the piping, crimped or crushed hoses or piping, a leaking or restricted charge air cooler and a turbocharger that is not producing the proper amount of air. This is why all intake system components should be checked frequently. The Pennsylvania law requires the operator to check the intake restriction gauge prior to using a piece of diesel powered equipment. The operator should become familiar with what the normal restriction is for his or her equipment. If the intake restriction is either higher or lower that normal the operator should have the system checked before using the equipment. The following quote comes from the Mechanics Diesel Maintenance Manual by Sean McGinn of Quebec, Canada **“In a mining application the intake system becomes the most critical engine**

system affecting exhaust emissions. Problems associated with intake air are magnified in every other engine system's performance." I absolutely agree with this assessment**.**

D. PROGRESSION OF ENGINE TECHNOLOGY

Since the inception of the Pennsylvania underground diesel law the diesel engine has undergone many technological improvements. This law was negotiated in 1995 and passed by the legislature in October of 1996, and became law on February, 17 1997. Most of the diesel engines that were around when this law was being negotiated are no longer being used in the mining industry. The Environmental Protection Agency (EPA) got involved with the emissions of the diesel engine and has since required diesel engines to be more efficient and less polluting. What was the make-up of a typical diesel engine in use in the mining industry in 1996:

First these engines were indirect injection engines (IDI); this means that the fuel is not injected directly into the main combustion chamber. Instead the fuel is injected into a pre-chamber or side car built into the cylinder. This pre-chamber is where the fuel started to burn. It then moved out into the main portion of the cylinder. This is what is known as indirect injection. An IDI engine has very low noise characteristic but is less efficient than today's direct injected (DI) engines where the fuel is injected directly into the cylinder. There is no need for a pre-chamber on today's diesels.

Most engines of this time period were naturally aspirated (NA) which means that when the intake valve opens and the piston drops down air is drawn into the cylinder naturally by the vacuum of the piston. The only problem is that you cannot get enough air into the cylinder to operate the engine on a lean AFR. Today this is taken care of with turbo charging, which allows for more air to be brought into the cylinder, which in turn leans the AFR.

The older engines operated on what was thought to be high fuel injection pressures, injection pressures in 1995 ranged from 3500 to 5000 psi. While these injection pressures sound high, they did not allow the complete atomization of the fuel. Today, fuel injection pressures are somewhere between 22,000 and 28,000 psi. These higher pressures allow for a more complete atomization of the fuel and. when coupled with the added air of the turbocharger, makes for a much more efficient burn of the fuel air mixture.

The older engines relied on mechanical injection of the fuel, which means that as the lobe on the cam shaft lifted the push rod the rod triggered the injection of the fuel. This type of injection causes the diesel engine to be very inefficient during idle and light load conditions. We still have many engines in the mining industry that are mechanically injected. But the newer engines use an electronic control unit (ECU). This small computer now feeds the fuel into the engine and can compensate somewhat for idle and light load conditions. However, the diesel engine is still less efficient in the idle and light load condition. That is why you should operate diesel equipment in a lower gear and higher rpm for as long as you can. This will insure that the engine is operating more efficiently.

What do the technology improvements made over the past twenty years mean to the coal miner? Let's take a look at the emissions output of two engines. One that was operating 20 years ago and one that has incorporated all of the above improvements. To do this we will use the Mine Safety and Health Administration (MSHA) approval data for two engines

First let's look at the old technology engine. For this we will look at the Caterpillar 3304 PCNA engine that produced 100hp @2200 rpm. This engine was;

- Indirect injected (IDI)
- Naturally aspirated (NA)
- 5000 psi fuel injection
- Mechanically injected

As you can see this engine incorporates all of the old technology we discussed. When MSHA approves an engine for use in underground coal they use a test called an ISO 8178-1, this is a steady state test that operates the engine in 8 different modes. The emissions output, both particulate and gaseous, of the engine are measured. The vent plate for the engine is set as well as a weighted average in grams per hour (g/h) for the particulate output. When this engine was tested its weighted particulate output was 25.49 g/h. and the CO baseline emissions were 330 ppm.

Now let's look at an engine that incorporates all of the above new technology. For this we will use the Mercedes OM904LA engine that produces 100 hp @ 2200 RPM. Nothing here has changed except the 904 is:

- Direct injected (DI)
- Turbo aspirated instead of NA
- Injection pressures of about 22,000 psi
- Electronic fuel injection (ECU)

When this engine went through MSHA testing its weighted particulate average was 4.14 g/h and its CO baseline was between 80 and 90 ppm. The use of the new technology removed approximately 20 g/h of particulate and approximately 250 ppm CO. These engine improvements have made the mine atmosphere much cleaner than it was twenty years ago. However, if you do not maintain the engine you will not reap the benefits of this new technology.

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TABLE OF CONTENTS

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 2

DIESEL EMISSIONS AND FACTORS THAT EFFECT EMISSIONS

Diesel Section 2

Diesel Emissions and Factors that Effect Emissions

When talking about diesel exhaust emissions we must note that diesel exhaust emissions are made up of a plethora of compounds. Gases as well as solids, many of these are released in trace forms. This study guide will look at only the major constituents of diesel exhaust.

Diesel exhaust is broken down into two classifications which are: Gaseous, which is self-explanatory, and the solid portion known as Diesel Particulate Matter (DPM). MSHA regulations at 30 CFR 7 define DPM as "any material collected on a specified filter medium after diluting exhaust gases with clean, filtered air at a temperature of #125 °F (52 °C), as measured at a point immediately upstream of the primary filter. This material is primarily carbon, condensed HC, sulfates, and associated water." In other words, DPM is any material suspended in the exhaust stream in the form of a minute solid particle or liquid droplets.

A. GASEOUS DIESEL EMISSIONS:

- Hydrocarbons
- Oxides of Sulfur (Sulfates)
- Carbon dioxide (CO_2)
- Carbon Monoxide (CO)
- Nitric Oxide (NO)
- Nitrogen Dioxide (NO_2)

1. **Hydrocarbons**: Hydrocarbons are present in the diesel exhaust stream as both a gas and a particulate. Hydrocarbons are formed when the fuel has not burned completely. This formation will usually happen in one of three ways. 1) If the air fuel mixture is too lean it will not fully propagate the combustion process and some fuel will not be burned and escape the combustion process. 2) If the air fuel mixture is too rich and there is not enough air to fully combust the mixture, unspent fuel will escape, and 3) The air and fuel not being mixed properly in the combustion chamber which in turn will leave pockets of lean and rich fuel areas in the combustion chamber, which will not burn completely and some unspent fuel will escape the combustion chamber. Every diesel engine will produce a certain amount of hydrocarbons, but you can minimize the production of hydrocarbons by performing proper engine maintenance.

- Three ways hydrocarbons are produced in an engine:

  1) The fuel air mixture is too lean. This condition can be minimized by making sure that the injection timing of your engine is correct. If the injection timing is overly advanced this will cause a very lean AFR and the result would be excessive hydrocarbon production. This would show up in an emissions test as higher than usual NOx readings. So if the NOx reading for a given engine are higher than usual, have the injection timing checked.

  2) If the air fuel mixture is too rich the engine is over fueled. Not only will this cause excessive hydrocarbon but the CO emissions will increase. This condition is not as easy

to explain. There can be many reasons for this over fueling such as intake air restriction, turbocharger problems, contaminated fuel and governor problems to mention a few .

3) The fuel air mixture is not being mixed well. This can easily be caused by a clogged or misfiring injector which does not atomize the fuel properly and creates larger than normal fuel droplets. These larger droplets of fuel cannot burn completely; therefore hydrocarbons are formed and will escape the combustion chamber.

- The scientific community has many names for this portion of diesel exhaust. Some of the hydrocarbons fall into a category called the soluble organic fraction of fuel (SOF) and some fall into what is known as volatile organic carbon (VOC) and there are many others. It might be easier to class the full portion of hydrocarbons, which is nothing but unspent fuel and lube oil as wet volatiles. (It is noteworthy to say that the occupational Health and Safety Administration (OSHA) does not have a personal exposure limit (PEL) for hydrocarbon.

2. **Oxides of Sulfur**: These compounds are released from the exhaust stream in trace quantities and as such do not pose a health problem for our miners. Sulfates are produced from the sulfur in the fuel. About 20 years ago when this law was passed the sulfur content of off road fuel could be as high as 2500 ppm by volume weight. At this concentration the burned sulfur would combine with condensate in the exhaust stream and act as a weak sulfuric acid on the exhaust piping. At that time it was not unusual to see stainless steel exhaust piping installed on many diesel engines to combat this. Since then, the sulfur content of the fuel has been dramatically reduced. Today we have ultra-low sulfur diesel fuel that by EPA regulation cannot have more than 15 ppm sulfur by volume weight.

Where sulfates once caused some maintenance issues with the exhaust piping, this is no longer the case. It stands to reason that if there is no sulfur in the fuel there will be no sulfate production. Sulfur in the fuel was also used as a fuel system lubricant for the seals, o-rings and high pressure pumps in the fuel system. Also of note: sulfur was taken out of the fuel to allow the use of diesel emissions after-treatment systems such as oxidation catalyst and catalyzed ceramic traps. The catalectic formulas on these items are very susceptible to sulfur poisoning.

3. **Carbon dioxide (CO_2):** As a coal miner, you should know about this gas. It is not just a diesel gas but a gas that we deal with every day in the old workings of the mine where air flow is at minimum and old timbers or posts are decaying.

- CO_2 is the product of complete combustion. The Threshold Limit Value (TLV) for CO_2 is 5000 ppm.
- CO_2 is not poisonous but in concentrations of 10 percent or higher can cause unconsciousness or death by displacing the oxygen.
- Its acute symptoms are headache, sweating, rapid breathing, increased heartbeat, shortness of breath, dizziness.
- There are no chronic symptoms for CO_2. Once the individual is moved into fresh air the symptoms abate.

4. **Carbon monoxide (CO):** All coal miners should know about Carbon Monoxide.

- CO is formed by incomplete combustion.
- CO is colorless, odorless and tasteless. It truly is a silent killer. If you listen to the news or read the newspapers, you will hear of someone being overcome by CO mostly during the winter months. This usually happens by a space heater not working properly or someone sitting in a car with the engine running. There are many ways that this can happen and it does happen. Remember CO overexposure does have some symptoms at low level exposure.
- Some of the symptoms of overexposure are: headaches; nausea; weakness, dizziness, confusion, hallucinations, cyanosis, and angina.
- The hemoglobin of your blood has a 200 to 300 times affinity for CO than it does for oxygen, which means that your blood likes the CO even though it is no good for you 200 to 300 times more than oxygen.. If there is any CO present in your lungs. It will attach to the hemoglobin. This is referred to as carboxyhemoglobin. The CO will be delivered to all parts of the body; eventually you will asphyxiate from the inside out. This is known as a chemical asphyxiation.
- How much CO can you handle?
    - 200 ppm for 1 hour
    - 400 ppm for 30 minutes
    - 600 ppm for 15 minutes
- These concentrations are not fatal but they are a measure of how long you will have motor skills at that concentration. This means that at 600 ppm you will have about 15 minutes where you will still have motor skills and be able to function. However 600 ppm can be fatal in 2 to 3 hours. So at this concentration you would only have about 15 minutes before you become disoriented and want to fall asleep.
- The Pennsylvania diesel law has set the Threshold Limit Value (TLV) of CO at 35 ppm. A TLV is the concentration that a person can be exposed to on a daily basis and have no ill effects. However even though 35 ppm is the TLV for CO it is treated as a ceiling in Pennsylvania. This means that at any time the concentration of CO is 35 ppm or greater in your air split you must be removed from the split and shut down all diesel equipment operating on that split and perform service as needed to keep the CO under the TLV.

5. **Nitric oxide (NO)**: Is formed during the combustion process of a diesel engine. The leaner the AFR the more NO it will produce. In today's diesel engine it is the most prevalent gas. Most diesel engines today will produce about 100 to 200 ppm of CO and about 25 to 70 ppm of NO_2, but they will produce about 350 to 700 ppm of NO.

- It is a colorless gas
- It is said to have an irritating odor, but the odor is not significant to provide warning. This means that by the time you can smell it you have already been over exposed.
- NO is not a gas that Pennsylvania or MSHA requires to be tested for in underground coal mines. Even though we are not required to test for NO in coal mines it is a gas that MSHA does use to set the vent plate air quantities. Vent plate is the quantity of air that is needed in an air split to dilute and render harmless the gaseous portion of diesel exhaust on a given engine. For about the past ten years, most, if not all, vent plates have been based on the NO exhaust concentration.

- The American Conference on Governmental and Industrial Hygienist (ACGIH) has set a TLV of 25 ppm for NO.
- NO is a precursor for NO_2. This means that NO in the presence oxygen will convert to NO_2 naturally. This conversion is spontaneous, but is not instantaneous which means that it does happen but not immediately. This conversion rate in underground mines depends upon temperatures, barometric pressures and humidity, but it is something to watch for in areas of the mine that are not well ventilated. If you are sitting in a spur or dead head entry with little or no air flow please shut off the diesel. Pennsylvania law does not allow unnecessary idling of diesel powered equipment.

6. **Nitrogen dioxide (NO_2):** Well we talked about NO being a precursor of NO_2 so let's look at NO_2

- It has a reddish brown color.
- It has a very irritating odor said to be a sickening sweet odor.
- When inhaled into the warm moist areas of the lungs it can form nitric and nitrous acids, which can burn the lining of the lungs and the air sacs and cause a pulmonary edema (fluid in the lungs).
- In low level concentrations it can irritate the mucus membranes which will cause your nose to run and your eyes to water.
- In concentration of 60 to 150 ppm you could have delayed symptoms of up to 6 to 24 hours, some of the symptoms are:
  1) Tightness and burning of the chest
  2) Restlessness
  3) Sleepiness
  4) Shortness of breath
  5) Cyanosis
- Pennsylvania has set the TLV for NO_2 at 3 ppm. As with CO even though this is a TLV it is treated as a ceiling. You must be removed from any air split that has a concentration of 3 ppm or more of NO_2.
- You may think that NO_2 was introduced into our coal mines by the diesel engine, but that is not the case. Old coal miners can remember when we used nitroglycerin based explosives (permissible blasting power). After putting off a shot as you were walking back into the area your nose would start to run and your eyes would burn and you could smell that real sweet smell. That was the effect of the NO_2 created by the blast. Nitroglycerin based explosive create far more NO_2 than a diesel engine. Although NO_2 is not new to our coal mines diesels have brought it to the foreground.

Of the gases listed above Pennsylvania only regulates the two gases that are very detrimental to humans, CO and NO_2, MSHA only regulates that same two gases in underground diesel coal mines.

Pennsylvania's TLVS

- Carbon Monoxide (CO) TLV -- 35 ppm
  Action Level- 75% - 26.2 ppm

- Nitrogen Dioxide (NO2) TLV -- 3 ppm
  Action Level- 75% - 2.2 ppm

Looking at the chart you will notice that the TLV for these gases are what was previously discussed. The action level is the level of concentration at which you must begin to take action to insure that the TLV is not breached. Remember, if the TLV for either of these gases is breached you must remove all personnel from the affected air split and shut down the diesel equipment operating on that split and service the engines until the concentrations of either of these gases are below the action level on that air split. The Diesel Equipment Operating Plan at your mine will have actions listed that you must take if the action level of either of the gases is reached on a given air split. Remember, the TLV is the concentration that a human can breathe for an entire shift with no ill effects. However, Pennsylvania has set an action level for these gases to further insure your safety.

MSHA's TLVS

- Carbon Monoxide (CO) TLV -- 50 ppm
- Action Level- 50% - 25 ppm

- Nitrogen Dioxide (NO2) TLV -- 5 ppm
- Action Level- 50% - 2.5 ppm

You will notice that MSHA does allow a higher concentration of each of these gases. That is because when MSHA law was written, they adopted the 1972 standard for these gases whereas Pennsylvania adopted a latter version from ACGIH. If you look at the action level for both MSHA and Pennsylvania you will notice that they are fairly close even though Pennsylvania has a lower TLV's for these gases. Pennsylvania has set 75 % of their TLV for the action level. MSHA has set 50% of their TLV for the action level. This is the reason for the closely matching action levels.

B. SOLID EMISSIONS (DPM)

We now will take a look at the solid emissions or DPM that is being emitted by the engine. The particulates are broken down into three categories which are:

- Elemental Carbon
- Organic Carbon
- Water Condensate and sulfates

We have already talked about the sulfates so we will not go into that again, but we will look at the other two.

1. **Elemental Carbon (EC):** This is your basic black rock. Almost everything you burn on earth emits some EC. The EC emitted from a diesel engine is very fine... MSHA states that more than 95% of all DPM is one micrometer or less in size. Today's engines will produce less DPM by gravimetric measurement, but will produce almost the same if not more particles by number; this is caused by the engines being much more efficient. The DPM that is created is now moving into the nanometer range. The size and weight of the DPM has been reduced but the number of particles is the same or more.

2. **Organic Carbon (OC)**: OC comes from the fuel and lube oil that is burned in the engine. OC is the reason that DPM cannot be accurately measured in a coal mine. Because coal is organic carbon, and it is not possible to distinguish what part of a sample is coal and what part is diesel exhaust. This is why Pennsylvania chose to write a law that regulated the diesel powered equipment for DPM instead of the mine atmosphere.

Pennsylvania requires all diesel engines to been tested and approved for use by MSHA; this gives us a known DPM output for each engine.

We then marry these engines to an after-treatment system with a known efficiency rate that will maintain the DPM below the standard set by the State.

This will work fine on a new engine but how can we be sure that as the engine wears it will not produce DPM that is above the regulated standard? That is done by baseline CO emissions testing of each new engine and repeating that test at intervals that cannot exceed 100 hours of engine operation. At this point Pennsylvania is going to make an assumption. We know that CO and DPM will track together. That is to say as CO output increases in an engine DPM will also increase. The CO and DPM increase is not lineally but we know that as CO increases so does DPM. At the point that CO doubles the original baseline, the engine will be taken out of service and service must be performed on the engine until it is operating within its original CO baseline. Pennsylvania is using CO as surrogate for DPM. So, when we take an engine out of service for high CO emissions we are really saying that the engine is now emitting more DPM than the law allows. This is the assumption that Pennsylvania makes, and I believe that this approach works very well.

