THE COLLIERY MANAGERS EXAMINATIONS NEW GUIDE

(Revised Edition)

THE

COLLIERY MANAGERS EXAMINATIONS NEW GUIDE

A Handbook for Students preparing for the Examinations of the Board of Mining and for Colliery Officials

by

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CERTIFICATED COLLIERY MANAGER. FIRST-CLASS HONOURS MEDALLIST IN MINING, BOARD OF EDUCATION. FIRST-CLASS HONOURS CERTIFICATE IN SURVEYING, CITY & GUILDS OF LONDON. FORMERLY COUNTY MINING LECTURER FOR LANARKSHIRE, AND CHIEF MINING LECTURER AT THE TECHNICAL COLLEGE, COATBRIDGE, AND AT HAMILTON TECHNICAL SCHOOL

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PREFACE TO REVISED EDITION

THE original *Colliery Manager's Guide*, written by me and published in 1915 by Messrs. Thomas Wall & Sons, of Wigan, was very well received by Mining Students and Colliery Officials alike in Great Britain and the Dominions, upwards of 10,000 copies being sold over a period of twenty years.

Subsequent changes in the syllabus of the Colliery Managers Examinations, covering modern mining practice and higher scientific knowledge, together with the much higher standard of work required by the Examinations Board for success in the examinations, led to the preparation of a new guide-book to replace that of 1915. This, the original edition of the *Colliery Manager's New Guide*, published by Messrs. Griffin & Co. in 1936, was received with an equal amount of enthusiasm and has been studied with profit by Mining Students in Great Britain and abroad; and after six years it has become necessary to publish a new and further revised edition.

The chief improvement made herein is that some of the older types of question have been replaced by those set at the examinations held in July, 1940, and July, 1941, thus bringing the subject-matter up to the standard of modern mining practice. As before, the specimen answers have been kept as short as possible to suit examination conditions, but they represent up-to-date knowledge of mining methods. It is hoped that the success of the work in its earlier forms will be maintained with the publication of this revised edition, and that the book will continue to be of service not only to Mining Students but also to Colliery Officials in dealing with everyday problems in connection with their duties in mines.

I have much pleasure in acknowledging valuable help from Mr. Donald Mackinnon, Mining Agent for Messrs. Bairds and Dalmellington, Ltd., in the Prestwick district of Ayrshire and a former colleague on the Mining Department Staff at the Technical College, Coatbridge, in connection with the checking over of the text and the valuable suggestions made while doing this.

HENRY C. HARRIS.

November, 1942.

CONTENTS

THE COLLIERY MANAGERS EXAMINATIONS NEW GUIDE

PART I.—WINNING AND WORKING

MAY 1931 EXAMINATION

Packing Waste Workings by a Scraper Loader

1. In a 5-foot seam, dipping 1 in 8, it is decided to pack solid on a longwall face by means of a scraper loader. Describe how you would do this, and how you would get the debris tight against the roof. Illustrate your answer by sketches. (50)

A. Fig. 1 shows the general arrangement of a longwall face for solid stowing of the waste by means of a scraper loader. The face advance is 4½ feet per day, and this width of stowing must be carried out after the coal is stripped off. Suitable dimensions are given on the diagram.

The debris for stowing the waste is conveyed to the top of the material road in tubs, and these are emptied by a suitable tippler at a convenient position for conveying down the face by a scraper conveyor.

The scraper loader of the *Beckett and Anderson* pattern is shown in side elevation. The scraper conveyor passes up the inclined part of the loader and discharges the debris through an opening in the horizontal or telescopic part. The conveyor is operated by a suitable hauler placed in the underframe of the loader. The latter is moved up the face by hand as the stowing of the waste progresses.

The debris is packed into the waste from the dip side towards the rise side. It is made tight at the roof by building-in the larger pieces, and by constructing a rough pack on the coal-face side.

Note.—Under Examination conditions, candidates are required to answer *six* questions from this Section. The figures in brackets indicate the maximum marks allotted to each question.

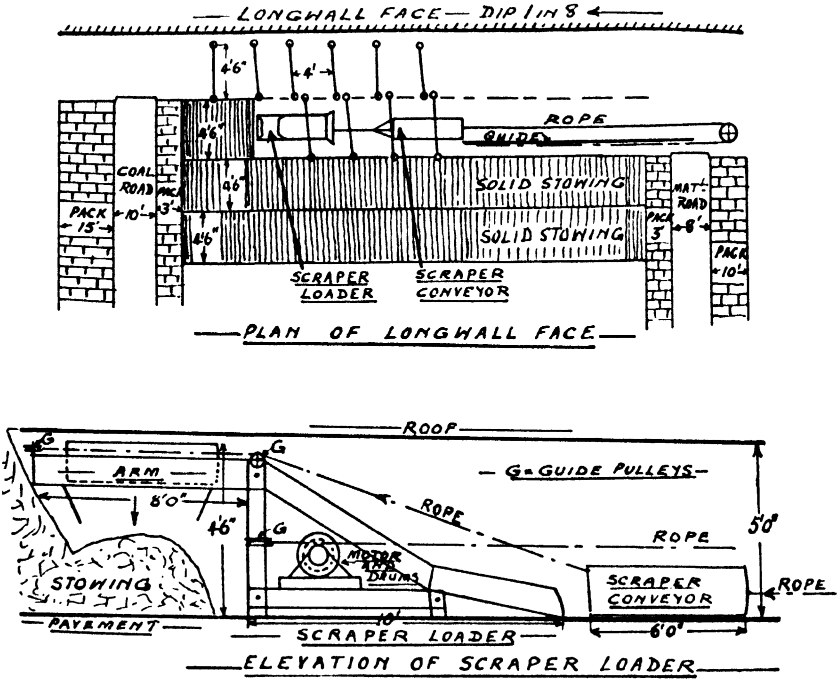

Fig. 1.—Details of Longwall Face with Scraper Loader for Stowing.

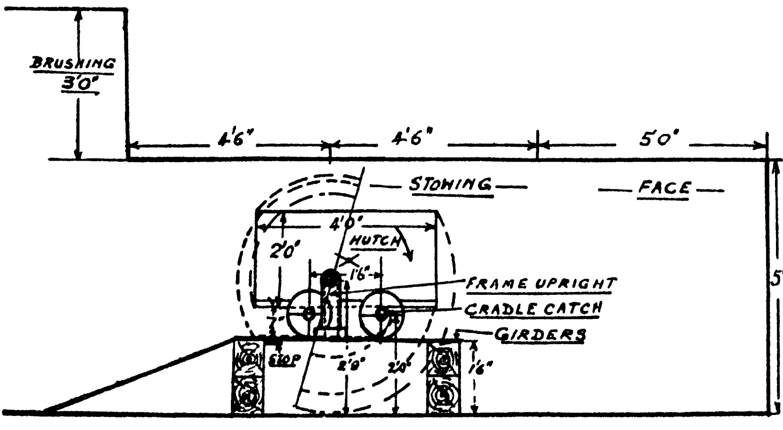

Fig. 2.—Debris Tippler.

Fig. 2 shows, in side elevation, the arrangement of a simple form of tippler of the cradle type, which might be installed at a suitable position for dealing with the stowing material. The platform, consisting of two girders about 6 feet long and 3 feet 6 inches apart, is raised about 1 foot 6 inches above the pavement level to allow of end tipping of the hutch. A suitable haulage might be installed near the face for hauling the hutch up to the tippler. The hutch is received by a cradle suspended by trunnions fixed to the platform beams.

When the front axle of the hutch is arrested by the cradle catch, the centre of gravity of the load is in advance of the trunnions upon which the cradle is suspended, and automatic tipping is obtained. After the debris is tipped on the pavement ready for the scraper conveyor to take it along the face to the loader, the cradle and empty hutch swing back automatically to the horizontal position.

Geological Formations

2. What are the principal geological formations above the Carboniferous? Name them in the order in which they were deposited. (40)

A. The following tabulation shows the order of the geological formations above the Carboniferous :—

Post-Tertiary	or	Pleistocene.
Tertiary	,,	Eocene, Miocene, Oligocene and Pliocene.
Cretaceous	,,	Chalk beds.
Liassic	,,	Oölites and Lias.
Jurassic	,,	New Red Sandstone beds.
Permian	,,	Magnesian limestone beds.
Carboniferous	,,	Coal-bearing beds.

Changing a Heavy Winding-Rope

3. Explain in detail how you would change a lock-coil winding-rope used in a shaft 600 yards deep. How often would you re-cap? State how much rope you would cut off each time, and by what means you would cut it. (40)

A. The cycle of operations in dealing with the above problem is as follows :—

(a) Bring the cage with the rope to be changed to the surface-level, and have it properly supported by strong beams across the lip of the shaft.

(*b*) Detach the old rope from the cage, cut off the hosing or capping, and by using a hand line, pass the free end of the rope over the pulley to the ground-level.

(*c*) Attach the free end of the rope to a suitable reel and start up the winding-engine slowly. The cage hanging in the shaft will be wound up, and the old rope will pass to the reel at the surface.

(*d*) Remove the old rope completely from the winding-drum by unfastening glands, etc.

(*e*) Bring the new rope to a suitable position near the winding-drum, and pass the free end through the drum and gland it in the usual manner, after allowing for some spare coils on the drum. The winding-engine is then started so as to coil the new rope on the drum, while the cage already in the shaft will go again to the pit bottom.

(*f*) By use of the hand line, pass the end of the new rope over the pulley, fit on the hosing, safety hook, links, etc., and again attach the rope to the cage resting on the beams at the shaft top.

(*g*) Finally the rope is adjusted to suit the cage levels.

(*h*) When the cage is lifted and the beams are withdrawn, winding can proceed in the usual way.

The winding rope should be re-capped at least every six months. Sufficient rope should be cut off to remove the part affected by internal corrosion, damaged wires, bulges, etc. A length of 6 to 10 feet is usually sufficient.

The rope might be cut by a hacksaw after being properly bound with wire at each side of the proposed cut. A mechanically operated cutting appliance is sometimes used and gives the best results.

Use of Steel Arches and Cambered Girders

4. Under what conditions would you use steel arches and cambered girders respectively? Give your reasons. What is your opinion of using steel arches near the face, and state the provision, if any, that you would make for the preliminary squeeze of the strata? (40)

A. Steel arches are greatly used on main roads and on roads near the working faces where the strata have become settled, and where it is desired to have great strength and durability of supports to protect the roof and sides. Sometimes they are applied to prevent heaving of the floor or pavement. The space between the arches might be packed with brickwork, concrete, or wood struts to give rigidity.

Cambered girders for roof support are used on similar roads under settled conditions, where the sides and pavement are of a hard nature and do not require support. The ends of such girders might rest on good sides, brick or concrete side walls, or strong supports of wood or steel. In the latter case, the cambered girders rest on bars running parallel with the road, while the vertical supports are fixed under the bars at each girder. Where props are used, displacement is prevented by securing them to old wire ropes by means of staples.

Where the following conditions have to be met, the use of steel supports is considered necessary :—

(*a*) Where adequate protection is desired for roads of large size for haulage and ventilation ; the systematic setting of the supports gives better control of roof and sides, so that falls are avoided.

(*b*) When stronger supports are necessary to give adequate protection to cables, signal wires, lighting installations and compressed-air mains.

(*c*) To obtain cleaner roads, increased output of coal, and a reduction of accidents.

(*d*) To have ultimately a reduced cost for upkeep of roads owing to less falls, while at the same time using supports which can be easily withdrawn and reconditioned.

Steel arches on roads near the working face are a decided advantage over wood supports. They have a much greater strength to resist roof pressure, and the roads are maintained at their proper height for much longer periods. The roof strata are often prevented from breaking after bending, and thus greater safety is obtained. The increased facilities for the haulage of the coal, and for passing an adequate current of air, are definite advantages.

To allow of the preliminary squeeze of the roof strata, the arches might be fixed with the uprights resting on old railway sleepers, into which they sink gradually and become settled. In numerous cases, wood stilts are fixed to the uprights by means of clamps, so that the arches subside gradually with the roof until stable conditions are attained.

Shaft-sinking Operations

5. A shaft of 21 feet diameter inside the lining is being sunk through dry strata. Give a description of the work as it proceeds, paying particular attention to the shot-firing. (40)

A. The procedure in sinking a shaft under dry conditions might be as follows :—

(*a*) The shaft should be made of sufficient size to allow of the permanent lining being put in, say 26 feet diameter, by blasting out the strata in a systematic manner. The shot-holes might be drilled in a proper direction, to a depth of 6 feet, by compressed-air drills fixed to a drilling frame in the shaft bottom. Three sets of holes could be arranged as shown in the plan view, Fig. 3, such as :—

(i) 6 inner sumping holes, 6 feet deep, 2 inches diameter, in a circle 4 feet in radius.

(ii) 16 outer sumping holes, 6 feet deep, 2 inches diameter, in a circle 9 feet in radius.

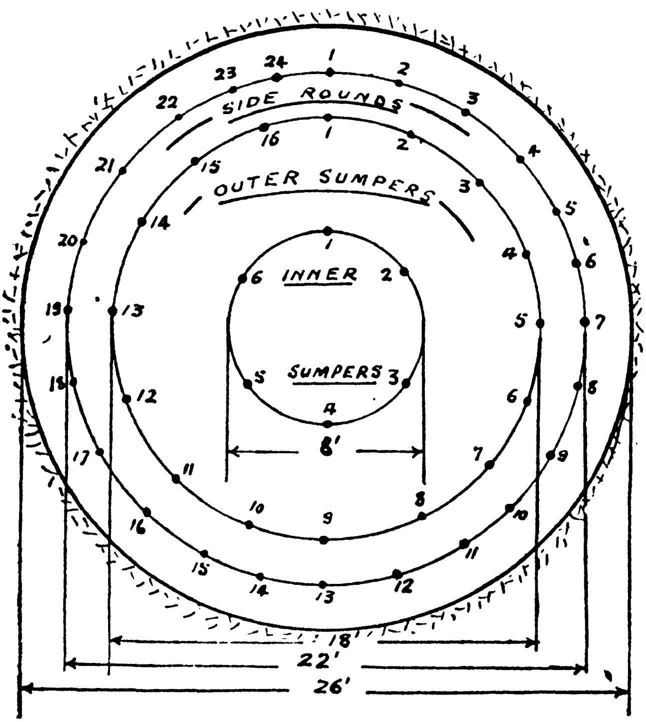

Fig. 3.—Plan View of Shot-holes for a Sinking Pit.

(iii) 24 side rounds, 6 feet deep, $1\frac{1}{4}$ inches diameter, in a circle 11 feet in radius.

Simultaneous firing of these holes could be accomplished by a battery at the surface, the holes being connected up in series fashion.

(*b*) After the drilling and firing shift is finished, the filling shift would proceed with the removal of the debris by filling it into suitable sinking kettles for delivery at the surface. During this shift or shifts, the temporary lining of the shaft sides would be carried out, using deals 1 inch thick, steel rings 4″ × 1″ at intervals of 3½ feet, and suitable connecting hooks between the various rings as shown in Fig. 4. The temporary lining could be secured by chains to the walling crib above, or from surface beams.

(*c*) After sinking has reached a depth of 10 to 15 yards, and a good hard bed is at hand, arrangements must be made for putting permanent lining of brickwork, or concrete, into the shaft. This lining is built up from a walling crib of cast-iron, and the temporary lining is removed for future use.

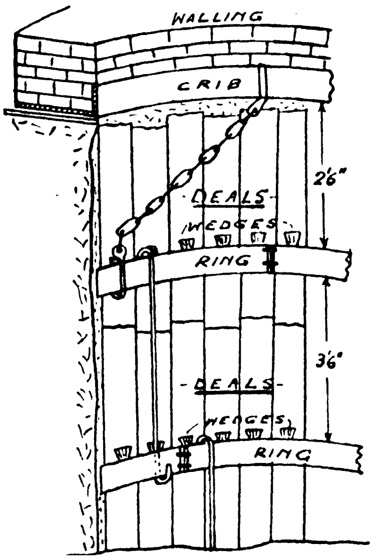

Fig. 4.—Elevation of Temporary Lining for a Sinking Pit.