C. HEALTH EFFECT OF DPM:

Acute effects are:

- It can be an eye irritant
- It can irritate the mucus membranes
- It could produce a cough or respiratory irritation
- It can cause an allergic response in some people

Some of the **Chronic** effects are:

- Long term effects of high concentrations are really unknown at this time, it is thought to cause possible:
- Cardiovascular problems
- Cardiopulmonary problems
- Respiratory problems
- DPM has been listed as a likely carcinogen

D. FACTORS THAT EFFECT EMISSIONS

Since we have looked at the emissions of a diesel engine we should next delve into some factors that affect these emissions. Below is a list of some of these common factors:

- Intake air restriction above OEM spec
- Engine backpressure above OEM spec
- Turbocharger failure
- Engine faults
- Fuel
- Cetane number
- Aromatic content

1. **Intake air restriction over OEM spec:** We have talked about this in a prior segment but because I believe this to be the most critical system on a piece of underground diesel powered equipment we will touch on it again. We talked about how any air restriction above the normal for a given engine can cause over fueling of the engine and in turn cause high CO readings, but we have not talked about the effect of leaks on the suction side (prior to the turbocharger) of the intake system. As an equipment operator you should be familiar with what the normal restriction is for your equipment, and if you notice that the restriction is below what is considered normal you should have the system checked for leaks. You cannot lose vacuum on the intake system unless you change the air filter which should bring the gauge reading back to normal. But, if you are seeing a gauge reading that is less than normal it may mean the system has been breached and contaminated air could be going into the engine. This will accelerate the wear process and cause the high emissions to the point of having the engine fail.

2. **Back pressure over OEM spec**: This condition mirrors the intake system restriction. The engine must be allowed to breathe. If your backpressure is high, you will create more pressure in the crankcase of the engine which in turn will create more engine oil blowby and

increase CO emissions. Also, high backpressure will not allow the turbocharger to operate at its full capacity, which in turn, will restrict the air that is going into the cylinders. This will also result in high CO emissions.

3. **Turbocharger failure**: We have touched on the coking of the turbocharger in a prior section, but prior to complete failure of the turbocharger as it is being coked it will usually start to leak oil. Any oil leak caused by a worn turbocharger bearing will cause oil to leak into the intake manifold and be burned in the engine which will cause high CO emissions as well.

4. **Engine faults**: There are many ways that an engine can fail but most engine problems are caused by not maintaining the parameters of the engine, such as intake restriction or suction side leaks, high backpressure, contaminated fuel and oil ingested into the engine etc. This is not to say that a given engine will not fail on its own but these failures are few a far between if regular maintenance is perform on the above listed engine systems.

5. **Fuel**: The fuel standard set by Pennsylvania for underground diesel engines has been set as a moving target. If the EPA approves a new over the road diesel fuel standard we would have to adopt this fuel for our underground engines. This happened in 2007 when the EPA adopted the ultra-low sulfur diesel fuel that we are now using. Ultra low sulfur diesel fuel does meet Pennsylvania's fuel standard.

6. **Cetane Number**: Perhaps the best place to start the explanation of cetane is to say that cetane is a measure of the volatility of the fuel. The higher the cetane number the quicker the fuel will light and a more complete burning of the fuel will result. It is also a known fact that higher cetane fuel will produce lower emissions in a diesel engine. It must also be stated that today most diesel engine manufacturers would like to have a cetane minimum of about 50 for their engines. The European Union (EU) has recognized the benefits of higher cetane fuels. The EU has set 51 as the minimum cetane number for diesel fuel. The only area in the United States that has set a cetane standard that is similar to the EU standard is California. The California Air Resource Board (CARB) has set 53 as the minimum cetane number for over-the-road-diesel-fuel. When the EPA adopted the new ultra low sulfur fuel they did not change the minimum cetane number of the fuel which still stands at 40 according to the ASTM-D975 standard. Even though the minimum cetane standard has not changed, most of the over the road fuel today will carry about a 42 to 45 cetane number. This slight increase in cetane has been achieved by the further refining of the fuel to remove the sulfur. This is why over the road ultra-low sulfur diesel fuel is the absolute minimum that should use to fuel our underground equipment.

7. **Aromatic content** – The aromatic content in diesel fuel is where a lot of compounds such as benzene, naphthalene and aldehydes, are formed during the combustion process. There are even different classes of aromatics such as polyaromatic hydrocarbon (PAH). All of these compounds are released in trace forms, but it does stand to reason the less exposure to any of these compounds the better. The original diesel law of 1996 required the aromatic content of diesel fuel to be not more than 35%. This requirement mirrors the ASTM D975 standard. That part of the EPA fuel standard has not changed. However, there are fuels available that have much less aromatic content than ultra-low sulfur diesel fuel. Here again if your serious about diesel engine emissions you should look into fuels that have less aromatic content.

The last item that was regulated in the original diesel law was sulfur, as stated the sulfur content was to be no more than 500 ppm. Today EPA regulated amount of sulfur is 15 ppm.

The last thing I would like to do in this section is to show the importance of good fuel in your diesel engine. The below listed chart was taken from an engine that was emitting high CO emissions; because of these high emissions it was thought that the engine should be dyno-tested in an attempt to figure out what the problem was with this engine.

Source: Davis Training & Consulting

The CO emissions are listed on the left side of the chart and the vertical lines are time increments (each black line represents about 2 minutes of engine operation). As stated, this engine was put on the dyno to find what was causing the high CO emissions. You can see that when the engine was first put in a heavy lug condition that the CO emissions were about 250 ppm and within 2 minutes they had risen to where they were over 300 ppm. This trend of peaks and valleys continue for about 8 or 10 minutes then the CO emissions flat line at about 150 ppm, which is about half of the original CO emissions. No adjustments were made to this engine prior to operating it on the dyno and none were made while it was operating on the dyno. So why did the CO emissions decrease to almost half of what we were seeing with the engine operating in the piece of equipment? You have to ask yourself what was different on the dyno test stand and the equipment that the engine came from. The answer is nothing but fuel. When this engine was shipped to the test facility it had residual fuel in the injector pumps and fuel lines and filters, the jagged line that you see is the burnout of this fuel and once this contaminated fuel had been flushed out and the good fuel was being burned, the CO emissions were reduced by about 50%. This illustrates the emissions benefits of a good fuel.

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Dyno Run Chart. Davis Training & Consulting.

TABLE OF CONTENTS

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 3

EMISSIONS CONTROL TECHNOLOGIES FOR UNDERGROUND COAL MINES

Diesel Section 3

Emissions Control Technologies for Underground Coal Mines

This section of the study guide will explore some of the ways and items that can be used to reduce the total diesel exhaust emissions that are being emitted by your diesel powered equipment. Many people are looking for that silver bullet that you simply put on a diesel engine and it will reduce the emissions and never need maintenance. All of the items listed below must be coupled with a high quality maintenance program to insure the emissions benefits. Following is a list of the emissions control technologies that we will cover in this section.

- The use of low DPM emitting engines
- High quality diesel fuel
- Proactive engine maintenance
- Diesel Oxidation Catalyst (DOC)
- Diesel Particulate Filter (DPF)

The above listed items coupled with a proactive maintenance program will greatly reduce the emissions output of any engine.

A. THE USE OF LOW DPM EMITTING ENGINES

In the diesel fundamentals section of this study guide we talked about the technology improvements to the diesel engine in the last 20 years. What we did not talk about was how and why these improvements came about. In 1997 when this (Pennsylvania) law was introduced, the Environmental Protection Agency (EPA) got involved with emissions of off-road diesel engines. This resulted in a series of regulations aimed at lowering the total oxides of nitrogen (NOx) and particulate (DPM) emissions of diesel engines. This is known as the EPA Tier schedule. It began in 1996 with Tier I and culminated with EPA tier IV in 2015. These regulations were phased in by engine horsepower rating between 1996 and 2015. The phase in schedule was:

- Tier I – 1996- 2003
- Tier II- 2002- 2007
- Tier III 2008- 2014
- Tier IV 2013- 2015

This phase-in is now over and all diesel engines that have been produced after 2015 must meet all EPA tier regulations. This does not mean that you must get rid of your existing engines and purchase Tier IV engines but if your fleet has engines that are pre EPA tier or even EPA Tier I you should consider upgrading these engines to at least an EPA tier II engine. Most of the diesel engines in use in underground coal in Pennsylvania today are EPA tier II or Tier III engines. As of 2016 MSHA has not approved any Tier IV engines with horsepower ratings that would be suitable for use in underground coal mines. Thus the technological improvements made on these engines cannot help reduce diesel emissions if your engine does not have them.

There is another way you can reduce the emissions output of a given engine. It is called engine deration. Engine deration is accomplished by decreasing the fuel load on a given engine. It is referred to as deration because the engine is no longer operating at its rated capacity. In other words, it has less horsepower. Tests have shown that a 10 % date of the engine can reduce the DPM emissions by about 20 to 30%. Deration will also extend the life of an engine, simply because it is not operating at its rated capacity and the wear rates for that engine will be reduced.

Approval Number	Engine Manufacturer	Model	HP@RPM at 1,000 ft. Elevation	Ventilation Rate CFM	Particulate Index CFM	DPM grams/hr. weighted
07-ENA040007	DEUTZ	BF4M 1013FC	173 @ 2300	7000	4000	6.2
07-ENA040007	DEUTZ	BF4M 1013FC	157 @ 2200	6500	3000	4.88

Source: WWW.MSHA.gov

This chart clearly shows what a small deration can do for your engine. Looking at the chart you will notice that the horsepower rating of the bottom engine has been dated by about 9% this has been done by decreasing the fuel load on the engine but that deration has reduced the DPM from 6.2 g/h to 4.88 g/h. That is about a 21% decrease in the DPM output of the engine. You will also note that the ventilation rate has been reduced by 500 cfm. This example shows what a small deration can do for total DPM reduction. It must also be said that you cannot de-rate all of your engines. Some engines need all of their capacity to operate properly. But if you have equipment that is overpowered, this is an excellent way to improve their DPM emissions**. Note: In order to have an engine de-rated you must contact an authorized representative of the engine manufacturer to perform this action. This would require an adjustment to the fuel delivery system, and as such we are not allowed to perform this work.**

B. HIGH QUALITY FUEL

In the last section we touched on the importance of high quality fuel. Since we have already looked at D2 fuel (petroleum based fuel) we will look at two alternative fuels they are:

1. Bio diesel: According to the CDC information Circular IC 9462 bio diesel is the monoalkyl esters of long-chain fatty acids derived from renewable lipid sources. This means it is derived from renewable resources such as soy beans, animal fats and cooking oils. Bio-diesel can be blended with regular D2 fuel the most common blend is 20% bio diesel and 80% D2, which is referred to as a B20 blend. It can also be used "neat," which is saying that you are running 100% bio-diesel. Bio-diesel has extremely low sulfur and no aromatic content. Therefore the use of biodiesel may result in a substantial reduction of unburned HC, CO, and DPM emissions. Also with no aromatic content, most of the diesel exhaust odors will not be present. The cetane content of bio-diesel is slightly higher but comparable with D2 fuel. However biodiesel is an oxygenated fuel which means that it will burn more complete than regular D2 fuel. A 1997 U.S. Bureau of Mines study reported by Howell and Weber showed a 50% reduction of DPM when neat bio-diesel was used instead of regular D2 fuel. There is one drawback for long term use of biodiesel. It will act to soften some rubber compounds sometimes found in diesel engine fuel systems such as o-rings and seals. So if you have older engines you should check with the engine manufacturer to see if the use of bio-diesel will affect your fuel system.

2. Fisher-Tropsch (FT): We touched on this process earlier in this study guide when we mentioned the CTL process to convert coal to liquid. This process is actually called the Fisher-

Tropsch process. The following statement explaining this process has been taken from a National Institute of Occupational Health and Safety (NIOSH) study by George H. Schnakenberg, Jr., Ph.D., and Aleksandar D. Bugarski, Ph.D, that was released in 2002**: Fisher-Tropsch (F-T) conversion is a gas-to-liquid process used for synthesis of HC from CO and hydrogen. Historically, the process was used for producing synthetic diesel fuel from coal, natural gas, and biomass resources in countries where petroleum fuel stock was in short supply. The process has received attention recently because of its ability to convert natural gas resources to liquid fuels and chemicals. The synthetic diesel fuel produced by this process is of very high quality and has the potential to significantly reduce exhaust emissions.**

The above-referenced study also reported that, FT fuel has a cetane number of about 70 coupled with very low sulfur content which is less than 10 ppm and an aromatic content of less than 3%. You can see that this is a much better fuel than regular D2 fuel that we talked about in the last section. This study also stated West Virginia University had conducted testing using this type of fuel on buses that had no other exhaust after treatment system and found that the fuel alone produced a 30% reduction of DPM. The one drawback in using this fuel is the cost. The process producing this fuel is not very efficient at this time which makes the fuel expensive.

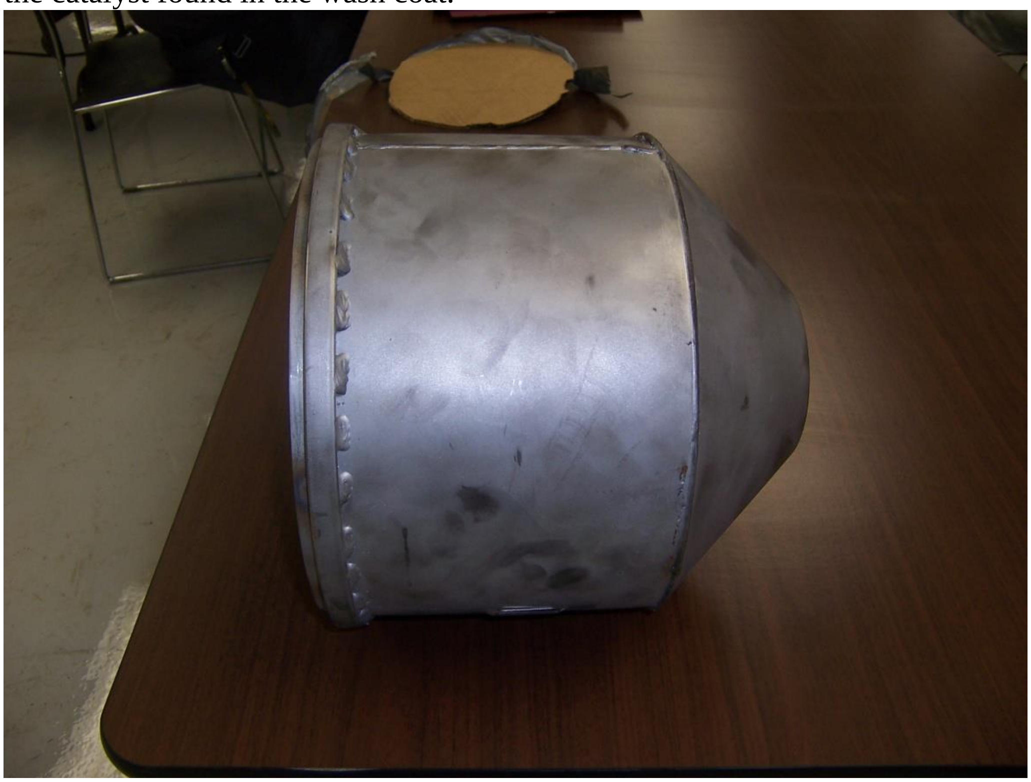
C. PROACTIVE ENGINE MAINTENANCE

We have touched on the maintenance of the diesel engine throughout this study guide but, this is the most important factor for reducing the total emissions of a diesel engine. The philosophy of "repair on failure" may provide short-term benefits but, in the long term, proactive engine maintenance delivers cleaner and more efficient engines that run longer with greater dependability. You must take a proactive approach to maintenance. The Pennsylvania law tries to instill this proactive approach to maintenance by requiring the “repeatable loaded engine operational test” found in Sections 417 and 418 of the law. As you are aware the Pennsylvania diesel law requires each engine to undergo a raw or untreated CO baseline emissions test prior to being put into service at a mine. This same test will then be repeated at intervals not to exceed 100 hours of engine operation. When the CO concentration is doubled or 100% higher than the original baseline, the engine must be taken out of service and service must be performed until the CO concentration is within the original CO baseline. This approach gives the impression that it is a proactive approach to engine maintenance. If you are operating your engines until the CO baseline is doubled, you are not maintaining your engines properly. Say for example you have an engine that has baselined at 100 ppm CO, and your next test is 160 ppm of CO. By law you do not need to work on this engine, but it is not operating properly. At this point you should perform some diagnostics on the engine to find the problem. You should check the turbo boost pressure to see if it is functioning properly, or check the intake system for leaks, or, perhaps the injectors are clogged and not firing properly. There are many things that can cause this increase in CO, but one thing for sure, something is causing it. By Pennsylvania law you would not have to remove this piece of equipment from service, but that does not mean it is OK to operate with a fault. If you do perform some of the checks listed above and you cannot deduce the problem, contact the engine representative and set a date when he can go over the parameters of the engine and find the fault. As stated, you can put this equipment back into service and work on it during its downtime. This would be a proactive approach. The emissions test that we perform can be used as a diagnostic tool for a starting point of maintenance. This is called "emissions based maintenance," and it is the crux of the Pennsylvania underground diesel law. So when you see an increase of 40 to 60 ppm in CO, you should start to look for the engine fault that is causing this increase, because there is a fault.

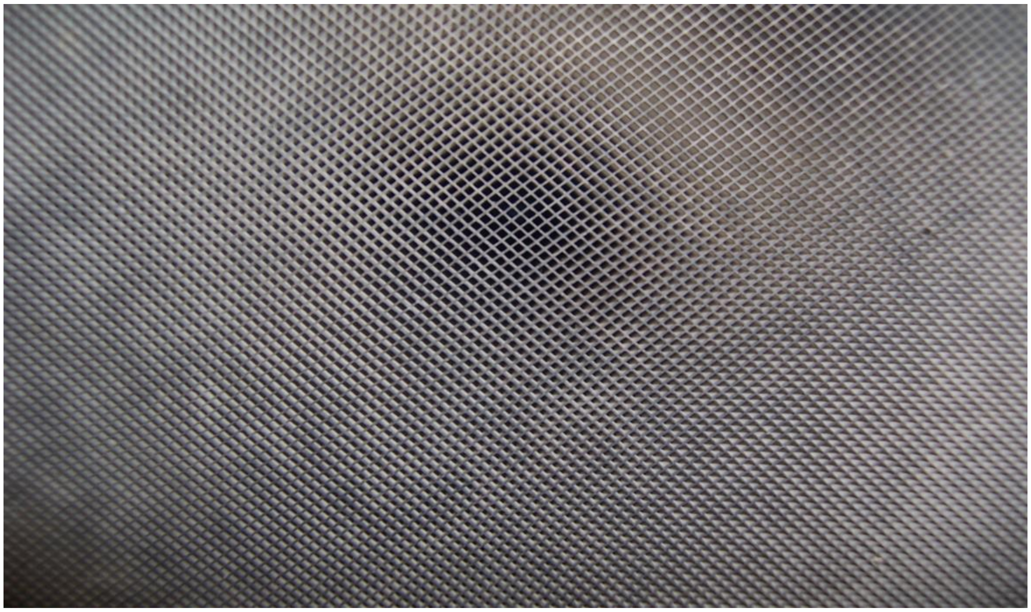
D. DIESEL OXIDATION CATALYSTS (DOC)

Section 403 of the Pennsylvania Underground Diesel Law states that each diesel engine must be fitted with: **An oxidation catalyst or other gaseous emissions control device capable of reducing undiluted carbon monoxide emissions to 100 parts per million or less under all conditions of operation at normal engine operating temperature range**. Even though the law does allow the use of "other gaseous emissions control devices" to date, the DOC has been used exclusively for this purpose in Pennsylvania. So let's take a look at some of the attributes and benefits of a DOC.

The DOC is made up of a monolith honeycombed substrate usually cordierite (ceramic) or steel. A wash coat of the actual catalyst is then applied to this substrate. The wash coat usually contains a precious metal such as platinum or palladium. Below are some pictures of a catalyst that is used in our mines. This catalyst substrate is made up of cordierite and the catalyst wash coat is applied into all of these small honeycombs. This is done to insure the exhaust flow contacts the catalyst found in the wash coat.



Source- Davis Training & Consulting



Source- Davis Training & Consulting

- Facts about the DOC:
  - According to diesel oxidation catalyst manufacturers such as, Nett Technologies. The catalytic wash coat is made up of a precious metal usually, platinum or palladium
  - The primary function is to oxidize CO to CO_2. This conversion rate is usually around 90 %.
  - Secondary function is to oxidize particulate hydrocarbons to water vapor. MSHA has tested this function, and found the conversion rate is about 20 to 30%.
  - The catalyst cannot perform its function until the light-off temperature of the catalyst has been achieved. Catalyst testing at the Mine Technology and Training Center in Ruff Creek, Pennsylvania has found this temperature to be about 500°F. A diesel engine will not achieve this exhaust temperature at low idle therefore your catalyst will not be functioning. This is another reason Pennsylvania does not allow unnecessary idling of diesel powered equipment. To insure the catalyst light-off temperature has been achieved, you should operate the equipment in a lower gear and higher RPM for as long as possible.
  - A catalyst by nature cannot be consumed nor altered while performing its function. This means that the catalyst should last the life of the engine. However, the catalyst wash coat can be poisoned by high sulfur fuels. Some components of engine oil and engine coolant (glycol). Any of these items entering the catalyst in sufficient quantities will render the catalyst useless.
  - A catalyst, because of the nature of the fine honeycombs can sometimes become clogged or coated to the point that the exhaust can no longer come into contact with the catalyst. This will show up as either excessive back pressure on the engine or lower CO conversion rates. When this happens the clog or the coating must be cleaned from the substrate.

Source-(CDC information circular IC 9462) (en.wikipedia.org/wiki/Catalytic_converter) (www.msha.gov) (Mine Technology and Training Center in Ruff Creek, Pennsylvania)

The picture below shows a catalyst that has undergone excessive idling.

Source- Davis Training & Consulting

This can be prevented by minimizing your engine idle time. We recognize that there are times when it is necessary to idle the engine. In these cases, be sure after any time you have idled the engine for an excessive period of time that you operate the engine in a lower gear and higher rpm which will help burn off some of this clog before it dries into a black cake such as illustrated.

The Diesel oxidation catalyst is an item that can greatly reduce the amount diesel emissions in our mine atmosphere if it is maintained properly.

E. DIESEL PARTICULATE FILTER (DPF)

The Pennsylvania underground diesel law does require a DPF to be used on every piece of diesel powered equipment that is operating in underground bituminous coal mines in the State. Even though these filters are being produced with many different materials we will limit this discussion to the types of filters that are being used in Pennsylvania mines.

- These filters fall into two categories:
  - **Disposable** (filters that are disposed after each used)
  - **Non-disposable** (filters that can be cleaned and reused)

1. Disposable filters - The disposable filters we are using are usually made of paper or a linen type product. The use of this type of filter mandates that the exhaust gas be cooled below 302° F prior to the exhaust entering the filter or the filter would be burned. This cooling is achieved in one of two ways, either by a wet bed scrubber, where the exhaust is pushed through a tank of water which in turn cools the exhaust, or the most common type which is the dry system that uses some type of dry heat exchanger either an air to air heat exchanger or a tube and shell heat exchanger. These disposable filters have a high efficiency rate on total DPM which is usually 95% or higher. This high efficient rate is achieved by the cooling of the exhaust which causes the vapor phase hydrocarbons and water vapor to condense into water droplets which are then captured by the filter.

The first piece of underground diesel powered equipment to be approved for use by the Pennsylvania Bureau of Mine Safety used an after-treatment system produced by a company named Dry Systems Technologies (DST). They incorporated the use of and oxidation catalyst coupled with a tube and shell heat exchanger to cool the exhaust and a paper filter, to capture the particulate. This is known as the DST system and it is still in use today. The disposable filter is probably the most common DPF filter in use in our mines today.

While these filters work well to reduce the total DPM emitted by our diesel equipment they are costly and they incur the maintenance of frequent filter changes.

2. Non disposable filters – Non disposable filters are filters that can be used cleaned and used again. The filters that fall into this class are usually made up of ceramic substances such as cordierite or silicon carbide (Sic). Below is a picture of a ceramic (cordierite) filter

Source- Davis Training & Consulting

You will notice that the substrate of this filter is the same as the substrate of the oxidation catalyst that was discussed in the last section. However, there is one difference. The oxidation catalyst is a flow through device and the filter is a DPM capture device which is commonly referred to as a “soot trap”. This trapping ability is achieved by blocking every other channel in the substrate which in turn traps the solid portion of the exhaust and allows the gases to pass through the porosity of the filter. Figure 1 below gives a better explanation of this process.

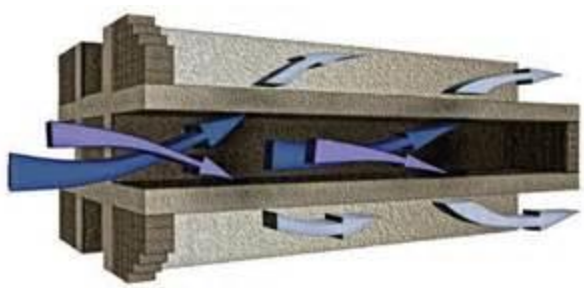


Figure 1. Source- www.msha.gov

This type of filter is slightly less efficient for total DPM than the paper filters. As stated paper filters are about 95% efficient whereas this type of filter is about 85 to 87% efficient. This drop in efficiency is caused by not cooling the exhaust prior to entering the filter which allows some of the vapor phase hydrocarbons and water condensate to pass through the filter as a gas. However, they are as (or more) efficient in capturing total carbon (EC+OC). Even though these filters are considered non disposable they must be cleaned after the engine back pressure is at its maximum. This cleaning is known as regeneration and can be achieved by either physically removing the filter from the equipment then placing it on a regeneration station (off board regeneration). Or by what is known as a CRT, which is a continuously regenerating trap. A CRT is a filter that has a catalytic wash coat that will lower the regeneration temperature of the diesel soot. Even with this wash coat it will take an exhaust temperature of about 700° F to for at least 30% of the equipment duty cycle for the CRT to work properly. Most of the equipment operating in our mines today does not have a duty cycle that will support the CRT. However, there are a few pieces of equipment operating in Pennsylvania with the CRT system and they have achieved upwards of 2000 hrs. of operating time without having to remove the filter from the equipment for cleaning. This is the exception not the rule. One drawback with these non-disposable filters is that these substrates are very delicate and can be broken by removing and replacing them back on the equipment.

There are other types of DPM filters that are available for use in our coal mines, such as the Microfresh filter which is a polypropylene filter and sintered metal filters which have been tested by MSHA and have produced a 99% efficiency rate. To see the full list of filters, that are available for use go to MSHA.gov and look at their DPM Filter list.

This section of the study guide has attempted to list some of the items that can be used to reduce the amount of DPM and gases that are being emitted by diesel equipment. In order to achieve the benefit of these items you must institute a proactive approach to maintenance. You can no longer simply allow the equipment to operate until it breaks down and then fix it. You must perform emissions based maintenance, which means, when the CO emissions are increasing in your engines you must perform maintenance to control this increase. This will allow you to achieve the most from your diesel exhaust after-treatment system.

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TABLE OF CONTENTS

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 4

DIESEL EQUIPMENT IN UNDERGROUND COAL MINES

Diesel Section 4

Diesel Equipment in Underground Coal Mines

Diesel powered equipment has been used in underground coal mines for over 40 years. However, the underground bituminous coal mine laws of Pennsylvania prohibited the use of diesel powered equipment until February 17th 1997 with the passage of Article II A. Article II A was revised in 2009 and incorporated into the Safety Laws of Pennsylvania for Underground Bituminous Coal Mines and is now simply known as Chapter 4 of the Safety Laws of Pennsylvania for Underground Bituminous Coal Mines.

The Pennsylvania diesel fleet has grown to about 400 pieces of diesel powered equipment currently approved for use in our underground coal mines. Prior to 1997 Pennsylvania was one of only three states that did not allow the use of this equipment; the other two states that did not allow the use of diesel equipment were West Virginia and Ohio. In 1996 the coal operators pushed for approval of diesel powered equipment but the current MSHA diesel rule did not afford the miners the protections they needed. The authors set out to write a law that would produce cleaner and better maintained diesel equipment and in turn protect our miners. I believe their goal has been accomplished. Since the passage of the Pennsylvania diesel law the legislatures of West Virginia and Ohio have also passed legislation allowing the use of diesel equipment. The legislation passed by these states is very similar and in many areas verbatim to Pennsylvania law. At its passage the Pennsylvania diesel law was one of the most, if not the most, stringent underground diesel law listed anywhere in the world.

The first piece of diesel powered equipment approved for use in Pennsylvania was a 20 ton rail mounted locomotive (MLD01) produced by Brookville Mining Equipment Corporation that incorporated a 3306 PCNA Caterpillar engine and a Dry Systems Technologies (DST) after - treatment system. This was the start of the Pennsylvania underground diesel fleet. The introduction of the diesel engine has made it much easier to move miners and supplies through the mine. There is no longer a need for trolley wire, and the need for battery charging stations has been greatly reduced. It is true that diesel equipment requires a high rate of maintenance, but if you consider the maintenance of the trolley wire and the battery charging stations the cost may be equalized.

A. TYPES OF DIESEL EQUIPMENT IN PENNSYLVANIA AS OF APRIL, 2016

An approximate breakdown of the types and numbers of diesel equipment follows:

- Personnel Carriers -213 –(both rail mounted and rubber tire)
- Rail mounted Locomotives - 70
- Load, Haul, Dump (Scoops) 57
- Stationary equipment - (rock dusters, pumps, hydraulic power packs etc.) - 60

1. **Personnel Carriers**: This is by far the primary use of diesel powered equipment with approximately 213 pieces of approved equipment falling into this category. These personnel carriers range from 18 man carriers to small 2-man maintenance vehicles and just about

everything in between. They also come in both rail mount and rubber tire versions. See the pictures below.

Source –Davis Training & Consulting


Source- Davis Training & Consulting

2. **Rail mounted locomotives:** Locomotives are the second most prevalent piece of diesel equipment in Pennsylvania with approximately 70 pieces in use in Pennsylvania. These locomotives are primarily used to deliver supplies to the working sections as well as moving long wall components in and out of the mine.

Source- Davis Training & Consulting

3. **Load, Haul, Dump (LHD):** This category takes in many types of equipment but in Pennsylvania this primarily refers to rubber tire scoops. There are approximately 57 pieces of equipment that fall into this category. Of these 57 pieces 53 pieces are listed as outby or non-permissible scoops and 4 are listed as permissible equipment. The majority of the non-permissible scoops are located in the low coal regions of Pennsylvania, where they are used to deliver supplies to the working sections as the locomotives are used in the rail mines. The permissible scoops are primarily used during longwall set-up and tear down. This equipment category has probably benefitted the most from the introduction of the diesel. This is because we no longer have to allocate an area for the battery charging station, nor do we have the down time of having to change the batteries every two to three hours. This is especially noticeable in the mines that are using non-permissible scoops to supply the working sections. These scoops can operate six to eight hours on one tank of fuel and never have to stop to change a battery. The introduction of the diesel has allowed these mines to be more productive.

Source- Davis Training & Consulting

4. **Stationary equipment:** This category is made up of non-self-propelled equipment such as rock dusters, hydraulic power-packs, and water pumps. This type of diesel equipment has been growing in numbers in the past few years and as a result there are now approximately 60 pieces of stationary of equipment approved for use in Pennsylvania mines. Stationary diesel powered equipment is also bringing new challenges to the equipment manufacturers. Since by nature this equipment is not operated in a transit mode as mobile diesel equipment is. But rather in a steady state mode, where the throttle is set at a high rpm. The engine may operate in that mode for hours at a time, while this will not hurt the diesel engine; it will produce more heat from both the engine and the exhaust. Combine this with the fact that many times this equipment may not be operated in the main air flow but in an area where the air flow is at the minimum needed to operate, and you could have a situation where the engine and exhaust piping will begin to overheat. Some manufacturers are trying to relieve this problem by placing fans or vortex tubes in certain areas of the engine compartment and near the exhaust piping to remove this residual heat.

As you can see each new type of diesel powered equipment brings its own challenges that must be addressed. The picture below is of a rubber tired rock duster.

Source- Davis Training & Consulting

NOTE: According to the underground coal diesel inventory, (MSHA.gov) there are about 5000 pieces of diesel powered equipment operating in our nations coal mines.

References

All pictures courtesy of Davis Training and Consulting.

TABLE OF CONTENTS

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 5

SAFETY LAWS OF PENNSYLVANIA FOR UNDERGROUND BITUMINOUS COAL MINES

Diesel Section 5

Safety Laws of Pennsylvania for Underground Bituminous Coal Mines

CHAPTER 4

A. Section 401. Underground use.

(a) General Rule.--Underground use of inby and outby diesel-powered equipment, including mobile equipment, stationary equipment and equipment of all horsepower ratings, shall only be approved, operated and maintained as provided under this chapter, used specifically for that purpose.

(b) Required attendant.--All diesel-powered equipment shall be attended while in operation with the engine running in underground mines. For purposes of this subsection, "attended" shall mean an equipment operator is within sight or sound of the diesel-powered equipment.

(c) Required certifications or approvals.--Inby and outby diesel-powered equipment may be used in underground mines if the inby or outby diesel-powered equipment uses an engine approved or certified by MSHA, as applicable, for inby or outby use that, when tested at the maximum fuel-air ratio, does not require a MSHA Part 7 approval plate ventilation rate exceeding 75 c.f.m.. per rated horsepower. If MSHA promulgates new regulations that change the MSHA Part 7 approval plate ventilation rate, the c.f.m.. requirement per rated horsepower shall be revised either up or down on a direct ratio basis upon recommendation of the technical advisory committee in accordance with section 424.

B. Section 402. Diesel-powered equipment package.

(a) Approval.--All diesel-powered equipment shall be approved by the department as a complete diesel-powered equipment package which shall be subject to all of the requirements, standards and procedures set forth under this chapter.

(b) Diesel engine approval.--Diesel engines shall be certified or approved, as applicable, by MSHA and maintained in accordance with MSHA certification or approval and approval by the department.

C. Section 403. Exhaust emissions control.

(a) Exhaust emissions control systems.--

(1) Except as provided in paragraph (3), underground diesel-powered equipment shall include an exhaust emissions control and conditioning system that has been laboratory tested with the diesel engine using the ISO 8178-1 test and has resulted in diesel particulate matter emissions that do not exceed an average concentration of 0.12 mg/m3 when diluted by 100% of the MSHA Part 7 approval plate ventilation rate for that diesel engine. If MSHA promulgates new regulations that change the MSHA Part 7 approval plate ventilation rate, the dilution percentage relative to the approval plate ventilation rate shall be adjusted either up or down on a direct ratio basis upon recommendation of the technical advisory committee in accordance with section 424.

(2) Except as provided in paragraph (3), the exhaust emissions control and conditioning system shall be required to successfully complete a single series of laboratory tests for each diesel engine, conducted at a laboratory accepted by the department.

(3) An exhaust emissions control and conditioning system may be approved for multiple diesel engine applications through a single series of laboratory tests, known as the ISO 8178-1 test, only if data is provided to the technical advisory committee that reliably verifies that the exhaust emissions control and conditioning system meets, for each diesel engine, the in-laboratory diesel particulate matter standard established by this subsection. Data provided to satisfy this paragraph shall include diesel particulate matter production rates for the specified engine as measured during the ISO 8178-1 test, if available. If ISO 8178-1 test data for diesel particulate matter production is not available for a specified engine, comparable data may be provided to the technical advisory committee that reliably verifies that the exhaust emissions control and conditioning system shall meet, for the specified diesel engine, the in-laboratory diesel particulate matter standard established by this subsection. This standard shall only be used for in-laboratory testing for approval of diesel-powered equipment for use underground.