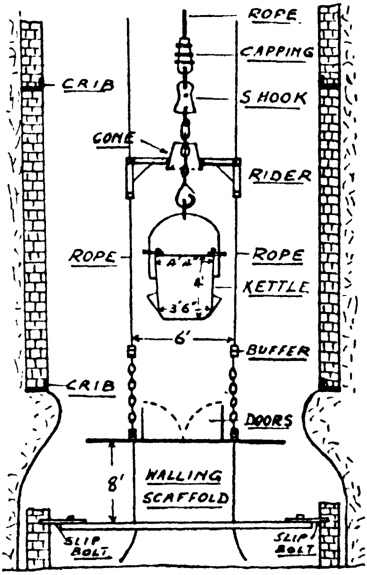

Fig. 5.—Section of Shaft Walling.

(*d*) Sinking and lining might be carried on safely at the same time by installing a walling scaffold of the double type, supported on security bolts resting on the walling or lining, and moved when necessary by a worm-geared winch at the surface. The winch ropes could be used as guide ropes for the kettle rider, as shown in Fig. 5.

By this method, or cycle of operations, the shaft sinking could be carried out expeditiously and safely, great care being taken in connecting up the shots to ensure that they are all properly exploded at the one time. The master sinker would be responsible for carrying out the instructions of the management and the provisions of the Mines Act, and he would have the shaft sides properly examined and secured after firing a round of shots.

Cementation Process apart from Sinking

6. To what uses in mining, apart from sinking, can the cementation process be successfully applied ? Describe fully the method of application in one of the cases you quote. (40)

A. The cementation process might be successfully applied to repair breaks and cracks in brickwork and concrete constructions, such as shaft wallings, stoppings for fires, plant foundations, retaining walls and roadway strata and linings in mines. It might also be applied for stopping leakages in cast-iron tubbing or similar metal linings in shafts. It has likewise been found to be very effective in dealing with broken strata near dams, and for stopping leakage of water through brickwork and concrete dams.

Fig. 6 shows section and front elevation of a brickwork dam which has been constructed in soft strata, and leakage of water to

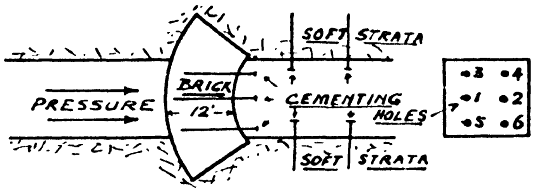

FIG. 6.—Section and Elevation of Brick Dam.

a small degree is taking place through the dam and adjacent strata. The dam is otherwise sound, and a retaining wall before cementing is considered unnecessary. The strata in front of the dam might be cemented by driving in holes 1½ to 2 inches diameter to a depth of 6 to 9 feet, and cementing at a pressure exceeding the pressure of water behind the dam. Similarly, six cementing holes of the same size might be driven into the dam for a distance of 9 feet. These holes could be cemented in pairs (as indicated by the figures 1 to 6) at a pressure greater than the pressure of water.

NOVEMBER 1931 EXAMINATION

Longwall Retreating System

1. Under what conditions would you consider the "longwall retreating" system of working the most suitable ? Sketch a lay-

out on this system sufficient to yield about 500 tons per shift from a 5-foot seam dipping 1 in 8, and give principal dimensions. (50)

A. The conditions under which the longwall retreating system might be applied are as follows:—

Where the seam of coal to be worked is fairly thick and there is not sufficient material available for building proper roads through the waste or gob; also when the seam to be worked is liable to spontaneous combustion, and the retreating face can move from the dip towards the rise of the seam. This system is often applied to small areas of coal where some difficulty exists in working, such as areas of bad roof, faults appearing in step fashion, gas given off freely from the coal and strata, and water issuing freely from the overlying strata. Gas is tapped during the advancing stage, or the opening out of panels.

This method of working is now becoming more general and is gradually taking the place of "pillar-and-stall" working in thick seams of coal. By its use coal-cutters and coal-conveyors can be installed, as with longwall advancing. Safer working conditions attend its use, and cheaper working of the coal results.

Fig. 7 shows in plan view the layout of workings on the longwall retreating system. Main levels in groups of four, for haulage and ventilation, are driven out from the shafts to the boundary. Rise headings in groups of three, for the same purpose, are driven to form panels 1,000 yards wide, the headings being connected at intervals of 220 yards for working with the double-unit system of conveyors. These headings are connected by levels in pairs at intervals of 540 yards.

The fast places might be driven 9 to 12 feet wide by arc-wall coal-cutters for cutting and shearing, and during the process of driving they might be ventilated by auxiliary fans and steel pipes 10 inches diameter by ⅛ inch thick.

The double-unit retreating faces, marked *A* on the plan, are 220 yards long, and are undercut by chain machines to a depth of 4 feet 6 inches. Shaker conveyors are arranged on the rise side of the face and belt conveyors on the dip side. These conveyors deliver into troughed-belt conveyors on the level roads marked *B* on the plan, the maximum distance of such roads being 450 yards. The troughed-belt conveyors deliver coal direct into tubs on the centrally-placed rise headings at the points marked *C* on the plan.

Each double-unit face produces an output of 500 tons per shift, and with four such units in operation an output of 2,000 tons per day can be obtained.

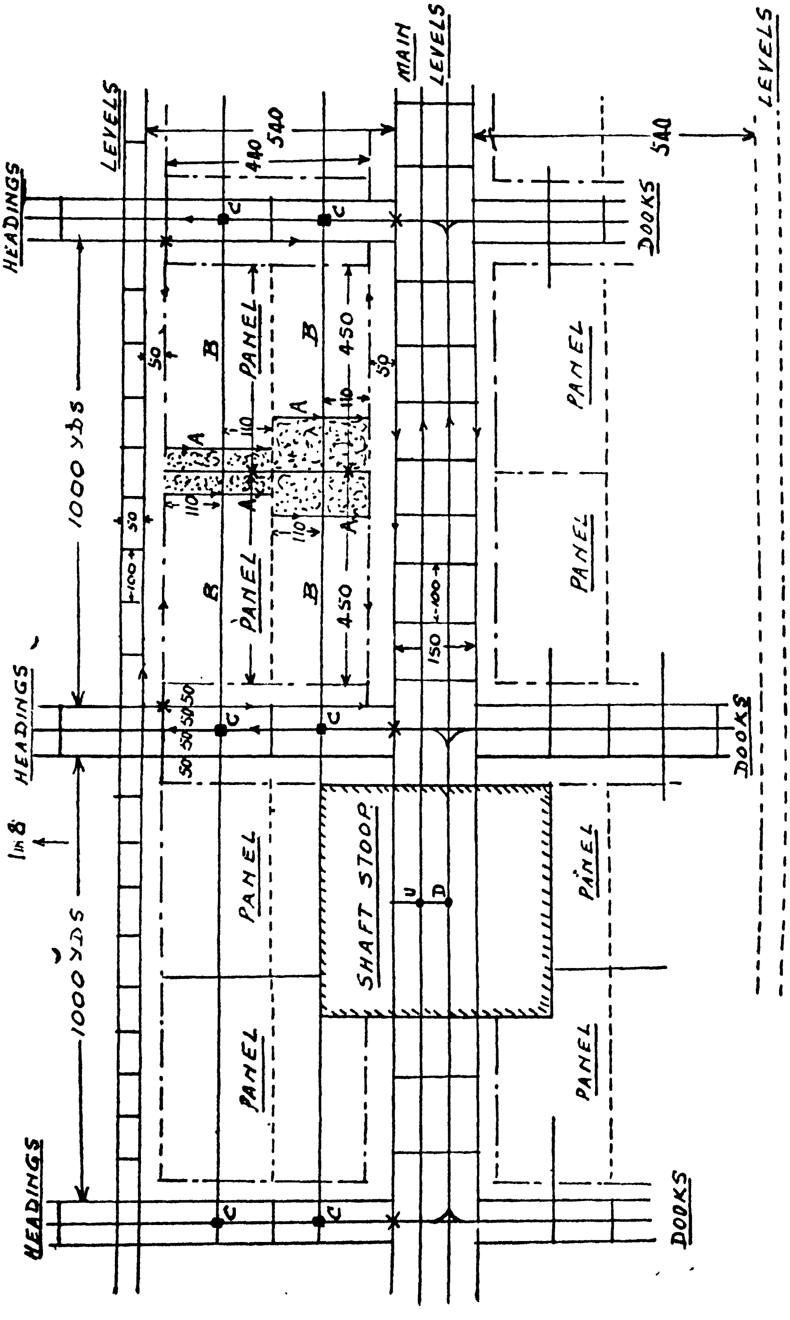

FIG. 7.—Plan showing Design of Longwall Retreating Workings.

Main-and-Tail Rope Haulage Curve

2. Make a sketch of a curve on a main-and-tail haulage road, where the road changes direction some 30 degrees. The tubs are 3 feet wide, and the gauge is 2 feet. Show the track with guard or check-rail and the rollers for the rope, and give main dimensions. (40)

A. Fig. 8 shows by plan and section the possible arrangement of a main-and-tail rope haulage curve. The main rope is guided

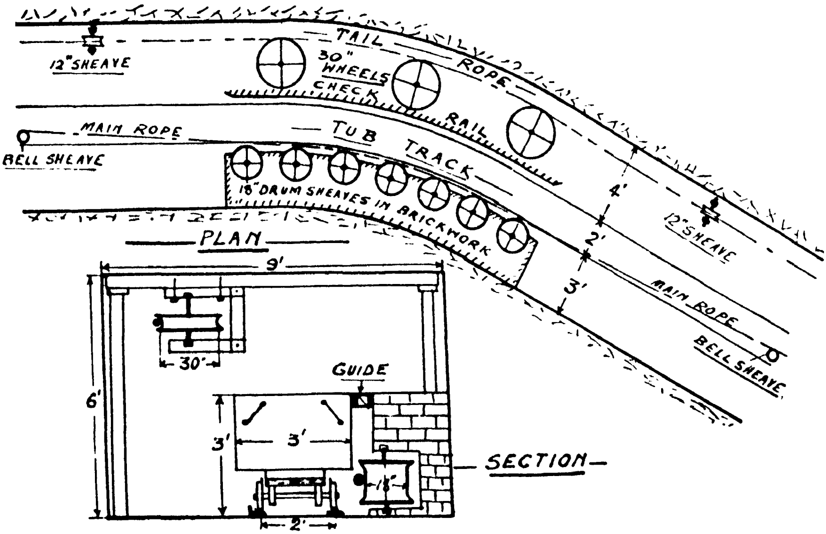

FIG. 8.—Plan and Section of Main-and-Tail Rope Haulage Curve.

round the curve at the proper level by 18-inch diameter drum-sheaves, placed 3 to 6 feet apart, secured in brickwork, and by bell sheaves at either end of the curve to give the rope proper direction. The tail rope is guided round the curve near the roof by suitable hanging sheaves 12 inches diameter, and by rope wheels 30 inches diameter. A check-rail is put round the curve on the outer side of the outer rail to prevent derailment of tubs, while a rubbing angle arranged at the inner side of the curve greatly facilitates the movement of the tubs round the curve, and should be continued until the main rope reaches the bell sheaves. Suitable dimensions are given on the sketch.

Subsidence caused by Working Coal Seams

3. In working a seam of coal on the longwall system, what observations would you make if instructed to report on subsidence of the overlying strata ? How would the subsidence be affected (*a*) in the case of the coal measures coming right to the surface ; (*b*) where the measures are covered by sand or clay ? (40)

A. Fig. 9 shows a plan of longwall workings, together with a surface area in advance of the workings so that observations of subsidence at the surface might be taken.

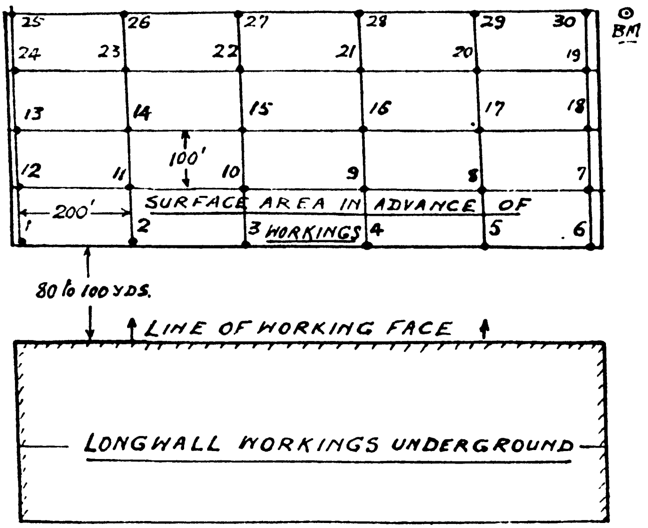

FIG. 9.—Plan illustrating Measurement of Subsidence.

At a distance of 80 to 100 yards in advance of the workings underground, a system of peg lines should be laid out, parallel with, and at right angles to, the faces. Pegs 4 inches square and 18 inches long should be driven into the surface at intervals of about 100 feet, and should be properly numbered. After this has been accomplished the level of each peg, with reference to an ordnance bench-mark in unaffected strata, must be determined. A good dumpy level and staff would be required for this work. Subsequent levels should be taken at periods of three to six months, and properly recorded in tabulated fashion in a suitable book. Sections could then be plotted along any line of pegs showing the amount of subsidence during the fixed periods.

When a seam of coal is worked underground, the overlying strata gradually crush down into the waste or gob, and a dome of subsidence is made in the overlying strata. The peak of this dome, or a part of it, might reach the surface and cause subsidence there. With solid beds extending to the surface, the area of surface subsidence is generally less than the area of coal worked, while if the dome penetrates into soft beds at the surface, the area of surface subsidence will be greater than with hard beds, but will be minimised vertically. With stiff clay beds at the surface, a certain amount of cushioning would result, and the surface subsidence would be less than with softer beds of sand and mud.

Working a Seam Inclined at 60 Degrees to the Horizontal

4. How would you open out and develop a 4-foot 6-inch seam of coal, which dips at 60 degrees to the horizontal? Show, by sketches, the timbering of a face and of a level haulage road. (40)

A. Fig. 10 shows section and elevation of the method of working a seam inclined at 60 degrees to the horizontal. Main levels are driven in the seam 14 feet high and 4 feet 6 inches wide at intervals of 300 feet. The working faces advance to the full rise of the seam

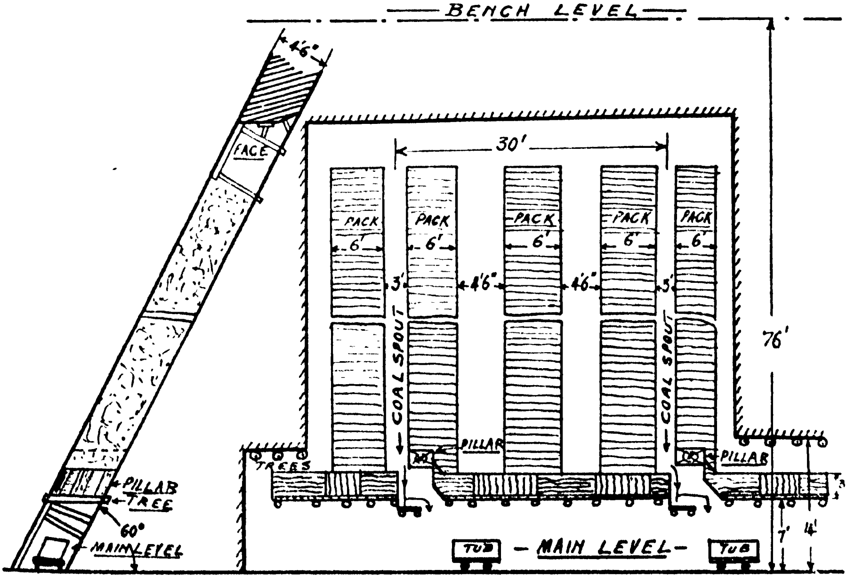

FIG. 10.—Section and Elevation of Steeply-inclined Workings.

from these levels, and suitable coal spouts are arranged every 30 feet along the face. These coal spouts do not extend to the full distance of 300 feet, but are cut off every 76 feet by bench levels. The latter feed into main inclines placed 600 feet apart, which in turn connect up the various main levels.