(b) Components of exhaust emissions system.--The exhaust emissions control and conditioning system shall include the following:

(1) A diesel particulate matter (DPM) filter that has proven capable of a reduction in total diesel particulate matter to a level that does not exceed the requirements of subsection (a)(1). However, the technical advisory committee may evaluate, in accordance with section 424, alternative technologies that have the ability to meet the 0.12 mg/m3 standard.

(2) An oxidation catalyst or other gaseous emissions control device capable of reducing undiluted carbon monoxide emissions to 100 parts per million or less under all conditions of operation at normal engine operating temperature range.

(3) An engine surface temperature control capable of maintaining significant external surface temperatures below 302 degrees Fahrenheit.

(4) A system capable of reducing the exhaust gas temperature below 302 degrees Fahrenheit.

(5) An automatic engine shutdown system that shuts off the engine before the exhaust gas temperature reaches 302 degrees Fahrenheit and, if water-jacketed components are used, before the engine coolant temperature reaches 212 degrees Fahrenheit. A warning shall be provided to alert the equipment operator prior to engine shutdown.

(6) A spark arrestor system.

(7) A flame arrestor system.

(8) A sampling port for measurement of undiluted and untreated exhaust gases as they leave the engine.

(9) A sampling port for measurement of treated undiluted exhaust gases before they enter the mine atmosphere.

(10) For permissible diesel equipment, any additional MSHA regulations must be met.

(c) Diagnostics systems.--Onboard engine performance and maintenance diagnostics systems shall be capable of continuously monitoring and giving readouts for paragraphs (1), (2), (3), (4), (5), (6), (7) and (8). The diagnostics system shall identify levels that exceed the engine or component manufacturer's recommendation or the applicable MSHA or bureau requirements as to the following:

(1) Engine speed.

(2) Operating hour meter.

(3) Total intake restriction.

(4) Total exhaust back pressure.

(5) Cooled exhaust gas temperature.

(6) Coolant temperature.

(7) Engine oil pressure.
(8) Engine oil temperature.

D. Section 404. Ventilation.

(a) Minimum quantities.--Minimum quantities of ventilating air where diesel-powered equipment is operated shall be maintained pursuant to this section.

(b) Approvals.--Each specific model of diesel-powered equipment shall be approved by the department before it is taken underground. The department shall require that an approval plate be attached to each piece of the diesel-powered equipment. The approval plate shall specify the minimum ventilating air quantity for the specific piece of diesel-powered equipment. The minimum ventilating air quantity shall be determined by the bureau based on the amount of air necessary at all times to maintain the exhaust emissions at levels not exceeding the exposure limits established under Section 419.

(c) Minimum air quantities.--The minimum quantities of air in any split where any individual unit of diesel-powered equipment is being operated shall be at least that specified on the approval plate for that equipment. Air quantity measurements to determine compliance with this requirement shall be made at the individual unit of diesel-powered equipment.

(d) Multiple units in operation.--Where multiple units are operated, the minimum quantity shall be at least the total of 100% of MSHA's Part 7 approval plate ventilation rate for each unit operating in that split. Air quantity measurements to determine compliance with this requirement shall be made at the most downwind unit of diesel-powered equipment that is being operated in that air split. If MSHA promulgates new regulations that change the MSHA Part 7 approval plate ventilation rate, the minimum quantity where multiple units are operated shall be revised on a direct ratio basis upon recommendation of the technical advisory committee in accordance with section 424.

(e) Minimum quantities of air in certain splits.—The minimum quantities of air in any split where any diesel-powered equipment is operated shall be in accordance with the minimum air quantities required in subsections (a), (b) and (c) and shall be specified in the mine diesel ventilation plan.

E. Section 405. Fuel storage facilities.

(a) General rule.--An underground diesel fuel storage facility shall be any facility designed and constructed to provide for the storage of any mobile diesel fuel transportation units or the dispensing of diesel fuel.

(b) Diesel fuel standards.--Diesel-powered equipment shall be used underground only with fuel that meets the standards of the most recently approved United States Environmental Protection Agency (EPA) guidelines for over-the-road fuel. Additionally, the fuel shall also meet the ASTM D975 standards with a flash point of 100 degrees Fahrenheit or greater at standard temperature and pressure. The operator shall maintain a copy of the most recent delivery receipt from the supplier to verify that the fuel used underground meets this standard.

(c) Requirements.--Underground diesel fuel storage facilities shall meet the following general requirements:

(1) Fixed underground diesel fuel storage tanks are prohibited.

(2) No more than 500 gallons of diesel fuel shall be stored in each underground diesel fuel storage facility.

(d) Location.--Underground diesel fuel storage facilities shall be located as follows:

(1) at least 100 feet from shafts, slopes, shops and explosives magazines;

(2) at least 25 feet from trolley wires, haulage ways, power cables and electric equipment not necessary for the operation of the storage facilities; and

(3) in an area that is as dry as practicable.

(e) Construction requirements.--

(1) Underground diesel fuel storage facilities shall meet the construction requirements and safety precautions under this subsection.

(2) Underground diesel fuel storage facilities shall meet all of the following:

(i) Be constructed of noncombustible materials and provided with either self-closing or automatic closing doors.

(ii) Be ventilated directly into the return air course using noncombustible materials.

(iii) Be equipped with an automatic fire suppression system complying with section 408. The technical advisory committee may recommend for approval an alternate method of complying with this section on a mine-by-mine basis in accordance with section 424.

(iv) Be equipped with at least two portable 20-pound multipurpose dry-chemical type fire extinguishers.

(v) Be marked with conspicuous signs designating combustible liquid storage.

(vi) Be included in the pre-shift examination.

(3) Welding or cutting other than that performed in accordance with paragraph (4) shall not be done within 50 feet of a diesel fuel storage facility.

(4) When it is necessary to weld, cut or solder pipelines, cylinders, tanks or containers that may have contained diesel fuel, the following requirements shall apply:

(i) Cutting or welding shall not be performed on or within containers or tanks that have contained combustible or flammable materials until the containers or tanks have been thoroughly purged and cleaned or rendered inert and a vent or opening is provided to allow for sufficient release of any buildup pressure before heat is applied.

(ii) Diesel fuel shall not be allowed to enter pipelines or containers that have been welded, soldered, brazed or cut until the metal has cooled to ambient temperature.

F. Section 406. Transfer of diesel fuel.

(a) General rule.--Diesel fuel shall be transferred as provided in this section.

(b) Pump transfers.--When diesel fuel is transferred by means of a pump and a hose equipped with a nozzle containing a self-closing valve, a powered pump may be used only if:

(1) the hose is equipped with a nozzle containing a self-closing valve without a latch-open device; and

(2) the pump is equipped with an accessible emergency shutoff switch.

(c) Compressed gas prohibition.--Diesel fuel shall not be transferred using compressed gas.

(d) Status of diesel engine.--Diesel fuel shall not be transferred to the fuel tank of diesel-powered equipment while the equipment's engine is running.

(e) Dry-system design.--Diesel fuel piping systems shall be designed and operated as dry systems.

(f) Standards for pipes, valves and fittings.--All piping, valves and fittings shall meet the following requirements:

(1) Be capable of withstanding working pressures and stresses.

(2) Be capable of withstanding four times the static pressures.

(3) Be compatible with diesel fuel.
(4) Be maintained in a manner that prevents leakage.

(g) Manual shutoff valves.--Vertical pipelines shall have manual shutoff valves installed at the surface filling point and at the underground discharge point.

(h) Exposed fuel pipelines.--Unburied diesel fuel pipelines shall not exceed 300 feet in length and shall have shutoff valves located at each end of the unburied pipeline.

(i) Horizontal pipeline prohibition.--Horizontal pipelines shall not be used to distribute fuel throughout a mine.

(j) Limitation on piping systems.--Diesel fuel piping systems shall be used only to transport fuel from the surface directly to a single underground diesel fuel transfer point.

(k) Restrictions related to boreholes.--When boreholes are used, the diesel fuel piping system shall not be located in a borehole with electric power cables.

(l) Inspections.--Diesel fuel pipelines located in any shaft shall be included as part of the required examination of the shaft.

(m) Location in entries.--Diesel fuel piping systems located in entries shall not be located on the same side of the entry as electric cables or power lines.

(n) Trolley-haulage limitations.--Diesel fuel pipelines shall not be located in any trolley-haulage entry, except that they may cross the entry perpendicular if buried or otherwise protected from damage and sealed.

(o) Protection.--Diesel fuel piping systems shall be protected to prevent physical damage.

G. Section 407. Containers.

(a) General rule.--Containers for the transport of diesel fuel shall meet the requirements of this section.

(b) Limitations on containers.--Diesel fuel shall be transported only in containers specifically designed for the transport of diesel fuel.

(c) Limitations on vehicle transport.--No more than one safety can, conspicuously marked, shall be transported on a vehicle at any time.

(d) Standards for containers other than safety containers -- Containers, other than safety cans, used to transport diesel fuel shall be provided with the following:

(1) Devices for venting.
(2) Self-closing caps.
(3) Vent pipes at least as large as the fill or withdrawal connection, whichever is larger, but not less than one and one-fourth inch nominal inside diameter.
(4) Liquid-tight connections for all container openings that are identified by conspicuous markings and closed when not in use.
(5) Shutoff valves located within one inch of the tank shell on each connection through which liquid can normally flow.

(e) Tanks with manual gauging.--When tanks are provided with openings for manual gauging, liquid- tight caps or covers shall be provided and shall be kept closed when not open for gauging.

(f) Capacity of containers.--Containers used for the transport of diesel fuel shall not exceed a capacity of 500 gallons.

(g) Certain containers as permanent fixtures.--Containers, other than safety cans, used for the transport of diesel fuel shall be permanently fixed to the transportation unit.

(h) Method of transportation.--Diesel fuel transportation units shall be transported individually and not with any other cars, except that two diesel fuel transportation units up to a maximum of 500 gallons each may be transported together.
(i) Prohibition.--Diesel fuel shall not be transported on conveyor belts.
(j) Fire extinguisher.--When transporting diesel fuel in containers other than safety cans, a fire extinguisher shall be provided on each end of the transportation unit. The fire extinguishers shall be multipurpose type dry-chemical fire extinguishers containing a nominal weight of 20 pounds.
(k) Fire suppression systems for diesel transportation units.--Diesel fuel transportation units shall have a fire suppression system that meets the requirements of section 408.
(l) Limitations where trolley wires are present.--In mines where trolley wire is used, diesel fuel transportation units shall be provided with insulating material to protect the units from any energized trolley wire, and the distance between the diesel fuel transportation unit and the trolley wire shall not be less than 12 inches, or the trolley wire shall be de-energized when diesel fuel transportation units are transported through the area.
(m) Parking restrictions.--Unattended diesel fuel transportation units shall be parked only in underground diesel fuel storage facilities.
(n) Emergency fueling restrictions.--Safety cans shall be used for emergency fueling only.
(o) Standards for safety cans.--Safety cans shall be clearly marked, have a maximum capacity of five gallons, be constructed of metal and be equipped with a nozzle and self-closing valves.

H. Section 408. Fire suppression for equipment and transportation.

(a) General rule.--Fire suppression systems for diesel-powered equipment and fuel transportation units shall meet the requirements of this section.
(b) Type system.--The system must be an automatic multipurpose dry-powder type fire suppression system suitable for the intended application and listed or approved by a nationally recognized independent testing laboratory. Installation requirements shall be as follows:
(1) The system shall be installed in accordance with the manufacturer's specifications and the limitations of the listing or approval.
(2) The system shall be installed in a protected location or guarded to minimize physical damage from routine operations.
(3) Suppressant agent distribution tubing or piping of the system shall be secured and protected against damage, including pinching, crimping, stretching, abrasion and corrosion.
(4) Discharge nozzles of the system shall be positioned and aimed for maximum fire suppression effectiveness in the protected areas. Nozzles shall also be protected against the entrance of foreign materials, such as mud, coal dust or rock dust that could prevent proper discharge of suppressant agent.
(c) Automatic fire detection and suppression.--The fire suppression system shall provide automatic fire detection and suppression for all of the following:
(1) The engine, transmission, hydraulic pumps and tanks, fuel tanks, exposed brake units, air compressors and battery areas, as applicable, on all diesel-powered equipment.
(2) Fuel containers and electric panels or controls used during fuel transfer operations on fuel transportation units.
(d) Fault and fire alarm annunciators.--The fire suppression system shall include a system fault and fire alarm annunciator that can be seen and heard by the equipment operator.

(e) Automatic engine shutdown.--The fire suppression system shall provide for automatic engine shutdown. Engine shutdown and discharge of suppressant agent may be delayed for a maximum of 15 seconds after the fire alarm annunciator alerts the operator.
(f) Manual actuators.--At least two manual actuators shall be provided, with at least one manual actuator at each end of the equipment. If the equipment is provided with an operator's compartment, one of the mechanical actuators shall be located in the compartment within easy reach of the operator. For stationary equipment, the two manual actuators shall be located with at least one actuator on the stationary equipment and at least one actuator a safe distance away from the equipment and in intake air.

I. Section 409. Fire suppression for storage areas.

(a) General rule.--Fire suppression systems for diesel fuel storage areas shall meet the requirements of this section.
(b) Type system.--The system shall be an automatic multipurpose dry-powder type fire suppression system or other system of equal capability, suitable for the intended application and listed or approved by a nationally recognized independent testing laboratory. The system shall meet the following installation requirements:
(1) The system shall be installed in accordance with the manufacturer's specifications and the limitations of the listing or approval.
(2) The system shall be installed in a protected location or guarded to minimize physical damage from routine operations.
(3) Suppressant agent distribution tubing or piping of the system shall be secured and protected against damage, including pinching, crimping, stretching, abrasion and corrosion.
(4) Discharge nozzles of the system shall be positioned and aimed for maximum fire suppression effectiveness in the protected areas. Nozzles shall also be protected against the entrance of foreign materials, such as mud, coal dust and rock dust that could prevent proper discharge of suppressant agent.
(c) Automatic fire detection and suppression.--The fire suppressant system shall provide automatic fire detection and suppression for the fuel storage tanks, containers, safety cans, pumps, electrical panels and control equipment in fuel storage areas.
(d) Types of alarms.--Audible and visual alarms to warn of fire or system faults shall be provided at the protected area and at a surface location that is always staffed when individuals are underground. A means shall also be provided for warning all endangered individuals in the event of fire.
(e) Manual actuators.--Fire suppression systems shall include two manual actuators with at least one located within the fuel storage facility and at least one located a safe distance away from the storage facility and in intake air.
(f) System operation.--The fire suppression system shall remain operative in the event of electrical system failure.
(g) Monitoring of certain systems.--If electrically operated, the detection and actuation circuits shall be monitored and provided with status indicators showing power and circuit continuity. If not electrically operated, a means shall be provided to indicate the functional readiness status of the system.
(h) Weekly visual inspection.--Fire suppression devices shall be visually inspected at least once each week by an individual qualified to make the inspection.

(i) Maintenance, testing and records.--Each fire suppression device shall be tested and maintained. A record shall be maintained of the inspection required by this subsection. The record of the weekly inspections shall be maintained at an appropriate location for each fire suppression device.
(j) (Reserved).
(k) Instructions.--All miners normally assigned to the active workings of a mine shall be instructed about any hazards inherent to the operation of all fire suppression devices installed and, where appropriate, the safeguards available for each device.

J. Section 410. Use of certain starting aids prohibited.
The use of volatile or chemical starting aids is prohibited.

K. Section 411. Fueling.
(a) Restrictions on fueling locations.--Fueling of diesel-powered equipment shall not be conducted in the intake escape-way unless the mine design and entry configuration make it necessary. In those cases where fueling in the intake escape-way is necessary, the mine operator shall submit a plan for approval to the department, which shall be investigated by the technical advisory committee in accordance with section 424, outlining the special safety precautions that will be taken to insure the protection of miners. The submitted plan shall specify a location, such as the end of the tail piece track or adjacent to the load out point, where fueling shall be conducted in the intake escape-way and all other safety precautions that shall be taken, which shall include an examination of the area for spillage or fire by a qualified individual.
(b) Spill cleanup.--Diesel fuel and other combustible materials shall be cleaned up and not be permitted to accumulate anywhere in an underground mine or on diesel-powered or electric equipment located in a mine.
(c) Trained individual on duty.--At least one individual specially trained in the cleanup and disposal of diesel fuel spills shall be on duty at the mine when diesel-powered equipment or mobile fuel transportation equipment is being used or when any fueling of diesel-powered equipment is being conducted.

L. Section 412. Fire and safety training.
(a) Training of underground employees.--All underground employees at the mine shall receive special instruction related to fighting fires involving diesel fuel. This training may be included in annual refresher training under MSHA regulations at 30 CFR Part 48 (relating to training and retraining of miners) or included in the fire drills required under MSHA regulations relating to program of instruction; location and use of firefighting equipment; location of escape-ways, exits and routes of travel; evacuation procedures; and fire drills.
(b) Training of miners.--All miners shall be trained in precautions for safe and healthful handling and disposal of diesel-powered equipment filters. All used intake air filters, exhaust diesel particulate matter filters and engine oil filters shall be placed in their original containers or other suitable enclosed containers and removed from the underground mine to the surface. Arrangements shall be made for safe handling and disposal of these filters within a timely manner after they have reached the surface.

M. Section 413. Maintenance.

(a) General rule.--Diesel-powered equipment shall be maintained in an approved and safe condition as described in this chapter or removed from service. Failure of the mine operator to comply with the maintenance requirements of this subsection may result in revocation of the department's approval of the complete diesel-powered equipment package, provided appropriate notification has been given to the mine operator and the procedures of this section have been followed. Upon receiving the appropriate notification, the mine operator shall have 30 days to submit a plan to achieve and maintain compliance. The plan shall be evaluated by the department and, upon approval, the mine operator shall implement the plan. The department shall monitor the mine operator's compliance. If the department then determines that the mine operator is unable or unwilling to comply, the department shall revoke the mine operator's approval.

(b) Acquisition and maintenance of approvals.--To acquire and maintain approval of a complete diesel-powered equipment package, the mine operator shall comply with the following requirements:

(1) All service, maintenance and repairs of approved complete diesel-powered equipment packages shall be performed by mechanics who are trained and qualified in accordance with section 422.