The levels are secured by trees let into the stone sides to secure the overhead coal. Trees and pillars are afterwards put in to support the packs and waste as the coal face advances. In addition to the above, the overhanging wall is supported by bars and trees, as shown in the sketch.

The overhanging wall at the face is supported by bars and trees, as in the level, while the coal is secured by suitable sprags along the face, as shown in the sketch.

Pit Headgear and Safety Appliances

5. Make a rough line sketch of a headgear, showing the compulsory safety appliances, and either sketch or describe any extra safety devices you would introduce. (40)

A. Fig. 11 shows in side elevation the outline sketch required in answer to the above question. Suitable dimensions are given

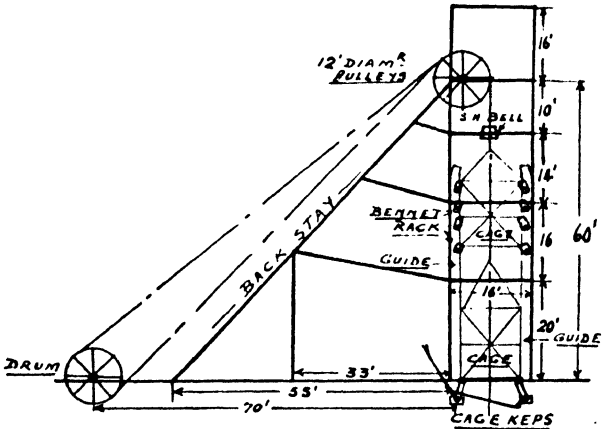

FIG. 11.—Side Elevation of Pit Headgear and Fittings.

on the same. The compulsory safety appliances required are: (*a*) the bell, or plate, for suspending the cage by means of the detaching hook in the event of an overwind, (*b*) the keps at the banking level for decking the cages at that point, and for holding the cage while men are entering or leaving the same, (*c*) rigid guides in the headgear for guiding the cage above the banking level after it has passed from the rigid guides in the shaft.

An extra safety device might consist of a rack arrangement fixed to the cage guides in the headgear. By this arrangement a series of studs, or boots, is forced back as the cage passes up the headgear in the event of an overwind, and these fall forward again and prevent the cage falling in the event of the safety hook not operating properly, or the cage chains breaking.

Firing a Shot at the Coal Face

6. Describe in detail, and in due order, the process of preparing for, and firing, a shot at the coal face. What dangers have to be guarded against? How many shots in a shift would you consider the maximum for one man to fire on a longwall face in a level seam 4 feet 6 inches thick? (40)

A. The following might be the order of procedure in firing a shot at the coal face :—

(*a*) The shot-firer must see that the shot-hole is properly placed and drilled according to the amount of coal to be blasted down, and that its direction is marked on the roof.

(*b*) He must also see that there are no breaks in the hole, and especially so at the back thereof, also that the hole is not bored beyond the holing.

(*c*) The charging and stemming of the shot must be carried out by the shot-firer using a permitted explosive, if the mine requires such.

(*d*) The place where the shot is to be fired, and all accessible places within a radius of 5 yards, must be treated with stone-dust or water, on the roof, floor and sides, unless special permission is obtained from the management not to carry out this operation. The shot-firer must also see that the strata are properly supported within this radius.

(*e*) Before firing the shot, the shot-firer must finally examine for gas the place where the shot is to be fired and all accessible places within a radius of 20 yards from that place. The shot must not be fired if gas is found in this examination, or if there is a cavity or break which is inaccessible and cannot be examined for gas.

(*f*) After these examinations and preparations are completed in a satisfactory way the shot-firer can proceed to fire the shot, after withdrawing the sprags and being assured that all workmen are in a place of safety. The face must be adequately fenced off to prevent anyone approaching the position of the shot. The exploder must be capable of firing only one shot at a time, and the shot-firer must see that this is adhered to.

(*g*) The actual firing of the shot is accomplished by fixing the detonator leads to the cable, which is then uncoiled from the face outwards to the battery and connected to it. The key of the battery must be in the possession of the shot-firer during these operations.

(*h*) After firing the shot the shot-firer must examine within the proper radius for gas and for bad stone before allowing the men back to their work.

In order to carry out properly the examinations and preparations referred to above, it is impossible to expect a shot-firer to deal with more than 6 shots per hour. If the work is proceeding in a satisfactory way, 35 to 40 shots might be fired during a shift, and this should be the maximum number.

Advantages and Disadvantages of Steel Props

7. Discuss the advantages and disadvantages of the various well-known types of steel props. (40)

A. The details required by the above question are shown in tabulated fashion at head of opposite page.

MAY 1932 EXAMINATION

Difficulties associated with Working Deep Seams

1. What are the particular difficulties associated with the working of deep coal seams ? What methods could you suggest to minimise these difficulties ? (50)

A. The usual method adopted in working deep seams of coal is to have large circular shafts capable of dealing with a large output from a large area of coal workings. This method calls for an organisation and design quite different from that applied in working shallow seams of coal.

The particular difficulties associated with the working of deep coal seams, and the methods of minimising these, are as follows :—

(*a*) The main roads leading to the various parts of the mine require to be of large size for ventilation and haulage ; their supports must be strong, while at the same time the cost for repairs

Type.	Advantages.	Disadvantages.
Rigid : Mild-steel tube 4″ diam. × ½″ thick. Soft wood core. Hard wood ends projecting from tube.	Yield strength 12 tons (high). Ultimate strength 65 tons (high). Great strength for strong roof. Straightened cold.	Difficult to set in varying heights. Difficult to withdraw after weighting. Must have regular system of setting and withdrawing.
Rigid : H-section steel joist. 3″ × 3″ to 6″ × 5″.	Great strength of 65 tons. Easily straightened in mine.	Same as above. Buckling is common. Very heavy, especially in thick seams.
Yielding : *Butterley.* Tube and cast-iron plug. Sliding sleeve and plug.	Yield load 20 tons (high). Ultimate load 30 tons. Long life. Easily set up.	Plug renewals costly. Buckling is common. Low margin between yield and buckling. Difficult to withdraw.
Yielding : *Tait.* Rolled steel joist stem 3″ × 3″. Top tapered into box with brackets and wedges. Steel cap at base.	Easily set up and withdrawn. Painted white. Yield load 20 tons (high). Ultimate loads 52 and 89 tons. Long life. High margin between yield and buckling.	Wedges are expensive. Straightening difficult.
Yielding : *New Sarre.* Upper and lower rolled steel joists 3″ × 3″ fitting into each other. Connecting links, steel wedge and cam.	Yield load 25 tons (high). Ultimate load 29 tons. Easily withdrawn.	Danger of springing out with hard pavement. Awkward to handle in a pit. Sharp edges and projections objectionable.

should be low. This might be accomplished by the introduction of steel linings, put up in a systematic manner. The smooth surfaces of such roadways, compared with irregular timbering, present less friction to the passage of the air-current.

(*b*) More firedamp is given off in working deep seams of coal than with shallow seams, while blowers of firedamp are not uncommon when approaching faults. Strict regulations are usually enforced for dealing safely with this gas. Good safety-lamps are required in working such seams ; and, again, the regulations in connection with these must be strict, to prevent explosions of firedamp.

(*c*) Owing to the high temperature of the strata in deep mines, and the high drying power of the air-current, these mines are termed *dry,* and deposits of dry coal-dust are present. Owing to the danger of coal-dust explosions, extreme care is necessary in dealing with firedamp and when firing shots. Good ventilation, and a regular and systematic treatment and cleaning of all roads, are absolute necessities. Spontaneous combustion is more likely to occur in deep mines, and the necessary precautions in working, and in preparation for the same, must be properly considered. The use of electricity at the face and explosives for blasting-down coal may be prohibited.

(*d*) The working faces in deep mines are subjected to a greater roof pressure, and steel supports are commonly used to prevent falls of roof and accidents. Strict attention must be given to the building of good packs for the support of the face and the roadways leading to it.

(*e*) The wet-bulb temperature of the air at the faces is much higher than in shallow seams, and to prevent early fatigue of the workmen during their working shift, a good current of air must be circulated at all times.

(*f*) Haulage systems in deep mines must be extensive and of ample power to deal with large outputs at a quick rate. The mechanical transport of the workmen long distances underground must receive proper consideration, and if instituted, generally results in great advantages to owners and workmen alike.

(*g*) Owing to the large percentage of moisture in the return air-current, special methods of support of roads, without the use of timber, is a necessary consideration.

(*h*) The colliery plant required for winding and ventilation in working deep seams must be of large size to give quick winding and good ventilating. The winding appliances and the shaft fittings, for quick winding in a deep shaft, must be constructed of the best and most suitable material. Reliable pumping plants of the turbo-type must be installed for pumping water up the shafts.

(*i*) Finally, a systematic organisation of officials must be put into operation by the management, sufficient to cover every working unit in all shifts during a 24-hour day. The officials must see that all rules and regulations are strictly adhered to. Safe working conditions and the prevention of accidents must always be of premier importance.

Visible Constituents of Bituminous Coal

2. Ordinary bituminous coal can be divided into four distinct

visible constituents. Name these, and give a short account of each. (40)

A. The four visible constituents of bituminous coal are as follows :—

(*a*) *Vitrain.*—This constituent of bituminous coal occurs in brilliant black bands having a glassy appearance, the bands being $\frac{1}{10}$ inch to 2 inches thick. It is brittle and soft in nature, and has a curved or conchoidal fracture. The formation of Vitrain is possibly due to larger pieces of vegetable matter being present, such as the trunks of trees. Under the microscope, Vitrain shows no traces of resistant plant remains, while only very faint outlines of woody cells are seen ; it appears like a red jelly with no structure. Vitrain forms a good coking coal which is very easily oxidised, but has only a fair oil content owing to the absence of spore coats and cuticles. The " ulmins," or decayed black material, form a large percentage.

(*b*) *Clarain.*—Ordinary bright coal, often finely banded in itself, is termed Clarain. It is generally soft and brittle and breaks into irregular pieces. Clarain has been formed from smaller pieces of vegetable matter in a kind of matrix. It has a larger proportion of spore coats and cuticles than has Vitrain. Examination under the microscope points to its being a mixture with layers of material like Vitrain, amongst which are strewn spore exines, pieces of plant cuticle, and other remains. It is formed from small pieces of vitranised wood, too small to be recognised as Vitrain, and mixed with the general refuse of the plant. Clarain has good coking properties, and a fairly good oil content owing to the presence of spore coats and cuticles. It contains less ulmins than Vitrain and is not so readily oxidised.

(*c*) *Durain.*—The dull variety of bituminous coal is termed Durain. It has a greyish, rough appearance, is hard and tough, and breaks with a rough dull fracture. Durain has been formed from very small pieces of vegetable matter in a kind of matrix. Under the microscope, spore exines and similar remains are seen in very great numbers, together with some clear woody fragments. Very many broken wood cells are evident. The ground mass contains opaque grains of fragments of wood cells, probably Fusain-like in nature. Durain contains clayey sediment, which the previous constituents do not contain, and which has been deposited at the same time as the plant particles. The properties of Durain are : non-coking ; not readily oxidised owing to low percentage of ulmins ; heavy owing to ash content ; and larger oil content owing to the presence of spore coats and cuticles. It inflames readily.

(*d*) *Fusain.*—This is the name given to " clod," or material having a charcoal-like appearance. It is a soft, fibrous material which powders easily and soils the hands. Fusain is found generally in small patches on the upper and lower surfaces of a lump of coal. It has been formed by heat and decay of the original coal material. Under the microscope Fusain is not transparent, but the original cell structure is evident as a fibrous structure. It has been made from woody material which has suffered a different form of decay from the remainder of the coal. The properties of Fusain are: non-coking, not readily oxidised, very slight oil content, and easily heated by crush, owing to its powdery form.

Controlling a Weak Roof to the Best Advantage

3. In a district of a seam having a weak roof, state by what various means you would endeavour to control it at the coal face to the best advantage. (40)

A. The following are important considerations when controlling a weak roof to the best advantage:—

(*a*) The coal should be worked on the longwall method with a straight line of face, and arrangements should be made for a regular and systematic advance of the face. The face should also be arranged to be not quite parallel with the lines of cleavage of the roof strata. Quick advance of the face is an important consideration.

(*b*) Steel props and steel crowns should be used for face supports, and these should be arranged in systematic fashion 3 feet 6 inches apart along the face, as shown in Fig. 12.

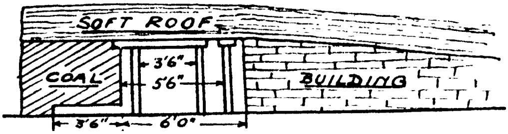

FIG. 12.—Section of Longwall Face.

(*c*) The roadside buildings and waste buildings should be kept well forward towards the face at all times and should be properly constructed.

(*d*) Supports should be withdrawn regularly from the face and advanced in systematic order, to allow of a distance of about 6 feet between the face and the waste at the beginning of the coal-stripping shift. With this arrangement of supports, the roof should bend down naturally upon the buildings without fracturing to any great extent, as shown in Fig. 12. This action will be brought about to a large extent by ripping the soft roof at sufficient points along the face to have only 12 to 15 feet of space between buildings.

(*e*) The main roads leading to the face might be supported by circular steel girders placed on stilts and set at intervals of about 3 feet.

Dealing with the Debris from a Sinking Pit

4. Sketch and describe the arrangement you would install at the top of a sinking pit for expeditiously disposing of the debris. (40)

A. The arrangement and plant necessary for dealing expeditiously with the debris from a sinking pit should include a good

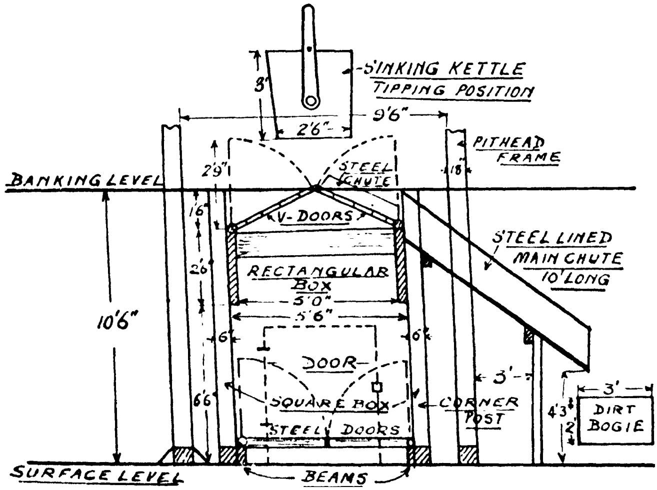

FIG. 13.—Sectional Arrangement of Surface Structure for Dealing with Sinking Debris.

temporary headgear and a good steam or electric winding-engine. The top of the sinking pit might be arranged as shown in the sectional elevation, Fig. 13.

The banking level is situated at a height of 10 feet 6 inches above the ground level, and a square box structure is built up to connect the two levels. The corner posts of the structure are supported on sole beams, and wood planks are secured to the former to complete the box and fence off the shaft. Hinged steel doors $\frac{3}{8}$-inch thick are placed on the sole beams at the ground level to facilitate loading of the kettle with materials for the sinking pit.

At a height of 5 feet 6 inches from the steel doors a rectangular wooden box is formed and fitted at the top with strong V-doors, resting on side cheeks and meeting at an inclination of 1 in 2. On the door at the tipping side a short steel chute is bolted to guide the contents of the kettle into a main chute and thence to the dirt bogie. The pitheadman's platform is made level with the top of the box structure.

The above arrangements greatly facilitate the work of dealing with the sinking debris, as the kettle is tipped and emptied without being taken from the winding-rope. The V-doors are opened by a lever attached to connecting-rods and cranks, the operation being assisted by means of balance weights.

Conditions suitable for Coal-cutting Machines

5. Under what conditions would you install (*a*) longwall coal-cutters ; (*b*) heading machines ; (*c*) pneumatic picks ? Which type of longwall cutter do you prefer for general use, and why ? (40)

A. *Longwall coal-cutters* are usually installed to undercut the coal where the face has been arranged in a straight line. A fairly good roof and an even pavement are an advantage with this type of coal-cutter, while the absence of faults on the line of face greatly facilitates the work. Thin seams of coal can be successfully undercut by machines and worked profitably, whereas by hand-work this is seldom possible. Sometimes dirt bands exist in a coal seam, and these might be removed completely by a coal-cutting machine at a lower cost than by hand-holing, and giving cleaner coal. In some instances coal is very hard and difficult to obtain by hand-work, while similar work by coal-cutting machines is less difficult and more profitable.

Heading machines are usually applied for driving narrow places that are required in opening-out for longwall workings. They might likewise be applied in thick seams of coal worked by the pillar-and-stall method, especially where the coal is of a hard nature. Where dirt bands exist in a seam of coal at some intermediate point between the pavement and roof in narrow work, heading machines might be used to advantage in removing the same. Sometimes shearing work is necessary in addition to undercutting, and this can be accomplished by some types of heading machines. They might also be used where the conditions of roof, pavement and cutting material are not suitable for other types of machines, and where faults and other irregularities are met with in working.

Pneumatic picks might be installed to work in headings of pillar-and-stall workings where the coal is of moderate hardness, thus dispensing with hand picks. They might also be applied to bring down moderately hard coal that has previously been undercut by longwall machines, thus dispensing with explosives and preventing roof breakages. Where difficulties exist owing to the presence of firedamp at the face, pneumatic picks might replace hand picks, in preference to coal-cutting and blasting.

The chain machine coal-cutter is considered the best for general use, as it can be successfully applied for undercutting both in narrow work and on longwall faces. This machine is comparatively light and is more easily operated than other types, in cutting, squaring corners, and cutting-in when starting to work.

It is likewise more suitable for undercutting in inclined seams. Less area of roof is exposed and unsupported during the cutting process by chain machines, and thus safer roof conditions exist. These machines can be applied successfully to cut either soft or hard coal, while at the same time the cuttings are automatically removed from the cut by the chain. Spragging of the coal can be carried out at a short distance from the face of the cut, and this is a great advantage.

The coal holings are " rough " and of market value.

NOVEMBER 1932 EXAMINATION

Layout of Pillar-and-Stall Workings

1. It is expected to wind 800 tons per shift from a seam of coal 5 feet thick, dipping 1 in 5. The shafts are situated in the centre of the royalty area. Sketch a layout to supply this quantity, the method of working to be pillar-and-stall. (50)

A. Fig. 14 shows a layout plan of pillar-and-stall workings as required by the above question. The sections are arranged to work to a rise of 1 in 7 in panels 660 yards wide, with pillars 55 yards square to centres of stalls. The stalls might be driven 12 feet wide by the use of arc-wall coal-cutting machines and pneumatic picks. Section A is fully developed in narrow work, and pillars are being extracted on the following-up system with shaker-conveyor faces 110 yards long. Section B is developing on the lines of Section A, the fast places being fully developed. Sections C and D are just starting to open out, and they will be developed from dip to rise by dooks and rise headings 660 yards apart, as shown on the plan.

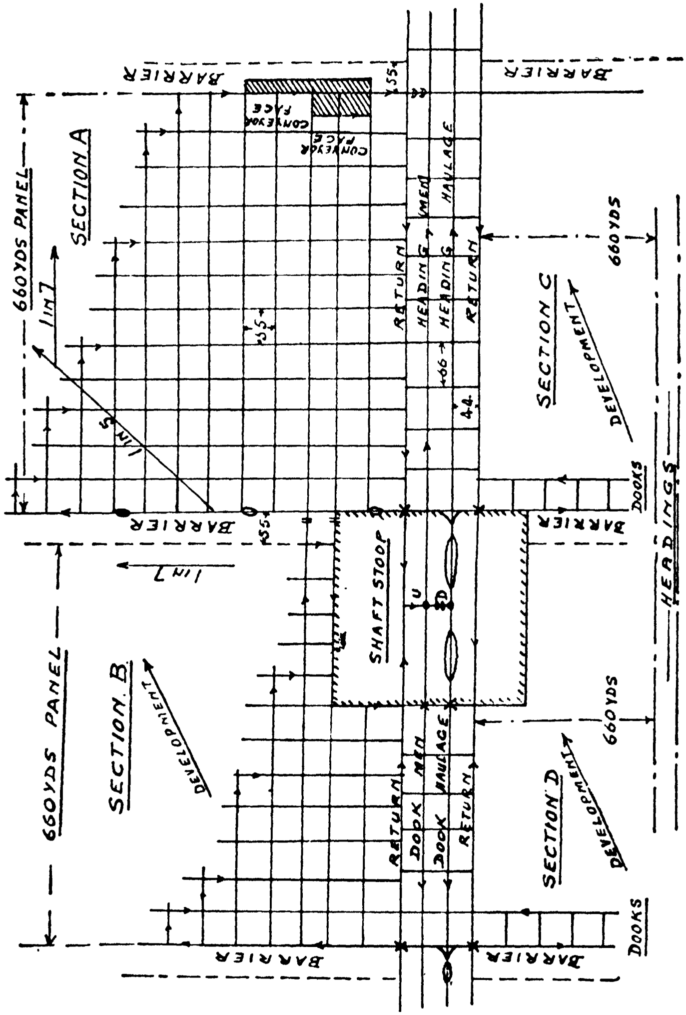

FIG. 14.—Plan showing Design of Pillar-and-Stall Workings.

Output from—

Section A:	18 fast places at 10 tons	= 180 tons
	2 conveyor faces $4\frac{1}{2}$ feet cut	= 450 ,,
Section B:	18 fast places at 10 tons	= 180 ,,
Sections C and D:	4 fast places at 10 tons	= 40 ,,

Total output = 850 tons per shift.

Counterbalancing Winding Loads by Balance Rope

2. What are the advantages and disadvantages of a balance rope under the cages? What method do you prefer for guiding the rope in the sump? Sketch how you would attach the balance rope to the cage. (40)

A. *Advantages of Balance Rope :—*

(*a*) It gives a perfect balance of winding loads in the shaft.

(*b*) Quick acceleration and easy retardation of the cages are obtained.

(*c*) It is smooth-running, reliable and efficient.

(*d*) There is very little risk of the engine losing its balance and running away.

(*e*) The decking of the cages at the surface and underground is not difficult.

(*f*) It is cheap in installation and upkeep.

Disadvantages of Balance Rope :—

(*a*) The rope might not loop properly in the sump and thus cause damage.

(*b*) There is an increased weight on the capping of the rope and the pulleys.

(*c*) If not applied properly there is an increased load on the keps at the surface.

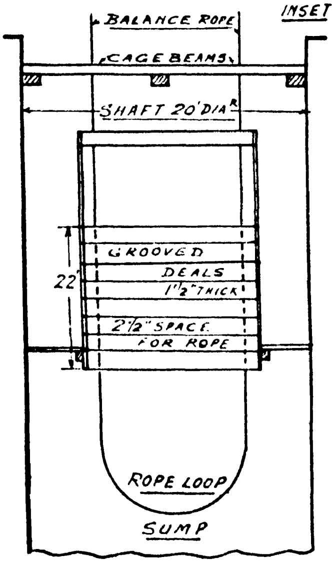

Fig. 15.—Guiding of Balance Rope.

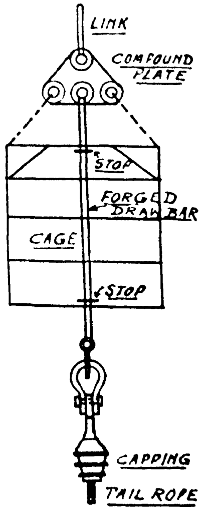

Fig. 16.—Attachment of Balance Rope to Cages.

Fig. 15 shows how a balance rope might be guided in the shaft sump by an arrangement of wooden deals. The rope loop is guided between the deals for a distance of 22 feet, so that a proper loop is formed in it, without the use of a pulley, in the sump.

The method of attaching the rope to the cages is shown in Fig. 16. The rope is capped in the usual way and is attached to the cage and chains respectively by a forged-steel drawbar, the latter containing suitable stops. When winding is in operation, the cage is against the stops on the drawbar. During decking operations the cage rests on the keps, and the weight of the balance rope is taken solely by the winding-rope.

Design of Shaft Pillar

3. Two shafts, 100 yards apart, the line joining their centres being east and west, are sunk through solid strata to a depth of 600 yards, to a seam of coal 5 feet thick dipping 1 in 5 to the south. Mark out a support pillar for the shafts and winding engine-houses, which are erected to the north of the shafts, and give sizes. Show the positions and approximate dimensions of the engine-houses on your drawing. (40)

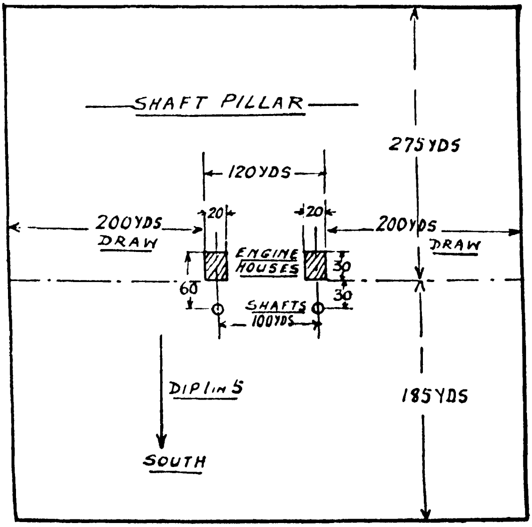

FIG. 17.—Design of Shaft Pillar.

A. Fig. 17 is a plan of the shaft pillar designed for the conditions given in the above question. The engine-houses are 30 yards by

20 yards in size and are situated 30 yards from the shafts. The rectangle formed by the shafts and engine-houses is 120 yards by 60 yards. The allowance for draw of the beds might be taken at the usual figure of 2 feet per fathom of depth for ordinary strata, or 200 yards in each direction from the boundaries of the rectangle already referred to. The shaft pillar will therefore be 520 yards on the level line and 460 yards along the dip-and-rise line.

Fig. 18 shows the possible fracture lines of the beds at right angles to the inclination of the seam, and explains, at the same time, why more coal must be left on the rise side than on the dip side of the

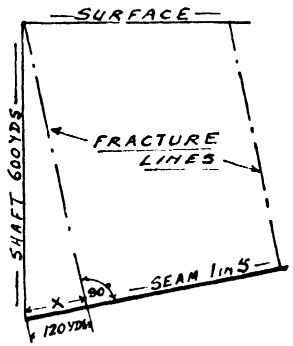

FIG. 18.—Possible Fracture Lines for Inclined Seam.

shafts. By drawing a perpendicular from the seam through the lip of the shaft, the dimension X, or deviation of fracture line from the vertical, is found to be approximately $\frac{600}{5}$ or 120 yards.

To find the rise and dip dimensions of the stoop, the following rule might be applied :—

$$\text{Rise side of Stoop} = \text{Dip side of Stoop} + \tfrac{3}{4}X \text{ (or 90 yards)} \qquad (1)$$

also

$$\text{Rise side} + \text{Dip side} = 460 \text{ yards} \quad . \quad . \quad . \quad (2)$$

and by substitution,

$$\text{Rise side} + \text{Rise side} - 90 = 460 \text{ yards} \quad . \quad . \quad (3)$$

$$\therefore \text{Rise side} = \frac{460}{2} + \frac{90}{2} = 275 \text{ yards.}$$

$$\text{and Dip side} = \frac{460}{2} - \frac{90}{2} = 185 \text{ yards.}$$

All dimensions are given on the design shown in Fig. 17.

Gate-End Loaders

4. Under what conditions has a gate-end loader certain advantages over filling coal from the conveyor direct into tubs ? Do you prefer the belt or scraper type of loader, and why ? When the face moves 4 feet 6 inches every 24 hours, describe the cycle of operations in connection with the gate-end loader. (40)

A. *Advantages.*—A gate-end loader has advantages over filling direct into tubs, as follows :—

(*a*) Where the pavement is soft, wet, or very hard, and it is not advantageous to cut it for making height and for roadside packs.

(*b*) Where the roof is suitable for the construction of roadside packs.

(*c*) Actual loading of coal takes place back from the coal face, and the main haulage might operate up to the loading point. Extensions to the loader might be carried out up to 25 feet, so that moving forward of the loader and haulage can be done weekly.

(*d*) The face of the loading road is always in line with the general line of face, and coal-cutting operations can be continued right down to the extremity of the face, thus giving a reduction in cost for narrow work and pick-work on the face.

(*e*) The difficulty of loading into tubs with a double-unit face is completely removed by the use of a loader.

A gate-end loader might have to deal with large stones from the ripping in addition to coal, and in these circumstances a scraper loader is preferable. Power-consumption cost is more for the scraper loader than for the belt loader, but the first cost and upkeep cost are less for the scraper loader. The scraper loader is positive in action, and there is no running back of the loader, or the material loaded thereon. Tensioning of the belt is a more difficult operation than chain adjustment of the scraper, while length adjustments for weekly operation are not more difficult with the scraper than with the belt loader.

The cycle of operations in connection with a loader might be as follows :—

(*a*) Protecting the loading end by timber during the operation of ripping the roof and building the roadside packs ;

(*b*) Filling any surplus material on to the loader for delivery into tubs and conveying to other parts of the mine ;

(*c*) Moving the loader forward into its new position by the use of sylvesters to coincide with the new position of the conveyor ;

(*d*) Finally, the loading-plates, rails or haulage are advanced according to requirements so as to be ready for coal work again.

Substitutes for Blasting

5. What methods, other than by means of explosives, can be used for breaking down hard coal after being undercut by a longwall machine? Give your opinion of the advantages and disadvantages of each. (40)

A. In addition to the various methods applied for wedging down coal, including the recently improved hydraulic wedge or Coal-burster, the use of the Hydrox shell and of pneumatic picks are two up-to-date methods of breaking down machine-cut coal.

The **Hydrox shell** is of steel, about 1½ inches in diameter and 3 to 3½ feet long, containing a firing head and a discharge head. A charge of chemicals, in powder or granular form, is inserted into the shell, which is then sealed by a steel disc and the discharge head. The charge is ignited by an electric powder-fuse in the firing head. The design of the discharge head allows rapid release of the gases, and by having the discharge holes swept back, there is a tendency to drive the tube forward in the hole, so that the gases escape where they are required to do the work. The pressure exerted at the back of the hole is generally sufficient to break down hard coal. The holes in the coal, to receive the shell, might be drilled quickly by the use of an electric drill of the rotary type.