(2) Service and maintenance of approved complete diesel-powered equipment packages shall be performed according to:

(i) the specified routine maintenance schedule;

(ii) onboard performance and maintenance diagnostics readings;

(iii) emissions test results; and

(iv) component manufacturers' recommendations.

N. Section 414. Records.

(a) General rule.--A record shall be made of all emissions tests, preoperational examinations and maintenance and repairs of complete diesel-powered equipment packages. The records made pursuant to this section shall meet the requirements of this section.

(b) Written certification.--The individual performing the emissions test, examination, maintenance or repair shall certify by date, time, engine hour reading and signature that the emissions test, examination, maintenance or repair was made.

(c) Results.--Records of emissions tests and examinations shall include the specific results of such tests and examinations.

(d) Content.--Records of maintenance and repairs shall include the work that was performed, any fluids or oil added, parts replaced or adjustments made and the results of any subsequently required emissions testing.

(e) Preoperational examination record retention.--Records of preoperational examinations shall be retained for the previous 100-hour maintenance cycle.

(f) Certain records to be countersigned.--Records of emissions tests, 100-hour maintenance tests and repairs shall be countersigned once each week by the certified mine electrician or mine foreman.

(g) Other record retention.--Except as specified in subsection (e), all records required by this section shall be retained for at least one year at a surface location at the mine and made available for inspection by the department and by miners and their representatives.

O. Section 415. Duties of equipment operator.

(a) Preoperational examination.--Prior to use of a piece of diesel-powered equipment during a shift, an equipment operator shall conduct an examination as follows:

(1) Check the exhaust emissions control and conditioning system components to determine that the components are in place and not damaged or leaking.
(2) Assure that the equipment is clean and free of accumulations of combustibles.
(3) Assure that the machine is loaded safely.
(4) Check for external physical damage
(5) Check for loose or missing connections.
(6) Check engine oil level.
(7) Check transmission oil level.
(8) Check other fluid levels, if applicable.
(9) Check for hydraulic, coolant and oil leaks.
(10) Check fan, water pump and other belts.
(11) Check the fan for damage.
(12) Check guards.
(13) Check the fuel level.
(14) Check for fuel leaks.
(15) Comply with recordkeeping requirements pursuant to section 414.

(b) Operational examination.--After the engine is started and warmed up, the equipment operator shall conduct an examination as follows:

(1) Check all onboard engine performance and maintenance diagnostics system gauges for proper operation and in-range readings. The equipment operator shall immediately shut down the engine and notify the operator if the onboard readings indicate any of the following:

(i) Intake restriction at full engine speed is greater than the manufacturer's recommendation.
(ii) Exhaust restriction at full engine speed is greater than the manufacturer's recommendation
(iii) Coolant temperature is at or near 212 degrees Fahrenheit.
(iv) Low engine oil pressure.
(v) High engine oil temperature.

(2) Check safety features, including, but not limited to, the throttle, brakes, steering, lights and horn.

(3) Comply with recordkeeping requirements pursuant to section 414.

P. Section 416. Schedule of maintenance.

At intervals not exceeding 100 hours of engine operation, a qualified mechanic shall perform the following maintenance and make all necessary adjustments or repairs or remove the equipment from service:

(1) Wash or steam clean the equipment.
(2) Check for and remove any accumulations of coal, coal dust or other combustible materials.
(3) Check the equipment for damaged or missing components or other visible defects.
(4) Conduct electrical and safety component inspections.
(5) Replace engine oil and oil filter.
(6) Check the transmission oil level and add oil, if necessary.

(7) Check hydraulic oil level and add oil, if necessary.
(8) Check the engine coolant level and add coolant, if necessary.
(9) Check all other fluid levels and add fluid, if necessary.
(10) Check for oil, coolant and other fluid leaks.
(11) Inspect the cooling fan, radiator and shroud. Remove any obstructions and make necessary repairs.
(12) Check all belts. Tighten or replace, if necessary.
(13) Check the battery and service as necessary.
(14) Check the automatic fire suppression system.
(15) Check the portable fire extinguisher.
(16) Check the lights.
(17) Check the warning devices.
(18) With the engine operating, check and replace or repair the following:
(i) Oil pressure.
(ii) Intake air restriction at full engine speed.
(iii) Exhaust gas restriction at full engine speed.
(iv) Exhaust flame arrestor.
(v) All gauges and controls.
(19) Conduct repeatable loaded engine-operating test in accordance with section 418.
(20) If the equipment is approved with a non-disposable diesel particulate filter, a smoke dot test of the filtered exhaust must be performed at this time. The results of the smoke dot test shall be recorded on the 100-hour emissions form. If the interpreted smoke dot number is greater than three, the technical advisory committee shall be notified and shall investigate to determine if the filter is functioning properly.
(21) Evaluate and interpret the results of all of the above tests and examinations and make all necessary repairs or remove the equipment from service.
(22) Comply with the recordkeeping requirements pursuant to section 414.

Q. Section 417. Emissions monitoring and control.

(a) General rule.--Emissions for diesel-powered equipment shall be monitored and controlled as provided in this section.
(b) Determination of baseline emission values.--When any diesel-powered equipment first enters service at a mine, baseline emission values shall be determined by a qualified mechanic. Unless the technical advisory committee in accordance with section 424 recommends an alternate procedure, the qualified mechanic shall:
(1) Verify that the seal on the engine fuel injector is in place and that the proper fuel pump is on the equipment.
(2) Install a new clean intake air cleaner, measure and record the intake restriction pressure.
(3) Check the level of engine oil.
(4) Change the engine lubrication oil if not fresh.
(5) Check the level of the transmission fluid.
(6) Measure and record the exhaust backpressure. If exhaust gas back pressure is above that recommended by the manufacturer, steps must be taken to bring the exhaust gas back pressure within the manufacturer's recommended limit prior to beginning the test described in this subsection.
(7) Test the brakes.
(8) Place the equipment into an intake entry.

(9) Set the brakes and chock the wheels.
(10) Install an exhaust gas analyzer into the untreated exhaust gas port.
(11) Start the engine and allow it to warm up to operating temperature.
(12) Put the engine into a loaded condition. For this section, the loaded condition for the baseline emissions testing shall be determined by the technical advisory committee by determining CO_2 values that are representative of the MSHA lug curve readings for that engine model and horsepower.
(13) Start the exhaust gas analyzer and allow the engine to operate in the loaded condition for a sufficient length of time not less than a 90 second duration to insure proper CO readings. The qualified mechanic shall record both CO and CO_2 readings. Note: Baseline CO values shall be determined by the technical advisory committee based upon MSHA lug curve readings for that engine model and horsepower. If the baseline CO values are greater than the MSHA lug curve values, the technical advisory committee shall investigate and either recommend approval or disapproval or recommend alternate methods of meeting the requirements of this section.
(14) Comply with recordkeeping requirements pursuant to section 414.
(15) An alternative to the testing provided in paragraphs (1) through (14) may be developed by the technical advisory committee in accordance with Section 424.
(16) Emissions test procedures for this section shall be submitted to the technical advisory committee in accordance with section 424 prior to being implemented for each engine and equipment type.

R. Section 418. Diagnostic testing.

(a) Tests.--At intervals not exceeding once every 100 hours of engine operation, a qualified mechanic shall perform equipment maintenance diagnostic testing of each piece diesel-powered equipment in the mine. The qualified mechanic shall do all of the following:
(1) Verify the identification numbers on the equipment.
(2) Check the level of the engine lubricating oil.
(3) Check the level of the transmission fluid.
(4) Set the brakes and chock the wheels.
(5) Install the portable carbon monoxide sampling device into the untreated exhaust port coupling provided in the operator's cab.
(6) Start the engine and allow it to warm up to operating temperature.
(7) Check the intake restriction and the exhaust back pressure at high idle speed.
(8) If the intake restriction is more than the manufacturer's maximum recommended intake restriction, replace the intake filter with a clean one.
(9) If exhaust gas back pressure is above that recommended by the manufacturer, take steps to bring the exhaust gas back pressure within the manufacturer's recommended limit prior to beginning the test described in this section.
(10) Put the engine into a loaded condition. As used in this paragraph, the term loaded condition shall mean a condition in which the carbon dioxide values are representative of the MSHA lug curve values for that engine model and horsepower rating.
(11) Take the following steps:
(i) Start the exhaust gas analyzer.

(ii) Allow the engine to operate for a sufficient time, not less than 90 seconds, to insure proper carbon monoxide readings and record both carbon monoxide and carbon dioxide readings.

(12) Install the exhaust gas analyzer into the treated exhaust port and repeat steps set forth in paragraphs (10) and (11).

(13) If the average carbon monoxide reading for untreated exhaust gas is greater than twice the baseline established under section 417(b) or if the average carbon monoxide reading for treated exhaust gas is greater than 100 parts per million, the equipment has failed and shall be serviced and retested before it is returned to regular service.

(14) Comply with recordkeeping requirements under Section 414.

(b) Procedures.--Emissions test procedures for this section must be submitted to the technical advisory committee under section 424 prior to being implemented for each engine and equipment type.

(c) Alternative procedure.--An alternative to the testing provided in subsection (a) may be developed by the technical advisory committee under Section 424.

S. Section 419. Exhaust gas monitoring and control.

(a) Concentration.--In monitoring and controlling exhaust gases, the ambient concentration of exhaust gases in the mine atmosphere shall not exceed 35 parts per million for carbon monoxide and three parts per million for nitrogen dioxide. The concentration of these exhaust gases shall be measured at the equipment operator's or equipment attendant's position and by the last piece of diesel-powered equipment operating in the same split of air. Measurements shall be made weekly or more often if necessary by a qualified individual and shall be conducted under the requirements of this section.

(b) Measurement.--Measurement of exhaust gases shall be made with a sampling instrument no less precise than detector tubes.

(c) Changes.--If the concentration of a gas listed in subsection (a) is at least 75% of its exposure limit, changes to the use of the diesel equipment, the mine ventilation or the mining process shall be made.

(d) Excessive exposure.--If the concentration of a gas listed in subsection (a) exceeds the exposure limit, the diesel equipment operating in that split shall be removed from service immediately, and corrective action shall be taken. After corrective action has been taken by the mine operator, the diesel equipment may be returned to service in its regular operating mode for emissions testing purposes only; and emissions testing shall be conducted immediately to assure that the concentration does not exceed 75% of the exposure limit. Corrective action shall be taken until the concentration does not exceed 75% of the exposure limit before the diesel equipment can be returned to full operation.

(e) Compliance.--The mine operator shall comply with the following requirements:

(1) Repair or adjustment of the fuel injection system shall only be performed by qualified mechanics authorized by the engine manufacturer.

(2) Complete testing of the emissions system in accordance with section 418 shall be conducted:

(i) prior to any piece of diesel-powered equipment being put into service; and (ii) after any repair or adjustment to the fuel delivery system, engine timing or exhaust emissions control and conditioning system.

(3) Service and maintenance of the intake air filter, exhaust particulate filter and the exhaust system shall be performed at specific time intervals based on the component manufacturer's recommendation and compliance with the engine or emissions control operation specifications and, as needed, based on the on-board diagnostics or emissions test results. Accurate records shall be maintained of service and maintenance under this paragraph.

T. Section 420. Training and general requirements.

(a) Approval.--Training course instructors and training plans required by this section shall be approved by the department. Operator training and qualification shall meet the requirements of this section.

(b) Conduct.--

(1) Training shall be conducted in the basics of the operation of a diesel engine, Federal and State regulations governing their use, company rules for *safe* operation, specific features of each piece of equipment and the ability to recognize problems.

(2) Training shall be provided to each equipment operator and the mine health and safety committee if one exists. This training shall be designed to bring every operator to a level of good understanding of diesel equipment operation.

(3) Each operator shall be qualified by attending a minimum eight-hour course, including classroom training on diesel fundamentals and equipment-specific hands-on training on the job. Training shall include instruction in the following classroom subjects:

(i) Engine fundamentals. This subparagraph includes an introduction to the function of a diesel engine and recognition of major components and their functions.

(ii) Diesel regulations. This subparagraph includes an introduction to Federal and State regulations governing the use of diesel equipment.

(iii) Diesel emissions. This subparagraph includes an introduction to diesel emissions and their adverse health effects.

(iv) Factors which affect diesel emissions. This subparagraph includes a detailed presentation of engine faults and diesel fuel quality, their effect on emissions and the preventive actions which can be taken to minimize emissions levels.

(v) Emissions control devices. This subparagraph includes a detailed presentation of the different emissions control devices employed to *reduce* emissions and details about actions the operator must take to keep the devices in working order.

(vi) Diagnostic techniques. This subparagraph includes a presentation of techniques which can be employed by the operator to assure the equipment is in safe operating condition and instruction about how to recognize and diagnose certain engine faults which may cause increases in emissions.

(vii) Preoperational inspection. This subparagraph includes a presentation of the purpose, benefits and requirements of the preoperational inspection.

(viii) Ventilation. This subparagraph includes an introduction to special ventilation requirements for areas where diesel-powered equipment will operate.

(ix) Fire suppression system. This subparagraph includes an introduction to the fire suppression system and its function and when and how to activate the fire suppression manually.

(x) Operating rules. This subparagraph includes a detailed presentation of the driving rules, safe driving speeds, traffic control devices and equipment limitations.

(xi) Emergency procedures. This subparagraph includes discussion of:

(A) emergencies, such as fire, diesel fuel spills, component failure, loss of ventilation air and emergency escape procedures; and

(B) potential use of the diesel-powered vehicle as an emergency escape vehicle in case of a mine emergency.

(xii) Recordkeeping and reporting procedures. This subparagraph includes a presentation on required recordkeeping and reporting procedures for problems or unsafe conditions, high emissions levels and preoperational inspections made by the equipment operator.

(c) Certificate.--Upon successful completion of both training sessions, the operator shall be issued a certificate of qualification which qualifies the operator to operate a specific type of diesel-powered equipment. An operator may be qualified to operate more than one type of equipment by completing additional equipment-specific training covering differences specific to each additional type of equipment.

(d) Refresher training.--Refresher training, separate from that required by MSHA regulations at 30 CFR Pt. 48 (relating to the training and retraining of miners), shall be required annually.

(e) Annual certificate.--A new certificate of qualification shall be issued annually after the equipment operator has received the annual refresher training.

U. Section 421. Equipment-specific training.

(a) Approval.--Training course instructors and training plans required by this section must be approved by the department.

(b) Description.--

(1) Equipment-specific hands-on orientation training shall be given in an area of the mine where the equipment will be operated. This orientation shall be specific to the type and make of the diesel machine and shall be presented in small groups.

(2) The following subjects shall be included in the training:

(i) Equipment layout. This subparagraph includes familiarization with the layout of the equipment, the operator's compartments and the controls.

(ii) Pre-operation inspection. This subparagraph includes familiarization with the pre-operation inspection procedure and review of specific details of the inspection and location of the components to be inspected.

(iii) Equipment limitations. This subparagraph includes instruction relating to equipment performance, speeds, capacities and blind areas.

(iv) Operating areas. This subparagraph includes instruction relating to areas in which the equipment may be operated.

(v) Operation. This subparagraph includes familiarization with the controls, gauges and warning devices and safe operating limits of all indicating gauges.

(vi) Refueling procedure. This subparagraph includes familiarization with fuel handling, permissible refueling areas, spill prevention, cleanup and potential hazards from diesel fuel.

(vii) Emergency devices. This subparagraph includes instruction relating to the location and use of the fire extinguisher and fire suppression devices.

(viii) Driving practice. This paragraph includes supervised operation of the equipment.

V. Section 422. Diesel mechanic training.

(a) Approval.--Training course instructors and training plans required by this section must be approved by the department.

(b) General rule.--Diesel mechanic training and qualification shall meet the requirements of this section.

(c) Skills.--Diesel mechanics shall be trained and qualified to perform maintenance, repairs and testing of the features of diesel equipment certified by MSHA and the department.

(d) Qualification.--To be qualified, a diesel mechanic shall successfully complete a minimum of 16 hours of a training program approved by the department regarding the general function, operation, maintenance and testing of emissions control and conditioning components. The diesel mechanic shall be qualified to perform these tasks on the specific machines used at the mine or mines where they are employed. Additional engine-specific training shall be provided to diesel mechanics in accordance with a plan approved by the department.

(e) Retraining.--Annual retraining programs for diesel mechanics shall be required and shall be approved by the department. Retraining shall include refresher training as well as new procedure and new technology training as necessary. Retraining shall be separate from refresher training pursuant to MSHA regulations at 30 CFR Pt. 48 (relating to training and retraining of miners) and electrical training required by MSHA.

(f) Programs.--The minimum diesel mechanic training programs shall include training in the following minimum subject requirements:

(1) Federal and State requirements regulating the use of diesel equipment.

(2) Company policies and rules related to the use of diesel equipment.

(3) Emissions control system design and component technical training.

(4) Onboard engine performance and maintenance diagnostics system design and component technical training.

(5) Service and maintenance procedures and requirements for the emissions control systems.

(6) Emissions testing procedures and evaluation and interpretation of test results.

(7) Troubleshooting procedures for the emissions control systems.

(8) Fire protection systems test and maintenance.

(9) Fire and ignition sources and their control and elimination.

(10) Fuel system maintenance and safe fueling procedures.

(11) Intake air system design and components technical training and maintenance procedures.

(12) Engine shutdown device tests and maintenance.

(13) Special instructions regarding components, such as the fuel injection system, which may only be repaired and adjusted by a qualified mechanic who has received special training and is authorized to make the repairs or adjustments by the component manufacturer.

(14) Instruction on recordkeeping requirements for maintenance procedures and emissions testing.

(15) Other subjects determined by the department to be necessary to address specific health and safety needs.

W. Section 423. Operation of diesel-powered equipment.

(a) General rule.--In addition to other requirements of this chapter, diesel-powered equipment shall be operated pursuant to the standards set forth in this section.

(b) Attended equipment.--Diesel-powered equipment shall be attended while in operation with the engine running in underground mines.

(c) Idling.--Unnecessary idling of diesel-powered equipment is prohibited.

(d) Access.--Roadways where diesel-powered equipment is operated shall be maintained as free as practicable from bottom irregularities debris and wet or muddy conditions, which affect control of the equipment.

(e) Speed.--Operating speeds shall be consistent with conditions of roadways, grades, clearances, visibility and traffic and type of equipment used.

(f) Control.--Equipment operators shall have full control of the mobile equipment while it is in motion.

(g) Traffic rules.--Traffic rules, including speed, signals and warning signs, shall be standardized at each mine and posted.

(h) Maintenance.--

(1) Diesel-powered equipment shall be maintained in a safe operating condition which does not threaten health of human beings.

(2) Diesel-powered equipment not maintained in accordance with paragraph (1) or not maintained in accordance with the engine or emissions control operating specifications shall be removed from service immediately and shall not be returned to service until all necessary corrective actions have been taken.

X. Section 424. Technical advisory committee.

(a) Establishment.--The Technical Advisory Committee on Diesel-Powered Equipment is established.

(b) Membership.--The advisory committee shall consist of two members, who shall be residents of this Commonwealth.

(1) The Governor shall appoint one member to represent the viewpoint of the coal operators in this Commonwealth within 30 days from receipt of a list containing one or more nominees submitted by the major trade association representing coal operators in this Commonwealth.

(2) The Governor shall appoint one member to represent the viewpoint of the working miners in this Commonwealth within 30 days from receipt of a list containing one or more nominees submitted by the highest ranking official within the major employee organization representing coal miners in this Commonwealth.

(c) Terms.--Each member of the technical advisory committee shall be appointed for a term of three years. If renominated and reappointed, a member may serve an unlimited number of successive three-year terms.

(d) Functions.--The technical advisory committee has the following functions:

(1) Advising the department regarding implementation of this chapter.

(2) Evaluating alternative technology or methods for meeting the requirements for diesel-powered equipment as set forth in this chapter.

(3) Providing technical assistance to operators regarding diesel equipment technologies.

(4) Conducting investigations relating to implementation of this chapter.

(5) Providing training regarding diesel equipment emission controls and emission testing.

(e) Compensation.--Members of the technical advisory committee shall be compensated at the appropriate per diem rate based on the prevailing formula administered by the Commonwealth, but not less than $150 per day, plus all reasonable expenses incurred while performing their official duties. Compensation shall be adjusted annually by the department to account for inflation based on the rate of inflation identified by the Consumer Price Index for All Urban Consumers, Bureau of Labor Statistics. The individual member may waive his right to all or part of the compensation set forth in this provision.

(f) Meetings.--The technical advisory committee shall meet at least twice during each calendar year.

(g) Quorum.--Actions of the technical advisory committee require the participation of both members.

(h) Support.--

(1) The department shall make clerical support and assistance available to enable the technical advisory committee to carry out its duties. Upon the request of both members of the technical advisory committee, the department may draft proposed conditions of use and reports or perform investigations.

(2) The department shall purchase for the technical advisory committee equipment for testing diesel engine exhaust emissions and measuring diesel engine surface temperatures and exhaust gas temperatures. Alternative technology or methods recommended by the technical advisory committee or approved by the secretary shall not reduce or compromise the level of health and safety protection afforded by this chapter.

(i) Alternative technologies.--

(1) Upon application of a coal miner, coal mine operator or diesel-related technology manufacturer, or on its own motion, the technical advisory committee shall consider requests for the use of alternative diesel-related health and safety technologies with general underground mining industry application which are consistent with this chapter. The following apply:

(i) Upon receipt of an application, the technical advisory committee shall conduct an investigation, which shall include consultation with a representative of the major trade association representing coal operators in this Commonwealth and with a representative of the major employee organization representing coal miners in this Commonwealth.

(ii) Approval of an application made under this subsection shall make the alternative technology or method available for use by a coal mine operator in this Commonwealth but shall not be construed to require that a coal mine operator use the approved alternative technology or method.

(2) Upon application of a coal mine operator, the technical advisory committee shall consider site-specific requests for use of alternative diesel-related health and safety technologies. The committee's recommendations on applications submitted under this subsection shall be on a mine-by-mine basis. Upon receipt of a site-specific application, the technical advisory committee shall conduct an investigation, which shall include consultation with the mine operator and the authorized representatives of the miners at the mine. Authorized representatives of the miners shall include a mine health and safety committee elected by miners at the mine and an individual employed by an employee organization representing miners at the mine or an individual authorized as the representative of miners of the mine in accordance with MSHA regulations at 30 CFR Pt. 40 (relating to representative of miners). If there is no authorized representative of the miners, the technical advisory committee shall consult with a reasonable number of miners at the mine.

(3) Within 180 days of receipt of an application for use of alternative technologies or methods, the technical advisory committee shall complete its investigation and make a recommendation to the secretary. The technical advisory committee members shall only recommend approval of an application if, at the conclusion of the investigation, the committee members have made a determination that the use of the alternative technology or method will

not reduce or compromise the level of health and safety protection afforded by this chapter. The time period under this paragraph may be extended with the consent of the applicant.

(4) The technical advisory committee shall forward to the secretary three possible recommendations:

(i) A unanimous recommendation to approve the application for use of alternative technologies or methods. A recommendation under this subparagraph must be made in writing and include the results of the investigation and specific conditions of use for the alternative technology or method.

(ii) A unanimous recommendation to reject the application for use of alternative technologies or must be made in writing and outline in detail the basis for the rejection.

(iii) A divided recommendation in which one member of the technical advisory committee recommends approval of the application for use of alternative technologies or methods and one member of the advisory committee recommends rejection of the application for use of alternative technologies or methods. For a recommendation under this subparagraph, each member of the committee must submit a detailed report to the secretary within 14 days of the committee's vote outlining the member's position for or against the application.

(5) The secretary shall proceed as follows:

(i) Alternative technologies or methods may be approved by the secretary if they do not reduce or compromise the level of health and safety protection afforded by this chapter.

(ii) If a recommendation under paragraph (4)(i) or (ii) is forwarded to the secretary by the technical advisory committee, the secretary shall have 30 days in which to render a final decision adopting or rejecting the advisory committee's recommendation and the application.

(iii) The secretary may only approve or reject a recommendation under paragraph (4)(i) or (ii) without modification unless the modification is unanimously approved by the technical advisory committee.

(iv) If a recommendation under paragraph (4)(iii) is forwarded to the secretary, the secretary shall convene, within 30 days, a meeting with the members of the technical advisory committee to discuss the reasons for the divided recommendation and to determine whether additional information and further discussion might result in a unanimous recommendation by the committee.

(v) The following apply:

(A) The secretary shall render a decision on the application within 30 days from the date of the meeting with the technical advisory committee or, if no meeting is convened, within 60 days of forwarding of the recommendation.

(B) Upon consent of the applicant, the time period under clause (A) may be extended.

(C) Except as set forth in clause (B), if the secretary does not comply with the time requirements to render a decision under this subparagraph, the technical advisory committee's recommendation shall be deemed rejected.

(6) Action taken by the secretary under this subsection is subject to 2 Pa.C.S. Ch. 7 Subch. A (relating to judicial review of Commonwealth agency action) and the act of July 13, 1988 (P.L.530, No.94), known as the Environmental Hearing Board Act.

(j) Shaft and slope construction.--The secretary shall establish, based on recommendations made by the technical advisory committee, conditions of use for the use of diesel-powered equipment in shaft and slope construction operations at coal mines. Conditions of use proposed by the technical advisory committee shall be considered by the secretary and shall be adopted or rejected by the secretary without modification, except as approved by the technical advisory committee.

Reference

Commonwealth of Pennsylvania. Department of Environmental Protection. Safety Laws for Underground Bituminous Coal Mines. Act 55, SB 949 Session of 2008. Effective Date January 3, 2009. 2nd Printing (2/3/2009).

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 6

PENNSYLVANIA DIESEL LAW

QUESTIONS

Note: answers must be verbatim with Chapter 4 of the Pennsylvania Safety Laws

1. Chapter 4 of the Pennsylvania Safety Laws covers the use ____________ _________________.

    a. of permissible and non-permissible equipment
    b. of diesel powered equipment
    c. of both Heavy duty and light duty equipment
    d. of both inby and outby equipment

2. Chapter 4 of the Pennsylvania Safety Laws covers all the following equipment except:______________________________

    a. permissible diesel powered equipment,
    b. equipment with horsepower ratings less than 30
    c. non-permissible stationary equipment,
    d. emergency fire-fighting equipment

3. All diesel-powered equipment shall be attended while in operation with the engine running in underground mines. For the purpose of this subsection attended shall mean an equipment operator is ______________________________________.

    a. within sight of the equipment
    b. within sight or sound of the equipment
    c. within sound of the equipment
    d. at the controls

4. Diesel-powered equipment must use an engine that is:_________________.

    a. approved by MSHA
    b. approved by the Department
    c. certified by MSHA
    d. Both a & c

5. All diesel-powered equipment shall be approved by the department:

    a. with and approved engine_____________________________.
    b. as a complete diesel powered equipment package
    c. with an approved after treatment system
    d. both a & c

6. Exhaust emissions control systems. -- Underground diesel-powered equipment shall include an exhaust emissions:____________________________________.
   a. control and conditioning system
   b. DPM after treatment system
   c. DPM filter
   d. Both b & c

7. Diesel particulate matter emissions cannot exceed an average concentration of____________________
   a. 0.12 mg/m^3
   b. 1.2 g/m^3
   c. 120 u/m^3
   d 01.2 $mg/\ m^3$

8. Exhaust emissions control systems. -- An exhaust emissions control and conditioning system may be approved for multiple diesel engine applications through a single series of laboratory tests, known as:____________________________.
   a. the MSHA underground engine approval test
   b. test the ISO 8178-1 test
   c. the ISO 8 mode
   d. the EPA engine approval test

9. Components of exhaust emissions system.- Diesel powered equipment must use a diesel particulate matter (DPM) filter that has proven capable of a reduction in total diesel particulate matter to _______________________________ subsection (a)(1).
   a. a seventy five percent level of
   b. a ninety five percent level of
   c. to a level that does not exceed the requirements of
   d. a minimum level required by

10. Components of exhaust emissions system. - All systems must include an oxidation catalyst or other gaseous emissions control device capable of reducing undiluted carbon monoxide emissions to__________ or less
   a. 100 parts per million
   b. 302 parts per million
   c. 50 parts per million
   d. 35 parts per million

11. Components of exhaust emissions system. - All systems must include an engine surface temperature control capable of maintaining significant external surface temperatures below_____________________________.:
   a. 302 degrees Fahrenheit
   b. the ignition temperature of diesel fuel
   c. the ignition temperature of methane
   d 306 degrees Fahrenheit

12. Components of exhaust emissions system - All systems must include an automatic engine shutdown system that will shut off the engine before the exhaust gas temperature reaches: ___________________________.

a. 306 degrees Fahrenheit
b. the ignition temperature of diesel fuel
c. the ignition temperature of methane
d 302 degrees Fahrenheit

13. Components of exhaust emissions system- All systems must include:

a. a spark arrestor
b. a sampling port for measurement of undiluted and untreated exhaust gases
c. a flame arrestor
d. all of the above

14. Onboard engine performance and maintenance diagnostics systems shall be capable of continuously monitoring and giving readouts for:_____________________________.

a. transmission pressure
b. PTO pressure
c. transmissions temperature
d. none of the above

15. Ventilation - The minimum quantities of air in any split where any individual unit of diesel-powered equipment is being operated shall be at least: _______________________.

a. 6000 cubic feet per minute
b. that specified by the MSHA required ventilation rate
c. that specified on the approval plate for that equipment
d. 9000 cubic feet per minute

16. Ventilation. - Air quantity measurements to determine compliance with this requirement shall: _______________________________________.

a. be taken on each air split
b. not include any common air
c. be made at the individual unit
d. not include any neutral air

17. An underground diesel fuel storage facility shall be any facility designed and constructed to provide ____ of any mobile diesel fuel transportation units or the dispensing of diesel fuel.

a. for the temporary storage
b. for the storage
c. for the permanent storage
d. all the above

18. __________________________________of diesel fuel shall be stored in each underground diesel fuel storage facility.

a. No more than 1000 gallons
b. No more than 500 gallons
c. No more than two 250 gallon units
d. No more than one five gallon safety can

19. Underground diesel fuel storage facilities shall be located as follows: _________________.

a. at least 50 feet from shafts
b. in an area as that is as dry as practicable
c. at least 25 feet from trolley wires
d. both a & c

20. Underground diesel fuel storage facilities shall meet the following:

a. constructed of noncombustible materials
b. ventilated directly into the return air course
c. equipped with a fire suppression system
d. all of the above

21. Diesel fuel shall not be transferred using:

a. a powered pump requiring more than 12 volts of electrical power
b. compressed gas
c. a nozzle containing a self-closing valve without a latch-open device;
d. a dry pipe fuel distribution system

22. Unburied diesel fuel pipelines shall not exceed ______________________ and shall have shutoff valves located at each end of the unburied pipeline.

a. 500 feet in length
b. 250 feet in length
c. 300 feet in length
d. 302 feet in length

23. Diesel fuel pipelines __________________________________, except that they may cross the entry perpendicular if buried or otherwise protected from damage and sealed.

a. shall not be located on the same side as trolley wire
b. shall not be located within 25 feet of trolley wire
c shall not be located in any trolley haulage entry
d. shall not be permitted in trolley haulage entries unless the trolley wire has first been de-energized

24. Diesel fuel shall be transported only in:___________________________.

a. MSHA approved diesel fuel transportation units
b. diesel fuel transportation units approved by the Bureau
c. containers specifically designed for the transport of diesel fuel
d any of the above

25. Containers. - Shutoff valves must be _________________ on each connection through which liquid can normally flow.

a. located within one inch of the tank shell
b. located as close as practicable to the tank shell
c. located within one half inch of the tank shell
d. located on the accessible side of the tank shell

26. Containers used for the transport of diesel fuel shall not exceed______________________.

a. the capacity set forth is this subsection
b. the capacity of the fire suppressant system located of the mobile unit
c. a capacity of 500 gallons
d. a capacity of 1000 gallons

27. Containers, other than safety cans, used for the transport of diesel fuel ___________.

a. shall be permanently fixed to the transportation unit
b. shall be approved by MSHA
c. shall be approved by the Bureau
d shall be approved by the department

28. Diesel fuel transportation units:____________________________________.

a. shall be transported separately and not with any other cars
b. shall be transported individually and not with any other cars
c. not be transported within five minutes of any mantrip
d. shall not be transported by means of trolley locomotives

39. Unattended diesel fuel transportation units:__________________________.

a. shall be removed from the mine forthwith
b. shall not be allowed in the underground area of the mine
c. shall be parked in a side track designated for diesel fuel storage.
d. shall be parked only in an underground diesel fuel storage facility

30. Safety cans shall be used ____________ .

a. by qualified diesel equipment operator
b. for equipment fueling only
c. for emergency fueling only
d. only when necessary

31. Fire suppression for equipment and transportation- The fire suppression system shall provide for automatic engine shutdown. Engine shutdown and discharge of suppressant agent may be delayed for a maximum of ___________ after the fire alarm annunciator alerts the operator.

a. a minimum of 10 seconds
b. a maximum of 10 seconds
c. a maximum of 15 seconds
d. a minimum of 15 seconds

32. Fire suppression for equipment and transportation - ________________________ actuators shall be provided with at least one manual actuator at each end of the equipment. If the equipment is provided with an operator's compartment, one of the mechanical actuators shall be located in the compartment within easy reach of the operator.

a. At least two manual
b. At least three manual
c. No less than three manual
d. A sufficient number of manual

33. Fire suppression for storage areas - Fire suppression systems shall include two manual actuators with _________________ _____________ the fuel storage facility and at least one located ____________________________ from the storage facility and in intake air.

a. at least one located within, a safe distance away
b. with at least one located adjacent to, fifty feet away
c. with at least one located within, one hundred feet away
d. with at least one located adjacent to, a safe distance away

34. The use of ________________ prohibited.

a. starting fluid is
b. all starting aids are
c. chemical starting aids are
d. volatile or chemical starting aids is

35. Fueling of diesel-powered equipment shall not_________________________ escape-way unless the mine design and entry configuration make it necessary

a. be conducted in the primary intake
b. be conducted in the intake
c. be conducted in any designated intake
d. be conducted in either the primary or secondary intake

36. All used intake air filters, exhaust diesel particulate matter filters and engine oil filters ___________________ ____________ containers and removed from the underground mine to the surface.

a. shall be repacked, in their original containers or other substantial enclosed
b. shall be replaced in their original containers or other suitable enclosed
c. shall be placed, in their original containers or other suitable
d. shall be sealed, in their original containers or other available enclosed

37. Diesel-powered equipment shall be______________________________________ as described in this article or removed from service.

a. maintained in an approved and healthful condition
b. maintained in an approved and safe condition
c. maintained in a safe and healthful condition
d. maintained in a workmen like manner and good operating condition

38. Upon receiving a notice of revocation, the ____________ __________ to submit a plan to achieve and maintain compliance.

a. mine superintendent shall have thirty days
b. mine operator shall have sixty days
c. mine superintendent shall have sixty days
d. mine operator shall have thirty days

39. All service, maintenance and repairs of approved complete diesel-powered equipment packages shall be performed by mechanics who are ______________________________.

a. designated by the operator and qualified in accordance Section 422
b. trained and certified in accordance with the requirements of Chapter 4
c. trained and qualified in accordance with Section 422
d. designated by the operator and certified in accordance with Chapter 4

40. Service and maintenance of approved complete diesel-powered equipment packages shall be performed according to:

a. the specified routine maintenance schedule
b .the on-board performance and maintenance diagnostics readings
c. the emissions test results; component manufacturer's recommendations
d. all the above

41. Records -. The person performing the emissions test, examination, maintenance or repair shall certify by _______________________________ that the emissions test, examination, maintenance or repair was made.

a. date, time and initial
b. date, time and engine serial number
c. date, time, engine hour reading and signature
d. date, time, engine hour reading, equipment number and signature

42. Records of preoperational examinations shall be retained for:

a. at least one year and stored at a location outside of the mine
b. the two previous one hundred hour maintenance cycles
c. the two prior one hundred hour maintenance cycles
d. the pervious one hundred hour maintenance cycle

43. Records of emissions tests, one hundred-hour maintenance tests and repairs shall be countersigned once each week by the:______________________________.

a. certified mine electrician and mine foreman
b. certified mine electrician or mine foreman
c. mine superintendent and mine foreman
d. mine superintendent or mine foreman

44. Preoperational examination- Prior to use of a piece of diesel-powered equipment during a shift, ____________________________ as follows:

a. the trained equipment operator shall conduct an examination
b. any trained diesel powered equipment operator shall conduct an examination
c. an equipment operator shall conduct an examination
d. a qualified equipment operator shall conduct an examination

45. The diesel powered-equipment pre-operational examination requires ________.

a. requires 5 steps prior to engine start-up.
b. requires 10 steps prior to engine start-up.
c. requires 12 steps prior to engine start-up.
d. requires 15 steps prior to engine start-up.