Fig. 19 shows in diagrammatic fashion the construction of the

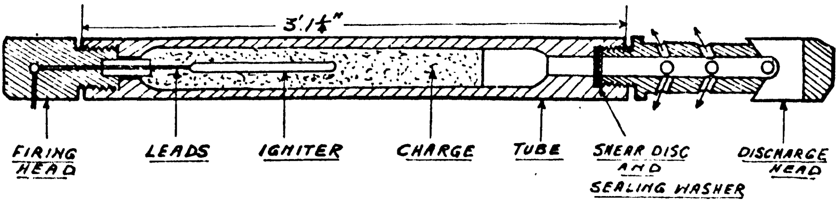

FIG. 19.—Hydrox Shell for Breaking-down Coal.

Hydrox shell. When the charge is ignited, a relatively large volume of gas is released by the rupturing of the disc in the steel tube. Such a device eliminates the necessity for having a costly plant to refill steel cylinders with liquid carbon dioxide, as in the case of the Cardox shell. The shell is recovered after use and sent to the surface for recharging and refitting of the disc. The following equation shows the reaction of the chemicals used:—

$$NH_4Cl + NaNO_2 = NaCl + 2H_2O + N_2$$

Ammonium chloride + sodium nitrite = sodium chloride + water + nitrogen

The products of combustion are thus non-inflammable, efficient preventers and suppressors of flame, and non-toxic.

Advantages of the Hydrox Shell :—

(*a*) Pulverising of the coal is greatly reduced compared with blasting, and the percentage of round coal is greatly increased.

(*b*) Dirt bands at the roof of a seam can be left intact until the coal is removed, and thus cleaner coal is obtained.

(*c*) The roof is not broken and weakened as with explosives, thus giving greater safety to the workmen employed.

(*d*) The cost of roof supports is greatly reduced.

(*e*) Greater safety exists as regards explosions of firedamp and coal-dust, the tube having been fired, under test, in the most explosive methane–air mixture without causing an ignition.

(*f*) Noxious fumes are not produced, and a cleaner atmosphere exists at the coal face than with explosive compounds.

(*g*) There is no risk in recharging a hole after a failure.

Disadvantages :—

(*a*) The danger of the shell being forced from the hole without breaking down the coal.

(*b*) Cumbersome for handling and storage in a mine.

The **pneumatic pick** weighs about 19 lb. and requires an air pressure of about 90 lb. per sq. inch. The compressed air might be conveyed to the face by pipes 3 inches in diameter, while a range of pipes 1 inch in diameter, with suitable valves for hose connections to the picks, might be run along the longwall face.

Advantages of Pneumatic Picks :—

(*a*) The amount of slack coal is greatly reduced compared with that produced by using explosives, while a similar reduction might be possible in the ash content of the slack coal.

(*b*) Dirt bands can be removed before the coal, and thus cleaner coal is produced.

(*c*) Better roof conditions exist than with explosives, and accidents at the face are reduced.

Disadvantages :—

(*a*) Vibration is considerable, and this causes breakages of the parts.

(*b*) The valves are liable to stick and cause stoppages.

(*c*) The maintenance cost is rather high.

(*d*) Men have to be trained in the use of the picks for successful application.

MAY 1933 EXAMINATION

Layout of Sidings and Main Roads near Shaft Bottom

1. Sketch a layout of the sidings and main haulage roadways near the shaft bottom to deal with an output of 1,500 tons per shift, from a seam dipping 1 in 4. The tub to hold 10 cwt. (50)

A. The general layout of the roads for the given conditions is shown in Fig. 20. The arrangement includes a design for working two-thirds of the coal to the rise and one-third to the dip. The shaft landings are 150 yards long and are driven to have a gradient of 1 in 40 in favour of the full tubs. The main roads are designed for endless-rope haulage, with a gradient of 1 in 15 in favour of the full load. The inclines to the full rise and dooks to the full dip are designed to work the coal in panels of 300 to 400 yards in width.

The details of the shaft landings and decking appliances are shown in Fig. 21. Duplicate full sidings are graded at 1 in 40, while the duplicate empty sidings are 1 in 30. The cages are assumed to be double-decked with four tubs on each deck. The whole arrangement is laid out for the full and empty tubs to run by gravity.

Fig. 21 shows also a sectional view of the decking arrangements for simultaneous decking with drops and sunk flats, 4 tubs being dealt with on the top flat and 4 at the sunk flat for each wind of the cages.

Explanation of Geological Terms

2. Sketch and describe what you understand by the following :—
(*a*) Fault ; (*b*) Reversed Fault ; (*c*) Roll ; (*d*) Dyke ; (*e*) Fissure ; (*f*) Syncline ; (*g*) Pot-hole ; (*h*) Wash-out. (40)

A. *Fault.*—This might be described as a slip or crack in strata, accompanied by a displacement of the beds. It is caused by earth movements in which part of the crust has slipped down to a lower level.

Reversed Fault.—Sometimes strata are subjected to a lateral pressure at a great depth, owing to masses of the crust slipping down. This results in one part of a bed being forced under or over another part of the same bed along shear planes, and a double thickness of strata is found. Such an occurrence produces a reversed fault.

Roll.—When a coal seam is indented at the roof or at the floor

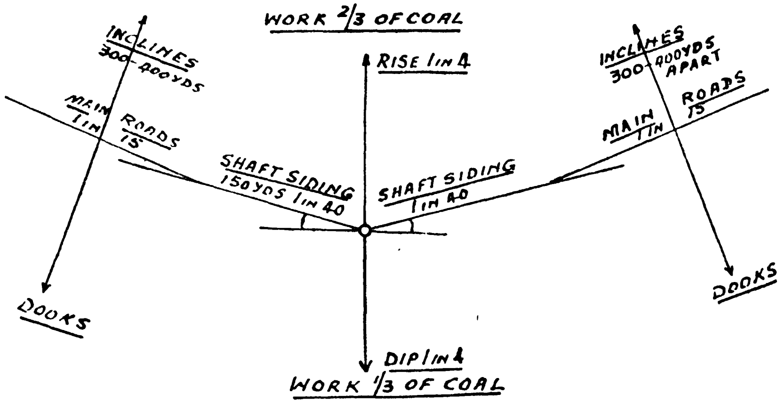

FIG. 20.—Plan of Sidings and Main Roads for Seam Inclined at 1 in 4.

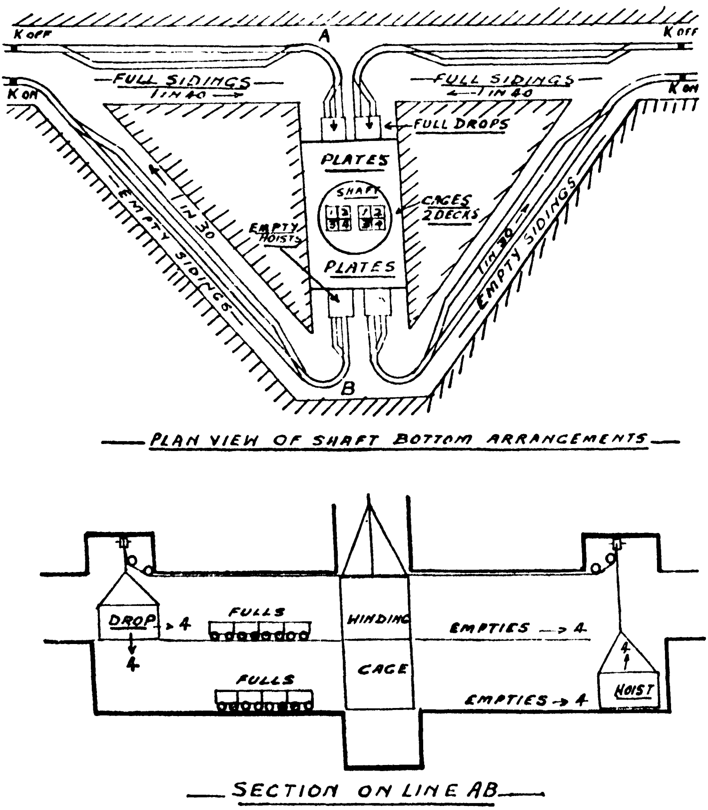

FIG. 21.—Shaft Bottom and Cage-decking Arrangements.

by a portion of strata of a hard nature, such an occurrence is known as a roll.

Dyke.—Strata are often cut through by a wall-like mass of igneous rock which has been forced through them in a molten state and has finally cooled and hardened. In the case of a coal seam, such a mass might displace the seam for several yards, with or without vertical displacement. An occurrence of this kind is known as "dyke formation."

Fissure.—When strata are merely cracked without any displacement of the beds, a fissure is said to be formed. It is from fissures in strata that water and gas issue into mines.

Syncline.—Beds are often found to be of a folded nature owing to movements of the earth's crust. The beds forming the depressions of the folds are generally preserved and are termed synclines.

Pot-hole.—Water running over rocks has a scouring action, and

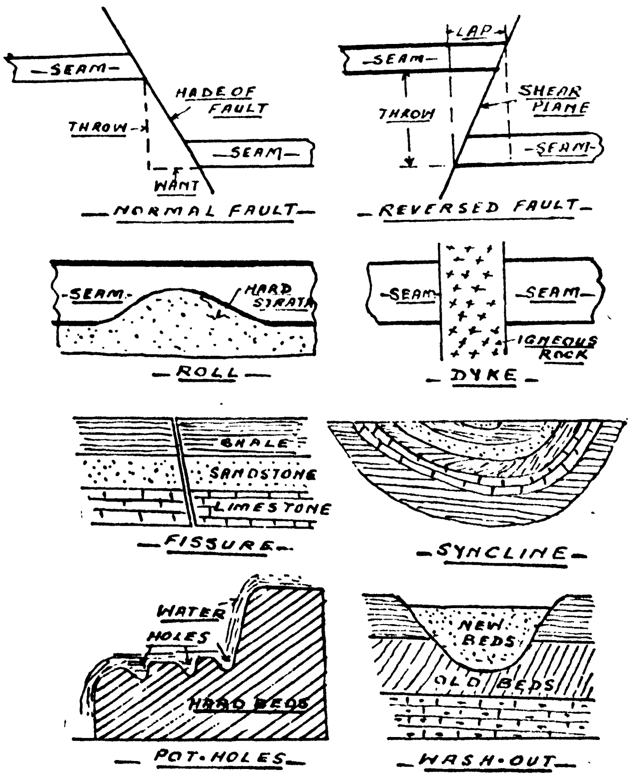

Fig. 22.—Illustrating Geological Terms.

softer portions are worn away in circular fashion to form pot-holes. Water falling upon strata from a higher level also generally produces pot-holes.

Wash-out.—A stream of water running over strata usually washes away part of the beds in the form of a trough. After a time, newer beds might be deposited in the trough. A wash-out is formed in this way.

Fig. 22 shows sketches of the various geological features described above.

Gate-End Loader *versus* Gate-Belt Conveyor

3. With machine mining at the coal face, under what conditions is it desirable to install a gate-end loader or a gate-belt conveyor? Under what conditions does the gate-belt conveyor justify the additional cost over the gate-end loader? (40)

A. *Conditions for Gate-End Loader :—*

(*a*) Where the roof of the seam is suitable for building good roadside packs for a large gate road.

(*b*) When the floor of the seam is soft, wet or very hard, and unsuitable for making roadside packs.

(*c*) Where the cost of ripping is low for the making of high roads.

(*d*) Actual loading takes place back from the face, where the main haulage starts, with easy gradients for haulage and movement of tubs.

(*e*) When the heading or gate road can be kept in line with the longwall face, and coal-cutting can be accomplished down to the rib side of the coal.

Conditions for Gate-Belt Conveyor :—

(*a*) When a double-unit face is working with belt conveyors.

(*b*) Where the cost of ripping is high for large roads.

(*c*) When peak loads are experienced during a working shift.

(*d*) In the event of the time being limited for moving the loader forward and re-laying flat sheets.

(*e*) When all the face is cut with no advance headings.

(*f*) Where the roof is of a soft nature and narrow roads are necessary.

The gate-belt conveyor justifies its additional cost under the following conditions :—

(*a*) Where the time is limited between shifts, the work of extending the belt conveyor being more easily carried out than that for the extension of the loader.

(*b*) When the inclination of the seam is high, thus giving more favourable conditions for the gate-belt conveyor.

(*c*) Where the gate roads are going to the dip of the seam.

(*d*) Where peak loads are experienced, better arrangements being possible on the main haulage road for the standage of tubs.

(*e*) Where haulage extension to the face is difficult to carry out in the area of roof settlement.

(*f*) Where roof settlement is going on, narrow roads up to 500 yards in length being more easily maintained in good condition in the area of settlement.

Cage Guides in a Winding Shaft

4. Under what conditions would you use rigid guides in a shaft? Which type do you prefer, wood or rail, and why? Show, by sketch, how both types are secured in the shaft. How far apart would you place the bearers or buntons? (40)

A. *Rigid steel guides* might be installed in shafts under the following conditions :—

(*a*) Where space is limited and insufficient for the installation of flexible guides.

(*b*) In wet shafts where rope guides would be subject to corrosion.

(*c*) When it is desired to have heavy loads and quick winding in deep shafts.

(*d*) In shafts designed for a long life, for smooth winding conditions and low upkeep cost.

The section of guide might resemble that of wood guides to give good working conditions.

Rigid wood guides might be installed in shallow shafts, with light loads and low winding speeds. If applied for deep shafts, unequal wear, loose bolts and splitting of the wood are liable to cause accidents.

Steel guides are preferable to wood guides owing to their great strength and freedom from breakages and mishaps.

Fig. 23 shows channel steel guides as fitted in a circular shaft and applied at the ends of the cage. They are built up in sections 18 feet long and weigh 22½ lb. per foot. The guides are fitted to hoops on buntons placed 6 feet apart in the shaft. The enlarged section shows bunton, hoop iron for holding the guide in position, and the section of the guide.

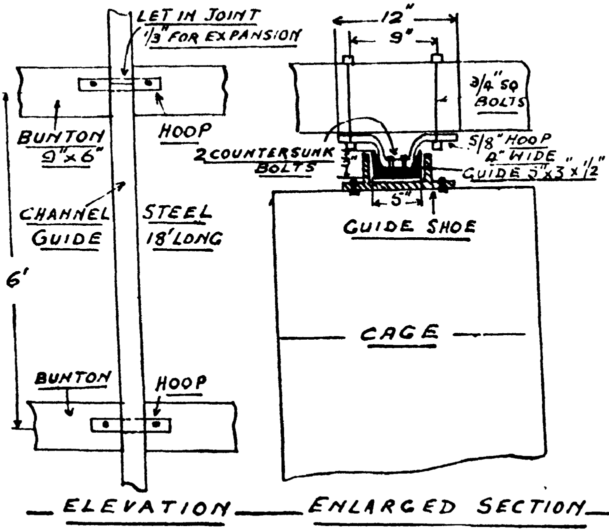

Fig. 23.—Channel-steel Cage Guides in Circular Shaft.

Fig. 24 shows wood guides applied to the side of the cage. The guides are fitted to buntons in the shaft and are installed in lengths of 12 to 40 feet. The enlarged view shows a butt joint of the guides supported and stiffened by a cover plate. Battens are used between the guides and the buntons to protect the latter in the event of excessive wear of the guides.

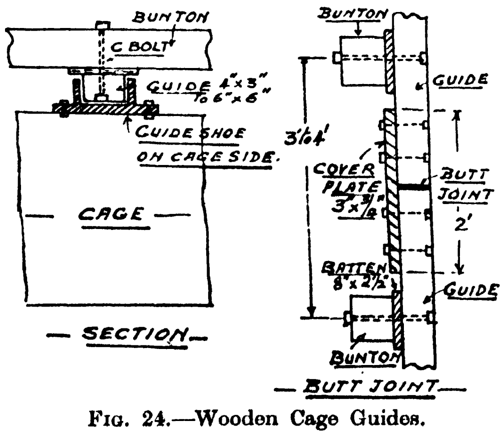

Fig. 24.—Wooden Cage Guides.

Buntons vary in size from 4″ × 4″ to 9″ × 6″ according to the size of shaft and method of guiding the cages. The smaller sizes are fitted 3 to 4 feet apart and the larger 6 feet apart.

Use of Sand for Stemming Shot-holes

5. It is now established that sand is better than clay for stemming shot-holes. What are the important advantages claimed for sand? What do you consider is the best way to treat the sand so as to make it of a suitable consistency for easy handling? (40)

A. The important advantages claimed for sand stemming are as follows :—

(*a*) Clay stemming produces a shattering action with detonation and noise, while sand stemming gives more of a breaking action with a rumbling noise and greater effect.

(*b*) There is a marked absence of flame and fumes with sand stemming, and greater safety is obtained by reducing blown-out shots and the danger of explosions.

(*c*) Owing to the explosive charge doing more useful work with sand stemming, there is a saving of upwards of 20 per cent. in the explosive charge.

(*d*) Experiments on a large scale have proved that 90 per cent. of clay stemming is ejected from the shot-hole, as compared with 6 per cent. of sand stemming.

(*e*) It is claimed that 1·75 feet of sand stemming are more effective than 4·5 feet of clay stemming for the same explosive charge.

The stemming material for shot-holes might consist of 3 parts sand and 1 part finely ground clay made into a stiff paste by the use of water. An addition of 3 to 5 per cent. of calcium chloride tends to keep the stemming material moist in the pit.

Sinking through Heavily Watered Strata

6. After sinking a shaft to a depth of 150 yards, heavily watered strata 40 yards in thickness are met with. Describe the method you would adopt whilst sinking through this ground. (40)

A. The most suitable method of sinking to be applied for the above conditions is the *François Cementation Method.* Cementing and sinking proceed alternately until the water-bearing strata are sunk through. The plant required consists of mixing tanks and cementing and chemical pumps. The cement milk is mixed in tanks provided with revolving paddles to ensure proper mixing. The milk thus produced is then properly sieved before passing to the cement pump and pipes. The pressure given to the milk cement depends upon the depth and nature of the beds.

Fig. 25 shows the arrangement of 40 cementing holes in a shaft 25 feet in external diameter. The holes are bored 3 inches in diameter to a depth of 15 feet and are fitted with 2-inch cementing tubes which project to 6 inches above the tops of the holes, the tubes being cemented and caulked with lead wool. The cementing holes are not continuous throughout the whole depth. They are tested by hydraulic pressure after being cemented, and fresh holes are drilled after sinking operations have been suspended owing to the presence of water.

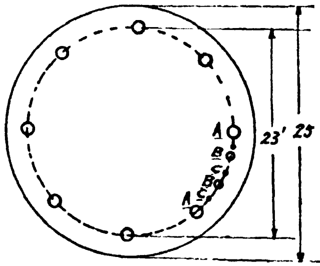

Fig. 25.—Cementation Holes for Sinking.

Cementation is carried out in three operations. Eight holes (A), equally spaced round the circumference, are advanced in stages of 10 to 20 feet, and they are given an injection of silicate of soda, silicate of ammonia and cement. The chemicals used are very soluble, enter the finest fissures and form a colloid to lubricate the cement, and at the same time produce rapid-hardening. Sixteen holes (B), equally spaced, are cemented separately, half of them being treated with chemicals. A further sixteen holes (C) are given a higher proportion of chemicals to ensure the filling up of fine fissures.

Sinking is carried on in the usual fashion, after all the holes have been cemented and tested, and until water increases in volume in the shaft. No side holes are used in shot-firing, the sides being chipped by pneumatic picks. The sinkers are protected by sheet steel lining in segments 4 feet 8 inches high, with 5 or 6 segments to the circle, and supported by $1\frac{1}{2}'' \times 1\frac{1}{4}''$ angles. This lining is not withdrawn from the shaft.

Fig. 26 shows the permanent shaft lining: (a) Timber supports in sump; (b) steel bedplate on $1\frac{1}{2}$-inch diameter steel plugs in bore-holes; (c) concrete base wedge with holes for water passage; (d) reinforced concrete lining with $\frac{3}{4}$ to $1\frac{1}{2}$-inch diameter bars placed horizontally, tied by vertical stirrups to the back sheeting angles, and faced with a falsework of light steel tubbing which is removed after the concrete has set; (e) permanent steel structure with 14-inch slit every 20 feet for filling in pebbles; (f) a space 3 inches wide filled with $1\frac{1}{2}$-inch pebbles and cemented after the concrete has set; (g) sheet-steel protection lining already referred to for protecting the sinkers.

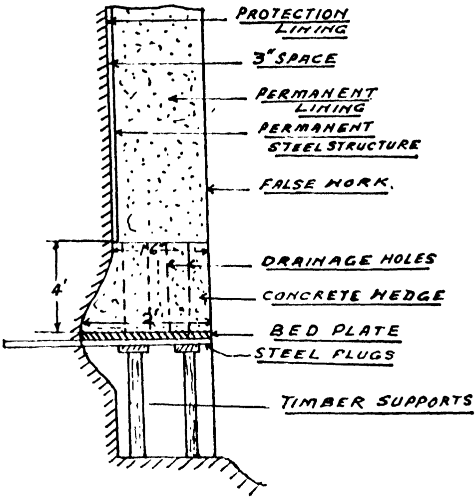

FIG. 26.—Permanent Shaft Lining after Cementation.

NOVEMBER 1933 EXAMINATION

Layout of Workings and Haulage Road

1. Sketch a layout of workings and haulage roads in a coal seam 5 feet thick and dipping at 1 in 4, to yield approximately 1,000 tons from one shaft in one shift. State the output you would expect from the separate roads, also the type of haulage you would use in each of these roads. (50)

A. Fig. 27 shows a layout of panel workings in a coal seam 5 feet thick dipping at 1 in 4. The narrow headings to the full rise, and the levels connecting the same, are driven by a machine of the arc-wall type, and are 10 feet wide with an undercut of 5 feet, each producing 8 tons per shift or about 100 tons in all from 14 places. The coals are delivered to the endless haulage at the point *A* by hutches in the usual way by the back headings. The panels are 700 yards wide by 500 yards long. Connecting levels in pairs are driven between each pair of headings every 500 yards to form the panels, and singly every 200 yards for belt roads.

The retreating faces are 200 yards long and are cut to a depth of 4 feet 6 inches by a chain machine. The double-unit faces cut in this way are worked by a shaker conveyor from the rise and a belt conveyor from the dip. Both conveyors deliver into a belt conveyor on the level road, and this in turn delivers to a main road belt conveyor of 400 yards maximum length. The latter conveyor delivers the coals to tubs from an endless-rope haulage, which in turn delivers to the shaft.

The panel shown fully developed is capable of an output of 450 tons from each double-unit face, or 900 tons from each pair of headings, in two shifts. The output per shift is therefore 900 tons from retreating faces and 100 tons from narrow places of developing panel.

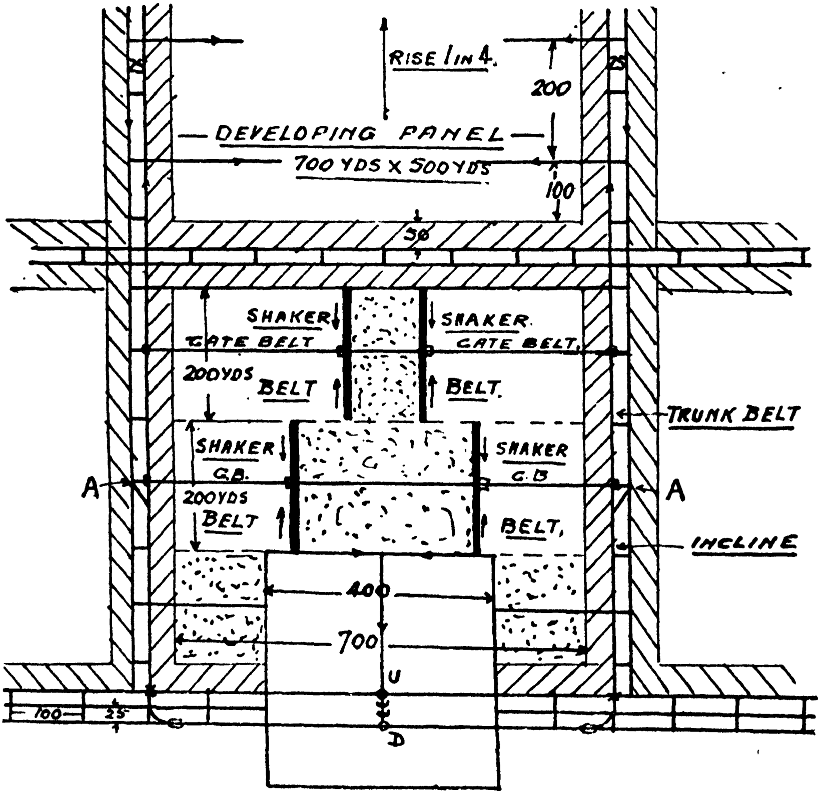

FIG. 27.—Plan showing Design of Panel Working and Machine Mining.

Materials forming the Roofs and Floors of Coal Seams

2. Specify the kinds of material that may form the roof and floor of a coal seam, and indicate how their qualities may affect the working of the seam. What difference would you expect to find in the material immediately below the seam, if the seam had been formed *in situ*, or if it had been deposited according to the drift theory ? (40)

A. **Roof Materials.**—(*a*) The roof of a coal seam might consist of soft shale, and the roof supports are liable to be pushed up into it, thus proving to be ineffective. Shearing of the shale might take place between the supports and along the line of face. A bending action without shear might result where proper supports are used, such as yielding supports and chocks. Under the first-named conditions, it might be necessary to work the seam by the pillar-and-stall method in preference to longwall, as applied for the latter conditions. (*b*) Sometimes a roof consists of brittle shale and bind, which tend to fracture with very little bending action. If the roof supports subside in any way there is a grave danger of failure of the roof along the working face. For these conditions good rigid supports and good stowing of the waste workings are essential for the longwall method of working. (*c*) If the roof of a coal seam consists of hard sandstone, which causes heavy crush on the face supports, the danger of large and extensive falls must be guarded against. Such a roof might overhang in the waste near the face, and, in addition to good rigid face supports, it would then be necessary to have strong waste-line supports in the longwall workings.

Floor Materials.—(*a*) The beds forming the floor of a seam might be of a hard nature and thus assist in properly supporting the roof. (*b*) Sometimes the beds under a coal seam are of a softer nature than the coal, and in such cases settlement of the supports and roof fractures are experienced. (*c*) The pavement of a seam might be wet and the underclay would then undergo a softening process, thus making a roof support difficult and causing creep in the workings.

In considering roofs, in general there are two sections to provide for, the near roof or immediate cover, and the far roof or portion which remains hanging for some time after the former has fallen. In longwall workings three systems might be applied, known as the Caving System, Solid-packing System, and Strip-packing System respectively. The Caving System might be applied where the roof is very strong and does not tend to bend and break immediately after working the coal; also where the pavement is hard and favourable for rigid supports. Such supports, combined with waste-line chocks, can be applied, and the caving of the roof behind them fills in the space left by the extraction of the coal. Good roadside buildings would be required to protect the gate roads. The Solid-packing System is usually applied where the roof and pavement are of a soft nature, and where great difficulty would be experienced in keeping the roads open by other methods of packing. Strip-packing at suitable distances apart is necessary in most cases where the roof bends freely before fracture after the coal is extracted by the longwall method of working.

Differences in Floor Material of Coal Seams.—When a coal seam has been formed *in situ* the floor material usually consists of underclays of a hard or soft nature. Sometimes there are intervening bands of hard sandstone (or " post ") and thin layers of coal. If formed in accordance with the *drift theory*, the floor of a coal seam might consist of older rocks of the Devonian and Silurian systems, as experienced in South Wales and the Sanquhar coalfield of Scotland. The material might also consist of granite rock, as found in the French coalfields.

Lining a Roadway with Concrete Blocks

3. A seam is to be worked under a cross-measure drift. Anticipating a considerable movement of the strata, it is decided to line the drift with concrete blocks. What shape of block would you use, and how would you carry out the work ? (40)

A. The *Schäfer system* of concrete lining would be suitable for the given conditions. Fig. 28 shows sectional views of the finished road. Arches made with concrete blocks are 20 inches wide and are placed in the roadway with 40-inch centres. The blocks are

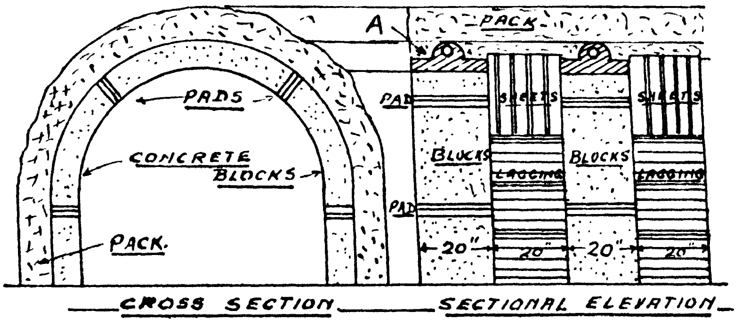

FIG. 28.—Schäfer Method of Road Lining.

tied together by wire ropes and are laid in liquid cement. Several compression pads of wood are included in each arch. The construction and shape of the concrete blocks are shown at *A* in the diagram ; they are of the stunted stem type. The intervening spaces are filled in with packs, laggings and corrugated-iron sheets.

There is also a packing between the surfaces of the roadway and the linings.

Conditions for Under-rope and Over-rope Endless Haulage —Haulage Clips

4. Discuss the conditions most suitable for under-rope and over-rope endless haulage respectively. How would you attach the tubs to the rope in each case ? (40)

A. The under-rope system of endless haulage is suitable where the inclination of the haulage road is uniform. It might be operated successfully in both flat and highly inclined seams. The loading of trams can be carried out at any required intermediate station to suit modern mining conditions. Curves and branches are easily worked by arranging for automatic detachment of the trams from the rope, so that they are free to pass at junctions, the pulleys and ropes being placed under the track at these points.

The over-rope system of endless haulage is suitable where the road inclination is moderate, and also where the road is undulating. Loading of the trams at intermediate stations is difficult, and the arrangement is not very suitable for modern mining conditions. Moreover, the rope tends to displace coal from the tops of the trams, and the system is not suitable where a certain amount of topping is carried out. The weighting of trams by heavy ropes at changes of gradient and at curves tends towards greater friction. Rope friction on roads is low, owing to the rope being above ground level ; for the same reason, however, branches are not easily worked.

Haulage Clips.—A clip suitable for under-rope haulage is the *Smallman* type. It gives a firm grip on the rope, and trams can

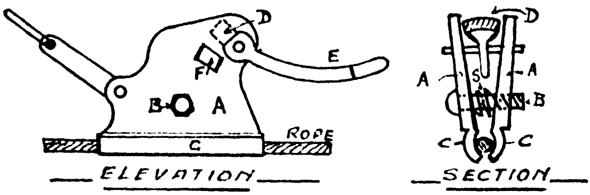

FIG. 29.—Smallman Haulage Clip.

be attached to the rope in sets instead of singly. This clip consists of two steel cheeks *A*, loosely connected near the base by the pin attachment of the link, by a similar attachment for the bent lever *E*, and by a bolt *B*, which is threaded sufficiently at the end for a nut. The two pins mentioned above are made a sliding fit for the holes in the cheeks. The bolt *B* passes through a winged collar fitting into a recess in one of the cheeks, and a winged nut fitting into a similar recess in the opposite cheek. A spring threaded on the bolt between the two cheeks keeps them sufficiently apart for the lever *E* and the block *D*. The distance between the cheeks is determined by the length of the bolt, which is adjustable in the winged nut.