46. Schedule of maintenance- At intervals _________________________ engine operation, a qualified mechanic shall perform the following maintenance and make all necessary adjustments or repairs or remove the equipment from service:

a. of one hundred hours of underground
b. not exceeding one hundred hours of
c. within two hundred hours of
d. not exceeding one hundred hours of underground

47. When any diesel-powered machine first enters service at a mine, ________________ determined by a qualified mechanic.

a. carbon monoxide (CO) emission values shall be
b. baseline emission values shall be
c. all gaseous emission values shall be
b. primary emission values shall be

48. When performing an emissions test the qualified mechanic shall: Allow the engine to operate for a ________________, to insure proper carbon monoxide readings and record both carbon monoxide and carbon dioxide readings

a. period time, that is at least 90 seconds duration,
b. sufficient time, not less than 90 seconds duration
c. period of time, of not more than 90 seconds duration
d. sufficient length of time

49. Diagnostic testing -. The qualified mechanic shall do the following: Put the engine into a _____________________________.

a. torque converter stall condition
b. loaded condition
c. full throttle condition
d. high idle condition

50. If the average CO reading for untreated exhaust gas is greater than _________________________ under section 417-A(b) or if the average CO reading for treated exhaust gas is greater than __________, the equipment has failed and must be serviced and retested before it is returned to regular service;

a. fifty percent of the baseline established, 100 parts per million
b. one hundred percent of the baseline established, 50 parts per million
c. twice the baseline established, 100 parts per millions
d. the baseline established, 50 parts per millions

51. The ambient concentration of exhaust gases in the mine atmosphere shall not exceed__________ for carbon monoxide and ___________ for nitrogen dioxide.

a. 35 parts per million, three parts per million
b. 35 parts per million, five parts per million
c. 50 parts per million, three parts per million
d. 50 parts per million, and, five parts per million

52. Ambient exhaust: Measurements shall be made __________ or more often if ________ individual and shall be conducted under the requirements of this section.

a. daily or more often if necessary by a, certified
b. weekly or more often if necessary by a, qualified
c. daily or more often if necessary by a, qualified
d. weekly or more often if necessary by a, certified

53. Measurement of exhaust gases shall be made with a____________________ no less precise than detector tubes.

a. hand held gas detecting device
b. portable gas detecting device
c. sampling instrument
d. multi gas detecting device

54. If the concentration of any of the gases listed in subsection (a) __________________ __________________, changes to the use of the diesel equipment, the mine ventilation or other modifications to the mining process shall be made.

a. is 50 percent or more of its exposure limit
b. is 75percent or more of its exposure limit
c. is 100 percent or more of its exposure limit
d. above the Threshold Limit Value set in this subsection

55. Repair or adjustment of the fuel injection system shall only be performed by qualified mechanics ___________________________________.

a. trained by the engine manufacturer
b. authorized by MSHA
c. trained by the Department
d. authorized by the engine manufacturer

56. Approval.--Training course instructors and training plans required by this section shall be approved ___________________________.

a. by the Secretary
b. by the Bureau
c. by the department
d. by the Coal Operator

57. Diesel operator- Refresher training.—Refresher training, separate from that required by MSHA regulations at 30 CFR Pt. 48 (relating to the training and retraining of miners), shall be _________________________________.

a. completed at least once every 12 months
b. required annually
c. required at the discretion of the department
d. completed on a yearly basis

58. Equipment-specific hands-on orientation training shall be given __________________________ where the equipment will be operated.

a. in an area of the mine
b. in an underground area of the mine
c. safe and secure area
d. any area of the mine

59. Operation of diesel-powered equipment - Roadways where diesel-powered equipment is operated shall be maintained as free as practicable from ___________________, which affect control of the equipment.

a. bottom irregularities
b. wet and muddy conditions
c. debris
d. all of the above

60. Traffic rules.--Traffic rules, including speed, signals and warning signs, shall be ________________ at each mine and posted.

a. standardized
b. governed by the mine foreman
c. governed by the department
d. governed by the mine superintendent

61. Technical Advisory Committee. - The advisory committee shall consist of ______________ who shall be residents of this Commonwealth.

a. sufficient number members
b. three members
c. two members
d. a contingent of members

62. Technical advisory committee Terms.—Each member of the technical advisory committee shall be appointed for a term of _______________

a. one year
b. two years
c. three years
d. four years

63. Technical advisory committee - ________________________, the technical advisory committee shall consider requests for the use of alternative diesel-related health and safety technologies with general underground mining industry application which are consistent with this chapter.

a. Upon application of a coal miner, coal mine operator
b. Upon application of a diesel-related technology manufacturer
c. On its own motion
d. all of the above

64. Technical advisory committee –Within ____________________ of an application for use of alternative technologies or methods, the technical advisory committee shall complete its investigation and make a recommendation to the secretary.

a. 30 days of receipt
b. 60 days of receipt
c. 90 days of receipt
d 180 days of receipt

65. Required certifications or approvals.--Inby and outby diesel-powered equipment may be used in underground mines if the inby or outby diesel-powered equipment uses an engine approved or certified by MSHA, as applicable, for inby or outby use that, when tested at the maximum fuel-air ratio, does not require a MSHA Part 7 approval plate ventilation rate exceeding _________________________.

a. the ventilating rate set by the department
b. 50 c.f.m.. per rated horsepower
c. 75 c.f.m.. per rated horsepower
d. the guidelines set forth in this subsection.

66. Diesel-powered equipment package. Approval.--All diesel-powered equipment shall be approved by the department as _________________ which shall be subject to all of the requirements, standards and procedures set forth under this chapter.

a. a complete diesel powered equipment package
b. a complete unit of diesel powered equipment
c. a complete piece of diesel powered equipment
d. a complete underground diesel powered equipment package

67. Exhaust emissions control -. Except as provided in paragraph (3), the exhaust emissions control and conditioning system shall be required to successfully complete a single series of laboratory tests for each diesel engine, conducted at a ___________________________.

a. laboratory certified by MSHA
b. laboratory certified by the department
c. laboratory approved by the department
d. laboratory accepted by the department

68. Approvals.--Each specific model of diesel-powered equipment shall be approved by the department _________________________________

a. before it is put into service
b. before it is taken underground
c. before it performs any work at the mine
d. before it can be used.

69. Ventilation - Multiple units in operation.--Where multiple units are operated, the minimum quantity shall be at least the total of ______________ approval plate ventilation rate for each unit operating in that split.

a.100% of MSHA's Part 7
b.100% of the department's
c. Pennsylvania's
d. the most stringent

70. Ventilation - The department shall require that an approval plate be attached to each piece of the diesel-powered equipment. The approval plate shall specify _____________ for the specific piece of diesel-powered equipment.

a. the minimum ventilating air quantity
b. the department's approval plate air quantity
c. the maximum ventilating air quantity
d. the MSHA approval plate air quantity

71. Diesel fuel standards.--Additionally, the fuel shall also meet the ASTM D975 standards with _________________________ Fahrenheit or greater at standard temperature and pressure.

a. an auto ignition temperature of 650 degrees
b. a flash point of 100 degrees
c. a flash point of 125 degrees
d. an auto ignition temperature of at least 302 degrees

72. Fuel storage facilities - Diesel fuel shall not be allowed to enter pipelines or containers that have been welded, soldered, brazed or cut until the metal has cooled to_________________.

a. ambient temperature
b. less than 302 degrees Fahrenheit
c. less than its flash point
d. less than 100 degrees Fahrenheit

73. Fuel storage facilities - Welding or cutting other than that performed in accordance with paragraph (4) shall not be done within ______________ of a diesel fuel storage facility.

a. 50 feet
b. 100 feet
c. 150 feet
d. 25 feet

74. Standards for safety cans.—Safety cans shall be clearly marked, _______________________.

a. have a maximum capacity of five gallons
b. be constructed of metal
c. be equipped with a nozzle and self-closing valves.
d. all of the above

75. Fire suppression for storage areas. - Types of alarms.--Audible and visual alarms to warn of fire or system faults shall be provided ________________________ always staffed when individuals are underground

a. in the storage facility and other areas of the mine that are
b. outside the storage facility and all areas of the mine that are
c. at the protected area and at a surface location that is
d. to a surface location that is

76. Trained individual on duty.--At least one individual ______________ and disposal of diesel fuel spills shall be on duty at the mine when diesel-powered equipment or mobile fuel transportation equipment is being used or when any fueling of diesel-powered equipment is being conducted.

a. certified in the cleanup
b. hazmat trained in the cleanup
c. specially trained in the cleanup
d. who has been qualified in the cleanup

77. Schedule of maintenance. If the equipment is approved with a non-disposable diesel particulate filter, a smoke dot test of the filtered exhaust must be performed at this time. The results of the smoke dot test shall be recorded on the 100-hour emissions form. If the interpreted smoke dot number __________________, the technical advisory committee shall be notified and shall investigate to determine if the filter is functioning properly.

a. is greater than two
b is greater than three
c. is greater than four
d. is greater than five

78. Fuel storage facilities- Must be equipped with at least _______________multipurpose dry-chemical type fire extinguishers.

a. one portable 20 pound
b. one portable 10 pound
c. two portable 20 pound
d. two portable 10 pound

79. Pump transfers.--When diesel fuel is transferred by means of a pump and a hose equipped with a nozzle containing a self-closing valve, a powered pump may be used only if: the pump _______________________________________.

a. is equipped with an accessible emergency shutoff switch
b. is provided with an on-off switch that is within easy reach
c. uses no more than 12 volts of electricity
d .is fully grounded to the electrical system

80. Diesel fuel piping systems - Horizontal pipeline prohibition.--Horizontal pipelines shall not be used ______________________.

a. to distribute fuel throughout a mine
b. to distribute fuel to more than one fuel storage unit
c. to fuel diesel powered equipment
d. in trolley wire mines

81. Limitations where trolley wires are present.—In mines where trolley wire is used, diesel fuel transportation units shall be provided with insulating material to protect the units from any energized trolley wire, and the distance between the diesel fuel transportation unit and the trolley wire shall ______________, or the trolley wire shall be de-energized when diesel fuel transportation units are transported through the area.

a. be at least 24 inches
b. not be less than 12 inches
c. not be less than 15 inches
d. be more than 12 inches

82. Fire suppression devices shall be visually inspected _________________ by an individual qualified to make the inspection.

a. at 30 day intervals
b. at least bi-annually
c. at least once each week
d. at 120 day intervals

83. Diesel fuel standards. - The operator shall maintain ______________________ to verify that the fuel used underground meets this standard.

a. a copy of the most recent delivery receipt from the supplier
b. a copy of a fuel analyses by a laboratory excepted by the department
c. a copy of a fuel analyses by a laboratory approved by the department
d. a copy of a fuel analyses by an independent laboratory

84. Underground diesel fuel storage facilities shall meet all of the following: Be marked with conspicuous signs designating________________________.

a. flammable liquid storage
b. combustible liquid storage
c. diesel fuel storage
d. volatile liquid storage

85. Underground diesel fuel storage facilities shall: Be included in _____________________________.

a. the pre-shift examination
b. the on-shift examination
c. the required weekly examination
d. all required underground examinations

DIESEL ENGINES AND AFTER-TREATMENT SYSTEMS IN UNDERGROUND BITUMINOUS MINES SECTION 7

PENNSYLVANIA DIESEL LAW

QUESTION ANSWER KEY

Note: answers must be verbatim with Chapter 4 of the Pennsylvania Safety Laws

3. Chapter 4 of the Pennsylvania Safety Laws covers the use ____________ _________________.

   a. of permissible and non-permissible equipment
   b. of diesel powered equipment
   c. of both Heavy duty and light duty equipment
   d. of both inby and outby equipment - 401(a)

4. Chapter 4 of the Pennsylvania Safety Laws covers all the following equipment except:_____________________________
   a. permissible diesel powered equipment,
   b. equipment with horsepower ratings less than 30
   c. non-permissible stationary equipment,
   d. emergency fire-fighting equipment - 401(a)

3. All diesel-powered equipment shall be attended while in operation with the engine running in underground mines. For the purpose of this subsection attended shall mean an equipment operator is _____________________________________.
   a. within sight of the equipment
   b. within sight or sound of the equipment - 401(b)
   c. within sound of the equipment
   d. at the controls

4. Diesel-powered equipment must use an engine that is:_________________.
   a. approved by MSHA
   b. approved by the Department
   c. certified by MSHA
   d. Both a & c - 401(c)

9. All diesel-powered equipment shall be approved by the department:
   a. with and approved engine____________________________.
   b. as a complete diesel powered equipment package- 402(a)
   c. with an approved after treatment system
   d. both a & c

10. Exhaust emissions control systems. - Underground diesel-powered equipment shall include an exhaust emissions:_________________________________.

a. control and conditioning system - 403(a)(1)

b. DPM aftertreatment system

c. DPM filter

d. Both B & C

11. Diesel particulate matter emissions cannot exceed an average concentration of___________________.:

a. 0.12mg/m^{3} - 403(a)(1)

b. 1.2 g/m^3

c. 120 *u*/m^3

d 01.2 mg/ m^3

12. Exhaust emissions control systems. -- An exhaust emissions control and conditioning system may be approved for multiple diesel engine applications through a single series of laboratory tests, known as:_________________________.

a. the MSHA underground engine approval test

b. **test the ISO 8178-1 test - 403(a)(3)**

c. the ISO 8 mode

d. the EPA engine approval test

9. Components of exhaust emissions system.- Diesel powered equipment must use a diesel particulate matter (DPM) filter that has proven capable of a reduction in total diesel particulate matter to ____________________________ subsection (a)(1).

a. a seventy five percent level of

b. a ninety five percent level of

c. to a level that does not exceed the requirements of - 403(b)(1)

d. a minimum level required by

10. Components of exhaust emissions system. - All systems must include an oxidation catalyst or other gaseous emissions control device capable of reducing undiluted carbon monoxide emissions to__________ or less

a. 100 parts per million - 403(b)(2)

b. 302 parts per million

c. 50 parts per million

d. 35 parts per million

11. Components of exhaust emissions system. - All systems must include an engine surface temperature control capable of maintaining significant external surface temperatures below__________________________.:

a. 302 degrees Fahrenheit - 403(b)(3)

b. the ignition temperature of diesel fuel

c. the ignition temperature of methane

d 306 degrees Fahrenheit

12. Components of exhaust emissions system - All systems must include an automatic engine shutdown system that will shut off the engine before the exhaust gas temperature reaches: ___________________________.

a. 306 degrees Fahrenheit
b. the ignition temperature of diesel fuel
c. the ignition temperature of methane
<u>d 302 degrees Fahrenheit - 403(b)(4)</u>

13. Components of exhaust emissions system- All systems must include:

a. a spark arrestor
b. a sampling port for measurement of undiluted and untreated exhaust gases
c. a flame arrestor
<u>d. all of the above - 403(b) (6)(7)(8)</u>

14. Onboard engine performance and maintenance diagnostics systems shall be capable of continuously monitoring and giving readouts for:_____________________________.

a. transmission pressure
b. PTO pressure
c. transmissions temperature
<u>d. none of the above - 403 403(c)</u>

15. Ventilation - The minimum quantities of air in any split where any individual unit of diesel-powered equipment is being operated shall be at least: _______________________.

a. 6000 cubic feet per minute
b. that specified by the MSHA required ventilation rate
<u>c. that specified on the approval plate for that equipment - 404 (c)</u>
d. 9000 cubic feet per minute

16. Ventilation. - Air quantity measurements to determine compliance with this requirement shall:_______________________________________.

a. be taken on each air split
b. not include any common air
<u>c. be made at the individual unit - 404 (c)</u>
d. not include any neutral air

17. An underground diesel fuel storage facility shall be any facility designed and constructed to provide ____ of any mobile diesel fuel transportation units or the dispensing of diesel fuel.

a. for the temporary storage
<u>b. for the storage - 405(a)</u>
c. for the permanent storage
d. all the above

18. ________________________________of diesel fuel shall be stored in each underground diesel fuel storage facility.

a. No more than 1000 gallons

<u>b. No more than 500 gallons - 405(c)(2)</u>

c. No more than two 250 gallon units

d. No more than one five gallon safety can

19. Underground diesel fuel storage facilities shall be located as follows: ________________.

a. at least 50 feet from shafts

b. in an area as that is as dry as practicable

c. at least 25 feet from trolley wires

<u>d. both a & c - 405(d) (2)(3)</u>

20. Underground diesel fuel storage facilities shall meet the following:

a. constructed of noncombustible materials

b. ventilated directly into the return air course

c. equipped with a fire suppression system

<u>d. all of the above - 405(e) (2)(i)(ii)</u>

21. Diesel fuel shall not be transferred using:

a. a powered pump requiring more than 12 volts of electrical power

<u>b. compressed gas - 406 (c)</u>

c. a nozzle containing a self-closing valve without a latch-open device;

d. a dry pipe fuel distribution system

22. Unburied diesel fuel pipelines shall not exceed ____________________ and shall have shutoff valves located at each end of the unburied pipeline.

a. 500 feet in length

b. 250 feet in length

<u>c. 300 feet in length - 406 (h)</u>

d. 302 feet in length

23. Diesel fuel pipelines _______________________________, except that they may cross the entry perpendicular if buried or otherwise protected from damage and sealed.

a. shall not be located on the same side as trolley wire

b. shall not be located within 25 feet of trolley wire

<u>c shall not be located in any trolley haulage entry - 406(n)</u>

d. shall not be permitted in trolley haulage entries unless the trolley wire has first been de-energized

24. Diesel fuel shall be transported only in:_________________________.

a. MSHA approved diesel fuel transportation units

b. diesel fuel transportation units approved by the Bureau

<u>c. containers specifically designed for the transport of diesel fuel - 407 (b)</u>

d any of the above

25. Containers. - Shutoff valves must be __________________ on each connection through which liquid can normally flow.

a. located within one inch of the tank shell - 407(d)(5)

b. located as close as practicable to the tank shell

c. located within one half inch of the tank shell

d. located on the accessible side of the tank shell

26. Containers used for the transport of diesel fuel shall not exceed________________________.

a. the capacity set forth is this subsection

b. the capacity of the fire suppressant system located of the mobile unit

c. a capacity of 500 gallons - 407 (f)

d. a capacity of 1000 gallons

27. Containers, other than safety cans, used for the transport of diesel fuel ____________.

a. shall be permanently fixed to the transportation unit - 407 (g)

b. shall be approved by MSHA

c. shall be approved by the Bureau

d shall be approved by the department

28. Diesel fuel transportation units:____.

a. shall be transported separately and not with any other cars

b. shall be transported individually and not with any other cars - 407(h)

c. not be transported within five minutes of any mantrip

d. shall not be transported by means of trolley locomotives

29. Unattended diesel fuel transportation units:____________________________.

a. shall be removed from the mine forthwith

b. shall not be allowed in the underground area of the mine

c. shall be parked in a side track designated for diesel fuel storage.

d. shall be parked only in an underground diesel fuel storage facility- 407 (m)

30. Safety cans shall be used _____________ .

a. by qualified diesel equipment operator

b. for equipment fueling only

c. for emergency fueling only - 407 (n)

d. only when necessary

31. Fire suppression for equipment and transportation- The fire suppression system shall provide for automatic engine shutdown. Engine shutdown and discharge of suppressant agent may be delayed for a maximum of ____________ after the fire alarm annunciator alerts the operator.

a. a minimum of 10 seconds

b. a maximum of 10 seconds

c. a maximum of 15 seconds - 408 (e)

d. a minimum of 15 seconds

32. Fire suppression for equipment and transportation - ________________________ actuators shall be provided with at least one manual actuator at each end of the equipment. If the equipment is provided with an operator's compartment, one of the mechanical actuators shall be located in the compartment within easy reach of the operator.