The rope is clamped between the lower jaws *C*, which are about 6 inches long and are bushed with soft iron. The block *D* carried at the end of the bent lever *E* moves in a recess in the cheeks, which becomes shallower towards the upper edges. When the lever is forced down, the block forces the upper edges of the cheeks apart, and the lower edges of the cheeks thus come together to grip the rope. This clip can be detached automatically from the rope by having a suitable device to lift up the lever *E* and thus set the rope free. The hole *F* in the cheeks allows of the block *D* passing into a wider space so that the jaws may be opened wide enough for slipping on to the rope.

The *Rutherford and Thompson* clip is suitable for over-rope endless haulage and is shown in Fig. 30. This clip is easily operated,

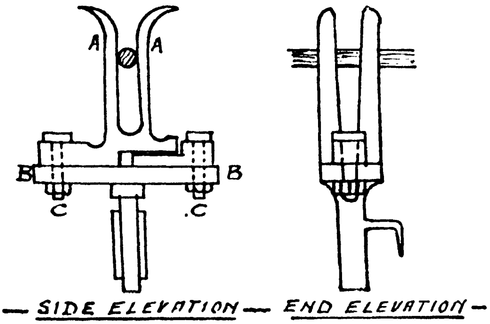

FIG. 30.—Rutherford and Thompson Haulage Clip.

has a good gripping power on the rope for steep and undulating gradients, and is automatic in action.

The clip is hung over the end of the tram by the hook on the inside, and the stem passes through a loop fixed in the front of the tram. An appliance might also be used to prevent the clip being lifted on undulating gradients. This clip consists of two Y-forked jaws *A*, mounted and geared together on a base-plate *B*, so that they can oscillate about two pins *C*. The teeth of the gear are specially shaped to limit the extent of the forward and backward movement of the jaws. When the clip comes into line with the rope, the motion of the latter turns the forks about the pins and causes them to grip. The stronger the pull the tighter is the grip. As the jaws can move either backward or forward, the clip works well on undulating gradients. The rope can be lifted from the hook by a pulley at any point where the tram is travelling on a down-grade. It can be attached to the clip at any point where the trams are standing on a level or on a rising gradient.

MAY 1934 EXAMINATION

Fractures in Roof on Longwall Face

1. A level seam, 5 feet thick, is undercut to a depth of 4 feet 6 inches each day on a longwall face. The roof is fairly hard shale and 60 per cent. thereof is supported by packs. Make a cross-section of the face, showing in detail how you would expect to find the lines of fracture in the roof; and under what conditions would you expect the roof (*a*) partially to lock itself, (*b*) to be liable to continued falls? What difference would you anticipate in the roof if the same seam was got by hand? (50)

A. Fig. 31 shows a cross-section of the face with details of fracture lines for the conditions given in the above question.

Area A.—In this area the strata are assumed at rest in advance of the workings.

Area B.—The strata in this area are set in motion by movements

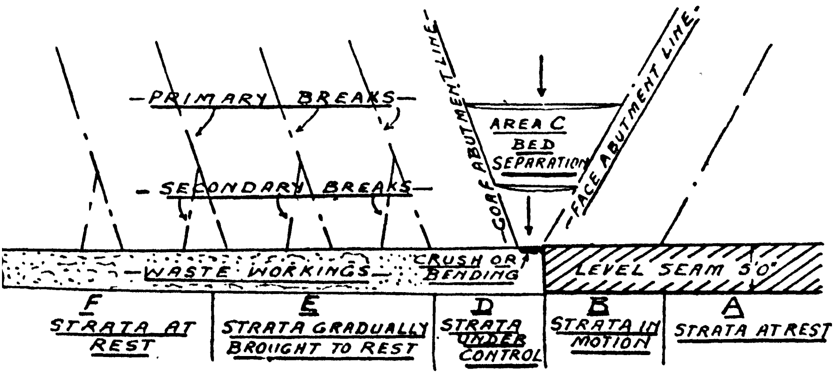

FIG. 31.—Cross-section of Face showing Roof Fracture Lines.

of the face caused by undercutting of the coal, stripping of the coal, and withdrawing of supports at the edge of the waste.

Area C.—Bed separation takes place in this area, between the face abutment line and the goaf abutment line. In the higher beds the weight of the strata is supported by abutments. In the lower beds the strata are supported by face supports.

Area D.—The beds in this area are controlled by face supports for the lower beds and by packs for the main beds. Breaks occur in this area when excessive subsidence takes place between the face and the waste workings, or when the rate of change of movement of the beds is high. Some of the beds are fractured parallel to and a few feet in front of the face, these fractures being mostly inclined towards the waste and known as "primary breaks." Sometimes local breaks are formed along the front line of props when the space is too wide, or the undercut coal is not spragged. Such breaks are termed "secondary breaks." They are nearly vertical, or are inclined towards the coal. The two breaks referred to form an inverted V which is liable to fall out.

Case I.—If the coal is properly spragged, props set close up to the undercut coal, strong face supports used in systematic fashion after stripping, and, finally, good packs inserted, the roof will bend over the face and packs with a low rate of change of movement, and safe working conditions will exist.

Case II.—If the above precautions are neglected, severe crush may arise, causing fracture of the beds with complete loss of strength. In this case falls of roof will occur.

Area E.—In this area the beds are brought to rest gradually by goaf and roadside packs.

Case I.—If the restoration of equilibrium is fairly quick, owing to good packs, the beds are unbroken, their strength properties are maintained, and there is a margin of strength to meet new disturbances.

Case II.—If the restoration of equilibrium is slow, owing to defective packs and weak roof supports, interlocking and crushing of broken beds take place, and the strength properties of the beds are nullified.

Area F.—The strata are once more at rest, equilibrium having been again established in the waste workings.

Roof partially locking itself.—This might occur by bed separation between abutments in the higher beds only, or by bed separation in the lower beds, combined with interlocking and crushing, and a lateral pressure at the break.

Numerous Falls.—In the latter case, the beds cannot recover their strength properties, and falls occur when they suffer reversal of curvature in working. Breaks will occur parallel to the face line. Secondary breaks will be induced, and numerous falls will result.

Hand-work.—Extensive stripping of coal on a longwall face, without undercutting, causes a series of sharp increases in the movement of beds. Near projections of coal, there are very irregular movements. Well-developed breaks in advance of the working face are produced, forming loose wedges of strata. Numerous falls may occur when the coal is removed. More shots are required to bring down the coal at the face, and in this way bad working conditions are aggravated.

The Carboniferous System and its Relative Geological Position

2. Name the subdivisions of the Carboniferous System and give the position it occupies in relation to the preceding and subsequent systems. (40)

A.

Subsequent Systems	Post-Tertiary System	
	Tertiary ,,	
	Cretaceous ,,	
	Jurassic ,,	
	Triassic ,,	
	Permian ,,	
Carboniferous System	Carboniferous Red Sandstone (Scotland)	
	Coal Measures	Upper
		Middle
		Lower
	Millstone Grits and Ganister Beds of Fireclay	
	Carboniferous Limestone with Coal (Yoredale Rocks)	
	Carboniferous Limestone without Coal	
	Calciferous Sandstone with Oil Shales (Scotland)	
Preceding Systems	Devonian and Old Red Sandstone Systems	
	Silurian System	
	Ordovician ,,	
	Cambrian ,,	
	Archæan ,,	

Layout of Incline for a Balanced Double-Drum Haulage

3. Five hundred tons a shift have to be brought up an incline 1,000 yards long and dipping 1 in 4. A balanced double-drum direct haulage, capable of hauling 20 tubs on each rope, is installed. Explain, with sketches, how you would lay the rails at the top of the incline, and explain in detail the system you would use at the top and bottom of the incline so as to get the ropes moving again expeditiously. (40)

A. Fig. 32 shows by a plan view the general layout at the top and bottom of the incline, the positions of haulage gear, main

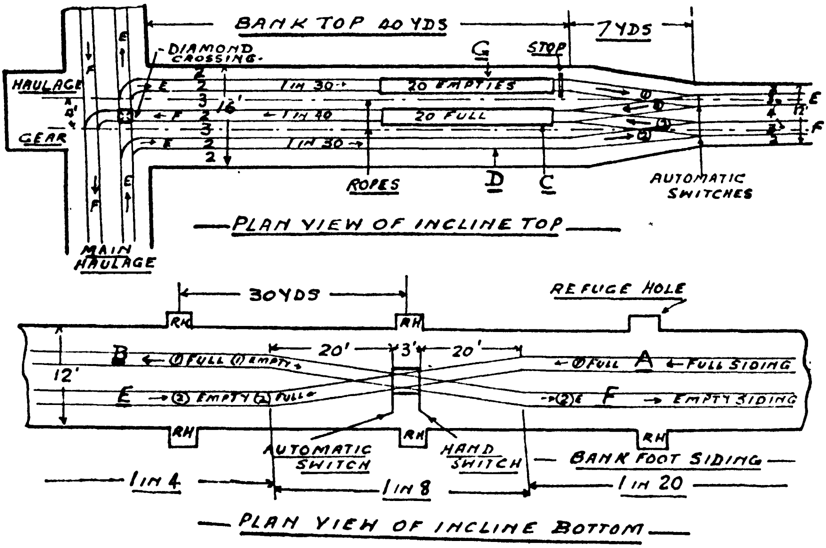

FIG. 32.—Arrangement of Balanced Double-drum Haulage Incline.

haulage road, and incline being included. The top of the incline is provided with three sets of rails for a distance of 40 yards, the centre one being for full tubs and the outside ones for empties. The bottom of the incline is graded down to allow of easy starting of the haulage, and it is provided with full and empty sidings. A full load is drawn to the top by way of roads *A*, *B* and *C*, while at the same time an empty load runs down by way of roads *D*, *E* and *F*. The ropes are easily changed by transferring from the full load in siding *C* to empty load in siding *G*, and from empty load in siding *F* to full load in siding *A*, so as to be ready for another trip. The full loads are drawn alternately from the full siding at the bottom by way of tracks *B* and *E* to the centre siding *C* at the top. The empty loads leave sidings *G* and *D*, alternately, for tracks *B* and *E*, and are switched into the empty siding *F* at the bottom of the incline.

Sinking through Drift Clay and Soft Water-bearing Strata

4. A shaft is to be sunk on the coast. There are 30 feet of drift clay, then 100 feet of soft water-bearing strata containing sea-water overlying the rock head. Explain generally how you would get through to the rock head. (40)

A. *Drift Clay*, 30 *feet*.—A simple and reliable method of sinking through drift clay would be the *Riemer method*, in which underhanging internally-flanged tubbing is used. Fig. 33 shows diagrammatically how this system is applied. The hanging ring for sus-

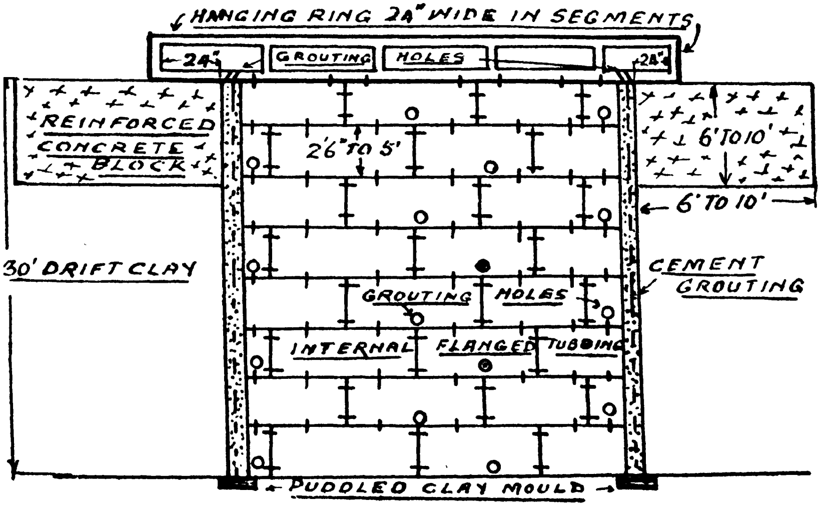

FIG. 33.—Riemer Method of Shaft-sinking.

pending the internally-flanged tubbing is laid upon a reinforced concrete block of sufficient dimensions to prevent subsidence. As the sinking proceeds in the usual way, the tubbing segments are secured to the hanging ring by truly turned bolts, sheet-lead packing, and caulked joints. Every alternate segment has a grouting hole for cementing the back of the tubbing. The clay is removed by sinkers in the usual way, using planks and short spiles, until sufficient distance is obtained to allow of more segments being added to those already in position.

Soft Water-bearing Strata, 100 *feet.*—Beds of this description might be penetrated safely by using the François Cementation Process, whereby the soft beds are cemented in advance of the sinking operations. In this way the flow of water into the shaft is arrested, and the sides are secured by permanent lining from a solid foundation, the lining being built up to the base of the underhanging tubbing. For details of this method of sinking, see Figs. 25 and 26.

Description of a Pneumatic Stowing-Machine

5. Describe a pneumatic gob-stowing machine. Under what conditions would you apply it ? What kind of dirt would you feed into it ? (40)

A. Fig. 34 is a diagram illustrating the *Meco pneumatic stowing-machine.* The stowage material is introduced at the top part of the machine and it falls into the path of compressed-air jets at a pressure of 75 lb. per sq. inch from pipes 2 inches in diameter. The material is in this way blown with considerable force from the

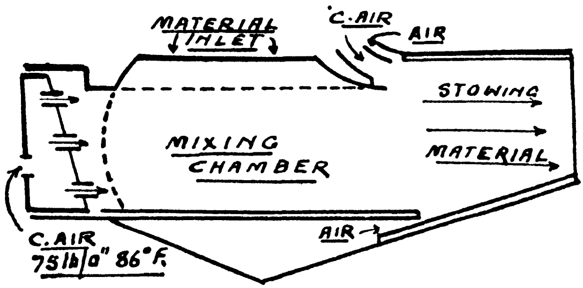

FIG. 34.—Diagrammatic Section of Meco Pneumatic Stowing-machine.

outlet end of the machine and penetrates the goaf. Fig. 35 shows the attachment of the jet. The machine is clamped to the troughs of a shaker conveyor, so that the material can be raised to the inlet at the top of the stower by a short length of trough. The jet with its picking-up tray can be unclamped and allowed to slide down the troughs for a yard or so at a time. There are no moving parts in the stower and it is foolproof in operation.

Conditions of Application.—The pneumatic stower might be applied where great difficulty exists in working safely by the longwall method, e.g. in conditions such as the following: Thick and strong rock roof which breaks off very heavily and causes roof breaks at the face every 3 to 4 yards with 15 inches or more of roof subsidence; face supports are liable to be displaced, falls occur, the output cannot be maintained, and dangerous face conditions exist.

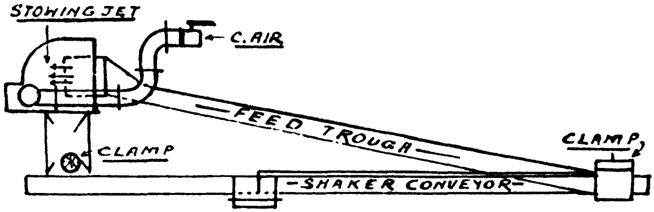

FIG. 35.—Attachment of Meco Stowing-machine Jet.

Under the above conditions, extra supports might bring about no improvement in working. Extra ripping and stowing of the usual kind might not be possible or might prove too costly for application. Pneumatic stowing has proved successful in these circumstances by keeping an unbroken roof, giving regular output, limiting roof subsidence, making safer workings, and allowing machinery to be applied at the face with greater safety. The coal is also easier to get.

Stowing Material.—Small material must be used for stowing the waste. Dry refuse is not suitable owing to the difficulty of getting it into position. Well-drained washery dirt is the most suitable for packing, as it binds well and keeps the face free from dust. It forms a hard wall in the goaf with an angle of repose approaching 90 degrees. The maximum subsidence after 6 months does not exceed 2 inches. A suitable siding must be arranged near the face to hold at least 100 tubs of dirt. A tippler of the rotary type, worked by compressed air, would be required to load the material into a gate-belt conveyor. The latter should be designed to feed the shaker pans and the pneumatic stowing-machine.