<u>a. At least two manual - 408(f)</u>
b. At least three manual
c. No less than three manual
d. A sufficient number of manual

33. Fire suppression for storage areas - Fire suppression systems shall include two manual actuators with _________________ _____________ the fuel storage facility and at least one located ____________________________ from the storage facility and in intake air.

<u>a. at least one located within, a safe distance away - 409(e)</u>
b. with at least one located adjacent to, fifty feet away
c. with at least one located within, one hundred feet away
d. with at least one located adjacent to, a safe distance away

34. The use of ________________ prohibited.

a. starting fluid is
b. all starting aids are
c. chemical starting aids are
<u>d. volatile or chemical starting aids is - 410</u>

35. Fueling of diesel-powered equipment shall not_________________________ escape-way unless the mine design and entry configuration make it necessary

a. be conducted in the primary intake
<u>b. be conducted in the intake- 411(a)</u>
c. be conducted in any designated intake
d. be conducted in either the primary or secondary intake

36. All used intake air filters, exhaust diesel particulate matter filters and engine oil filters ___________________ ____________ containers and removed from the underground mine to the surface.

a. shall be repacked, in their original containers or other substantial enclosed
<u>b. shall be replaced in their original containers or other suitable enclosed - 412 (b)</u>
c. shall be placed, in their original containers or other suitable
d. shall be sealed, in their original containers or other available enclosed

37. Diesel-powered equipment shall be______________________________________ as described in this article or removed from service.

a. maintained in an approved and healthful condition
<u>b. maintained in an approved and safe condition - 413 (a)</u>
c. maintained in a safe and healthful condition
d. maintained in a workmen like manner and good operating condition

38. Upon receiving a notice of revocation, the ____________ __________ to submit a plan to achieve and maintain compliance.

a. mine superintendent shall have thirty days
b. mine operator shall have sixty days
c. mine superintendent shall have sixty days
d. mine operator shall have thirty days - 413(a)

39. All service, maintenance and repairs of approved complete diesel-powered equipment packages shall be performed by mechanics who are ______________________________.

a. designated by the operator and qualified in accordance Section 422
b. trained and certified in accordance with the requirements of Chapter 4
c. trained and qualified in accordance with Section 422 - 413 (b) (1)
d. designated by the operator and certified in accordance with Chapter 4

40. Service and maintenance of approved complete diesel-powered equipment packages shall be performed according to:

a. the specified routine maintenance schedule
b .the on-board performance and maintenance diagnostics readings
c. the emissions test results; component manufacturer's recommendations
d. all the above - 413(b)(2)

41. Records -. The person performing the emissions test, examination, maintenance or repair shall certify by _______________________________ that the emissions test, examination, maintenance or repair was made.

a. date, time and initial
b. date, time and engine serial number
c. date, time, engine hour reading and signature - 414(b)
d. date, time, engine hour reading, equipment number and signature

42. Records of preoperational examinations shall be retained for:

a. at least one year and stored at a location outside of the mine
b. the two previous one hundred hour maintenance cycles
c. the two prior one hundred hour maintenance cycles
d. the pervious one hundred hour maintenance cycle - 414 (e)

43. Records of emissions tests, one hundred-hour maintenance tests and repairs shall be countersigned once each week by the:______________________________.

a. certified mine electrician and mine foreman
b. certified mine electrician or mine foreman - 414 (f)
c. mine superintendent and mine foreman
d. mine superintendent or mine foreman

44. Preoperational examination- Prior to use of a piece of diesel-powered equipment during a shift, ____________________________ as follows:

a. the trained equipment operator shall conduct an examination
b. any trained diesel powered equipment operator shall conduct an examination
c. an equipment operator shall conduct an examination - 415(a)
d. a qualified equipment operator shall conduct an examination

45. The diesel powered-equipment pre-operational examination requires ________.

a. requires 5 steps prior to engine start-up.
b. requires 10 steps prior to engine start-up.
c. requires 12 steps prior to engine start-up.
d. requires 15 steps prior to engine start-up. - 415 (a) (2)

46. Schedule of maintenance- At intervals _________________________ engine operation, a qualified mechanic shall perform the following maintenance and make all necessary adjustments or repairs or remove the equipment from service:

a. of one hundred hours of underground
b. not exceeding one hundred hours of - 416
c. within two hundred hours of
d. not exceeding one hundred hours of underground

47. When any diesel-powered machine first enters service at a mine, ________________ determined by a qualified mechanic.

a. carbon monoxide (CO) emission values shall be
b. baseline emission values shall be - 417(b)
c. all gaseous emission values shall be
b. primary emission values shall be

48. When performing an emissions test the qualified mechanic shall: Allow the engine to operate for a ________________, to insure proper carbon monoxide readings and record both carbon monoxide and carbon dioxide readings

a. period time, that is at least 90 seconds duration,
b. sufficient time, not less than 90 seconds duration, - 417(b)(13)
c. period of time, of not more than 90 seconds duration
d. sufficient length of time

49. Diagnostic testing -. The qualified mechanic shall do the following: Put the engine into a _____________________________.

a. torque converter stall condition
b. loaded condition - 418(a)(10)
c. full throttle condition
d. high idle condition

50. If the average CO reading for untreated exhaust gas is greater than __________________________ under section 417-A(b) or if the average CO reading for treated exhaust gas is greater than ___________, the equipment has failed and must be serviced and retested before it is returned to regular service;

a. fifty percent of the baseline established, 100 parts per million

b. one hundred percent of the baseline established, 50 parts per million

c. twice the baseline established, 100 parts per millions - 418(b)(13)

d. the baseline established, 50 parts per millions

51. The ambient concentration of exhaust gases in the mine atmosphere shall not exceed___________ for carbon monoxide and ____________ for nitrogen dioxide.

a. 35 parts per million, three parts per million - 419(a)

b. 35 parts per million, five parts per million

c. 50 parts per million, three parts per million

d. 50 parts per million, and, five parts per million

52. Ambient exhaust: Measurements shall be made ___________ or more often if _________ individual and shall be conducted under the requirements of this section.

a. daily or more often if necessary by a, certified

b. weekly or more often if necessary by a, qualified - 419 (a)

c. daily or more often if necessary by a, qualified

d. weekly or more often if necessary by a, certified

53. Measurement of exhaust gases shall be made with a_____________________ no less precise than detector tubes.

a. hand held gas detecting device

b. portable gas detecting device

c. sampling instrument - 419(b)

d. multi gas detecting device

54. If the concentration of any of the gases listed in subsection (a) ___________________ ___________________, changes to the use of the diesel equipment, the mine ventilation or other modifications to the mining process shall be made.

a. is 50 percent or more of its exposure limit

b. is 75percent or more of its exposure limit - 419(c)

c. is 100 percent or more of its exposure limit

d. above the Threshold Limit Value set in this subsection

55. Repair or adjustment of the fuel injection system shall only be performed by qualified mechanics ____________________________________.

a. trained by the engine manufacturer

b. authorized by MSHA

c. trained by the Department

d. authorized by the engine manufacturer - 419(e)(1)

56. Approval.--Training course instructors and training plans required by this section shall be approved ___________________________.

a. by the Secretary
b. by the Bureau
<u>c. by the department - 420(a)</u>
d. by the Coal Operator

57. Diesel operator- Refresher training.—Refresher training, separate from that required by MSHA regulations at 30 CFR Pt. 48 (relating to the training and retraining of miners), shall be __________________________________.

a. completed at least once every 12 months
<u>b. required annually - 420(d)</u>
c. required at the discretion of the department
d. completed on a yearly basis

58. Equipment-specific hands-on orientation training shall be given __________________________ where the equipment will be operated.

<u>a. in an area of the mine - 421(b) (1)</u>
b. in an underground area of the mine
c. safe and secure area
d. any area of the mine

59. Operation of diesel-powered equipment - Roadways where diesel-powered equipment is operated shall be maintained as free as practicable from ____________________, which affect control of the equipment.

a. bottom irregularities
b. wet and muddy conditions
c. debris
<u>d. all of the above - 423(d)</u>

60. Traffic rules.--Traffic rules, including speed, signals and warning signs, shall be _________________ at each mine and posted.

<u>a. standardized - 423(g)</u>
b. governed by the mine foreman
c. governed by the department
d. governed by the mine superintendent

61. Technical Advisory Committee. - The advisory committee shall consist of _______________ who shall be residents of this Commonwealth.

a. sufficient number members
b. three members
<u>c. two members - 424 (b)</u>
d. a contingent of members

62. Technical advisory committee Terms.—Each member of the technical advisory committee shall be appointed for a term of ________________

a. one year
b. two years
c. three years - 424(c)
d. four years

63. Technical advisory committee - __________________________, the technical advisory committee shall consider requests for the use of alternative diesel-related health and safety technologies with general underground mining industry application which are consistent with this chapter.

a. Upon application of a coal miner, coal mine operator
b. Upon application of a diesel-related technology manufacturer
c. On its own motion
d. all of the above - 424(i)(2)

64. Technical advisory committee –Within _____________________ of an application for use of alternative technologies or methods, the technical advisory committee shall complete its investigation and make a recommendation to the secretary.

a. 30 days of receipt
b. 60 days of receipt
c. 90 days of receipt

65. Required certifications or approvals.--Inby and outby diesel-powered equipment may be used in underground mines if the inby or outby diesel-powered equipment uses an engine approved or certified by MSHA, as applicable, for inby or outby use that, when tested at the maximum fuel-air ratio, does not require a MSHA Part 7 approval plate ventilation rate exceeding __________________________.

a. the ventilating rate set by the department
b. 50 c.f.m... per rated horsepower
c. 75 c.f.m... per rated horsepower - 401(c)
d. the guidelines set forth in this subsection.

66. Diesel-powered equipment package. Approval.--All diesel-powered equipment shall be approved by the department as __________________ which shall be subject to all of the requirements, standards and procedures set forth under this chapter.

a. a complete diesel powered equipment package - 402 (a)
b. a complete unit of diesel powered equipment
c. a complete piece of diesel powered equipment
d. a complete underground diesel powered equipment package

67. Exhaust emissions control -. Except as provided in paragraph (3), the exhaust emissions control and conditioning system shall be required to successfully complete a single series of laboratory tests for each diesel engine, conducted at a ____________________________.

a. laboratory certified by MSHA
b. laboratory certified by the department
c. laboratory approved by the department
d. laboratory accepted by the department - 403(a)(2)

68. Approvals.--Each specific model of diesel-powered equipment shall be approved by the department _________________________________

a. before it is put into service
b. before it is taken underground - 404(b)
c. before it performs any work at the mine
d. before it can be used.

69. Ventilation - Multiple units in operation.--Where multiple units are operated, the minimum quantity shall be at least the total of ______________ approval plate ventilation rate for each unit operating in that split.

a.100% of MSHA's Part 7 - 404 (d)
b.100% of the department's
c. Pennsylvania's
d. the most stringent

70. Ventilation - The department shall require that an approval plate be attached to each piece of the diesel-powered equipment. The approval plate shall specify _____________ for the specific piece of diesel-powered equipment.

a. the minimum ventilating air quantity - 404(b)
b. the department's approval plate air quantity
c. the maximum ventilating air quantity
d. the MSHA approval plate air quantity

71. Diesel fuel standards.--Additionally, the fuel shall also meet the ASTM D975 standards with _________________________ Fahrenheit or greater at standard temperature and pressure.

a. an auto ignition temperature of 650 degrees
b. a flash point of 100 degrees - 405(b)
c. a flash point of 125 degrees
d. an auto ignition temperature of at least 302 degrees

72. Fuel storage facilities - Diesel fuel shall not be allowed to enter pipelines or containers that have been welded, soldered, brazed or cut until the metal has cooled to_________________.

a. ambient temperature - 405(4)(ii)
b. less than 302 degrees Fahrenheit
c. less than its flash point
d. less than 100 degrees Fahrenheit

73. Fuel storage facilities - Welding or cutting other than that performed in accordance with paragraph (4) shall not be done within **______________** of a diesel fuel storage facility.

a. 50 feet - 405(d)(3)
b. 100 feet
c. 150 feet
d. 25 feet

74. Standards for safety cans.—Safety cans shall be clearly marked, **________________________**.

a. have a maximum capacity of five gallons
b. be constructed of metal
c. be equipped with a nozzle and self-closing valves.
d. all of the above - 405(o)

75. Fire suppression for storage areas. - Types of alarms.--Audible and visual alarms to warn of fire or system faults shall be provided **_________________________** always staffed when individuals are underground

a. in the storage facility and other areas of the mine that are
b. outside the storage facility and all areas of the mine that are
c. at the protected area and at a surface location that is - 409(d)
d. to a surface location that is

76. Trained individual on duty.--At least one individual **______________** and disposal of diesel fuel spills shall be on duty at the mine when diesel-powered equipment or mobile fuel transportation equipment is being used or when any fueling of diesel-powered equipment is being conducted.

a. certified in the cleanup
b. hazmat trained in the cleanup
c. specially trained in the cleanup - 411(c)
d. who has been qualified in the cleanup

77. Schedule of maintenance. If the equipment is approved with a non-disposable diesel particulate filter, a smoke dot test of the filtered exhaust must be performed at this time. The results of the smoke dot test shall be recorded on the 100-hour emissions form. If the interpreted smoke dot number **___________________**, the technical advisory committee shall be notified and shall investigate to determine if the filter is functioning properly.

a. is greater than two
b. is greater than three - 418(20)
c. is greater than four
d. is greater than five

78. Fuel storage facilities- Must be equipped with at least ________________multipurpose dry-chemical type fire extinguishers.

a. one portable 20 pound

b. one portable 10 pound

c. two portable 20 pound - 405 (e) 92) (iv)

d. two portable 10 pound

79. Pump transfers.--When diesel fuel is transferred by means of a pump and a hose equipped with a nozzle containing a self-closing valve, a powered pump may be used only if: the pump ____.

a. is equipped with an accessible emergency shutoff switch - 405(b)(2)

b. is provided with an on-off switch that is within easy reach

c. uses no more than 12 volts of electricity

d .is fully grounded to the electrical system

80. Diesel fuel piping systems - Horizontal pipeline prohibition.--Horizontal pipelines shall not be used ________________________.

a. to distribute fuel throughout a mine - 406 (i)

b. to distribute fuel to more than one fuel storage unit

c. to fuel diesel powered equipment

d. in trolley wire mines

81. Limitations where trolley wires are present.—In mines where trolley wire is used, diesel fuel transportation units shall be provided with insulating material to protect the units from any energized trolley wire, and the distance between the diesel fuel transportation unit and the trolley wire shall _______________, or the trolley wire shall be de-energized when diesel fuel transportation units are transported through the area.

a. be at least 24 inches

b. not be less than 12 inches - 407(l)

c. not be less than 15 inches

d. be more than 12 inches

82 Fire suppression devices shall be visually inspected __________________ by an individual qualified to make the inspection.

a. at 30 day intervals

b. at least bi-annually

c. at least once each week - 409 (h)

d. at 120 day intervals

83. Diesel fuel standards. - The operator shall maintain ______________________ to verify that the fuel used underground meets this standard.

a. a copy of the most recent delivery receipt from the supplier - 405(b)

b. a copy of a fuel analyses by a laboratory excepted by the department

c. a copy of a fuel analyses by a laboratory approved by the department

d. a copy of a fuel analyses by an independent laboratory

84. Underground diesel fuel storage facilities shall meet all of the following: Be marked with conspicuous signs designating________________________.

a. flammable liquid storage

b. combustible liquid storage - 405 (e) (2) (v)

c. diesel fuel storage

d. volatile liquid storage

85. Underground diesel fuel storage facilities shall: Be included in _____________________________.

a. the pre-shift examination - 405 (e) (2) (vi)

b. the on-shift examination

c. the required weekly examination

d. all required underground examinations