Research Work and Shot-firing

6. "Shot-firing has been made safer and more efficient by scientific research." Discuss this statement; also give your views on low-density explosives. (40)

A. The above statement, in my opinion, is perfectly correct. A large amount of research work has been carried through in recent years, and the mining industry has derived great benefit as a result. The following are the lines along which the research work has proceeded :—

(a) Explosive compounds have been thoroughly investigated and tested before being placed on a permitted list. Flame analyses have been carried out, and the products of combustion from explosives have been investigated.

(b) The dangers attending the use of small-diameter cartridges have been demonstrated. The proper contact of cartridges for good results has likewise received due attention.

(c) The correct position of the detonator in the charge for safety and economy has been demonstrated, and the advantages of using proper size detonators have been clearly shown.

(d) Safety shields for detonators have been designed. Sheathing material containing sodium bicarbonate has been applied to cartridges to reduce the risk of explosions of firedamp.

(e) The great reliability of low-tension detonators has been demonstrated.

(f) Explosive mixtures have been ignited by the compression caused by firing an explosive.

(g) The great advantages and the safety resulting from proper stemming have been fully demonstrated. The safety obtained by the use of sand stemming has received much attention, and valuable suggestions have been made as to method of application.

(h) Electrical exploders have been reconstructed so as to prevent ignitions of firedamp.

(i) The choice of explosive compounds for the work to be accomplished has been made less difficult.

Low-density Explosives.—Normal explosives have a specific gravity of about 1·4 while low-density explosives have a specific gravity of 0·6 to 0·7. The latter are produced from a low-density cellulose, such as the sawdust from special woods having a density of 0·4, or from other low-density carbonaceous materials. Low-density explosives more nearly approach gunpowder in their action, and have proved successful for friable coals and in seams where the distance from the undercut to the shot-hole is short or limited.

Putting down a Borehole from an Underground Road

7. Describe fully how you would put down a borehole to a depth of 35 yards from an underground roadway whose normal dimensions are 12 feet wide by 6 feet high. Give details of the various tools required. (40)

A. Fig. 36 shows the appliances and the arrangement required for putting a borehole down to a depth of 35 yards. The heading should be increased in height to, say, 8 feet for a distance of 16 feet to allow for the working of the bore lever. It should be well supported, and a bore pit 4 feet deep should be put down at a distance of about 5 yards from the face so as to facilitate the changing of the rods and the cleaning out of the hole.

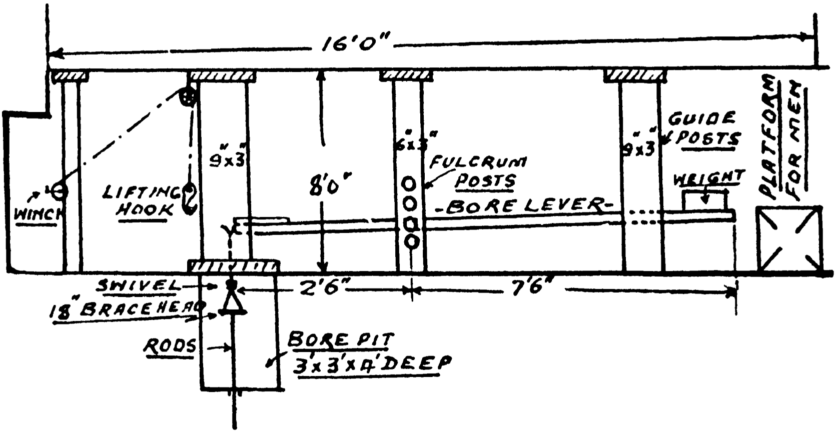

FIG. 36.—Boring-down from an Underground Road.

The first 6 to 9 feet might be bored by means of a ratchet drill, or power drill, after which the bracehead would be used for operating the rods directly by manual labour, until a depth of 60 to 80 feet is reached. After reaching this depth it might be necessary to fit up a bore lever, as shown in the sketch, to allow of the rods being operated. The rope or chain connecting the beam to the bracehead should contain a swivel to allow of the turning of the rods by the bracehead after each stroke. A light winch should be installed near the face to allow of the rods being lifted and unscrewed, and for cleaning out the hole. The men would lift the rods by stepping from the platform provided to the end of the lever; the weight of the rods would give the necessary stroke when the men stepped back again to the platform.

Boring Tools Required.—Bore rods 1 inch square and 6 feet long with adjusting pieces 12, 18 and 36 inches long; chisels for cutting at the end of the rods; hooks, keys and forks for unscrewing the rods; holding keys or clamps; sludger for cleaning out the hole and for examination of the cuttings.

NOVEMBER 1934 EXAMINATION

Layout of Pillar-and-Stall Workings for 600 Tons per Shift

1. Under what conditions would you decide to work a seam on the pillar-and-stall system ? Sketch a general layout to give 600 tons per shift, showing the roads and pillars, and explain how the pillars are extracted. (50)

A. *Conditions for Pillar-and-Stall Working :—*

(*a*) For thick seams of coal, 6 feet or more in thickness, with no dirt partings for stowing the waste workings and making roadside packs ; also where the roof and pavement strata are unsuitable for making packs.

(*b*) For soft seams of coal which are easily split up into pillars and worked profitably by hand without the use of machinery.

(*c*) In seams underlying surface property, and where it is necessary to leave a portion of the coal unworked.

(*d*) When the area to be worked has a bad roof with numerous faults and rolls, and which gives off firedamp readily as blowers with water.

(*e*) Seams with soft, irregular and heaving pavements are sometimes worked pillar-and-stall in preference to longwall.

(*f*) When the thickness of overlying strata is small and contains soft beds at the surface, the pillar-and-stall method is sometimes preferred.

Fig. 37 shows a general layout of pillar-and-stall workings by the panel method. The panels to the rise are 12 pillars in width and go to the boundary. Stooping is following up the whole workings in one of the panels. The chief dimensions are included in the sketch. The dip side coal is worked by cross-cuts driven at an angle of 45 degrees to the main roads, pillars being formed as in the previous workings by driving bords and walls. The bords are driven face on to the cleavages and are 15 feet wide. The walls are driven 9 feet wide, end on to the cleavages.

Output per Shift :—

Panel A.	8 wide bords,	2 men,	12 tons	= 96 tons	284 tons
	8 narrow walls,	1 man,	6 ,,	= 48 ,,	
	10 wide lifts,	2 men,	14 ,,	= 140 ,,	
Panel B.	8 wide bords,	2 men,	12 ,,	= 96 ,,	144 ,,
	8 narrow walls,	1 man,	6 ,,	= 48 ,,	
Cross-cut C.	10 wide bords,	2 men,	12 ,,	= 120 ,,	180 ,,
	10 narrow walls,	1 man,	6 ,,	= 60 ,,	
			Total output per shift	= 608 ,,	

The pillars are extracted by driving juds or lifts, 5 yards wide, to the rise and dip from the narrow walls, as shown by the enlarged plan view of pillars in Fig. 37.

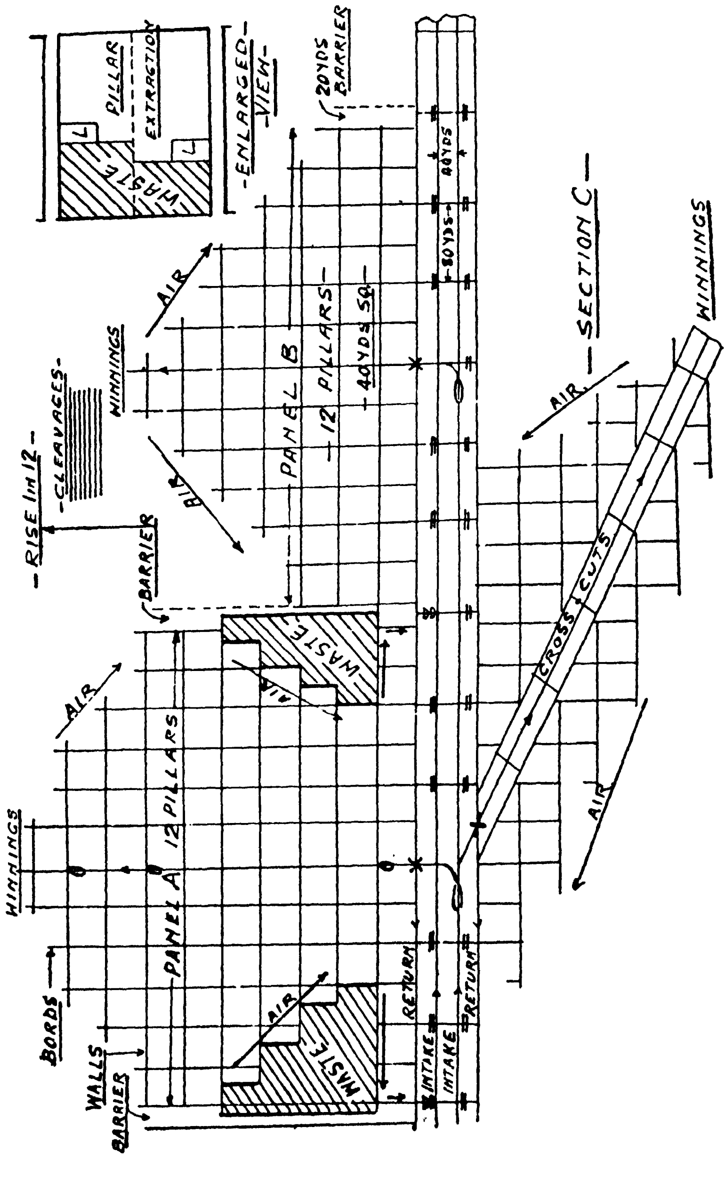

FIG. 37.—Plan of Layout of Pillar-and-Stall Workings.

Aqueous, Igneous and Metamorphic Rocks—Stratification and Cleavage

2. Describe aqueous, igneous, and metamorphic rocks. In which rocks do you find stratification ? What is meant by the term "Cleavage" ? (40)

A. *Aqueous rocks,* sometimes termed *sedimentary rocks,* are formed by the agency of water into regular beds or strata. They resemble accumulations of sand, mud and pebbles, which have been moved in a mechanical way by the action of water. Their formation has been brought about by the disintegration of older igneous rocks. Aqueous rocks are found on low spreading plains and on high mountain ranges, and they are often tilted, warped, bent and broken. They all contain fossil remains and are arranged into groups according to mineral character and the fossils found in them, such as mechanically-formed sandstone and shale, chemically-formed limestone and flint, and organically-formed chalk and coal.

Igneous rocks resemble material which has been poured out by volcanoes, or which has been injected into fissures in the earth's crust by volcanic action. Their formation has been brought about by the action of heat. Owing to their mode of formation they contain mineral crystals, and their composition and properties are best studied by slides under the microscope. Igneous rocks were at one time in a molten state, and they have since cooled and hardened, either near the surface or deep down in the crust. They are classified as: (*a*) Plutonic, or deep origin rocks, like granite; (*b*) Intrusive, or shallow origin rocks, like diabases and felsites; (*c*) Volcanic, or glassy rocks, like basalt and andesite; (*d*) Fragmentary or surface deposits, such as pumice, obsidian and pitchstone.

Metamorphic rocks are those aqueous and igneous rocks which have been altered greatly, deep down in the crust, by heat and pressure. They are, as a rule, distinctly crystalline. Some of them shade off into aqueous shale and sandstone, while others agree in chemical composition with the igneous rocks into which they might be traced. In other cases they represent original crystalline eruptive rock which has been subjected to such enormous pressure and shearing action that foliated structures, or re-crystallised minerals, have been formed. Metamorphic rocks are classified as: (*a*) Rocks not altered beyond recognition, such as quartzite, clay slate, marble, graphite; (*b*) Rocks entirely altered beyond recognition, such as gneiss, schists and foliated rocks; (*c*) Rocks or minerals changed entirely in substance, such as the garnets.

Stratification is found mostly in the aqueous rocks; such rocks generally split into layers along definite lines to form beds or strata varying in thickness from an inch or less to many feet. The lines along which separation takes place are known as lines of stratification.

Cleavage is common in metamorphic rocks, a good example being roofing slates. Under the influence of great pressure, the particles of which a rock is composed tend to rearrange themselves along the line of least resistance, thereby imparting to the rock a fissile structure known as cleavage. Cleavage is best developed in altered shales and clays. When a rock is cleaved, it loses its power of splitting along lines of stratification or bedding.

Working under an Extensive Building Area—Breaks and Subsidence

3. A 6-foot seam at a depth of 500 yards is to be worked by longwall under an extensive building area. What would you do to minimise the amount of subsidence? What kind of ground near the surface would you expect to cause (*a*) serious localised breaks; and (*b*) a more even subsidence extending over a much larger area? (40)

A. To minimise subsidence under the above conditions the following precautions should be taken:—

(*a*) The face should be worked forward in a regular manner and in a straight line by having regular cutting and stripping.

(*b*) The workings should be well supported by strong steel props and crowns at the face, and by a good system of stowing the waste.

(*c*) Strong and large roadside buildings should be put in, and the waste should be completely stowed if possible; failing this, sufficient roads and packs should be arranged along the line of face.

(*d*) Shot-firing should be ruled out, if possible, to prevent roof fractures. Pneumatic or hydraulic stowing of the waste would be beneficial.

The above arrangements should prevent roof fractures at the face and also allow the area to settle down in regular fashion on the packed waste.

Serious local breaks might be induced at the surface if the worked area is insufficiently stowed, and if the strata at the surface are of a firm nature. Faults in the area would also produce serious breaks.

An even subsidence over a wide area would result from bad packing underground and from the surface beds consisting of sand, mud, drift, etc.

Position of Pump House and Water Lodge Room—Construction of Water Lodge Room

4. Near the shaft bottom, pumps have to be installed to deal with 150 gallons per minute. Show the relative positions of the pump house and lodge room, and give a cross-section of the latter. The seam is 4 feet thick, dipping 1 in 3. There is a hard roof, and under the coal 2 feet of soft shale, then a hard bed. (40)

A. Fig. 38 shows an arrangement of pump house and lodge room suitable for the above conditions. The pump house near the

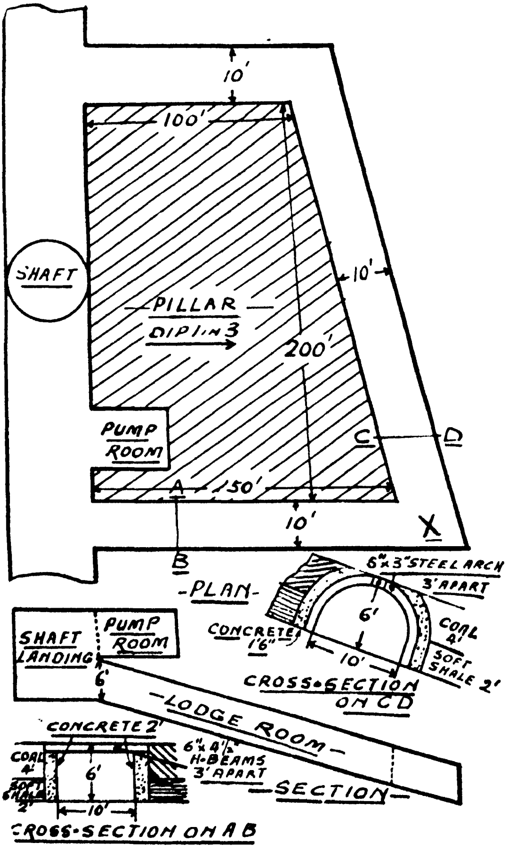

Fig. 38.—Arrangement of Pump House and Water Lodge Room.

shaft landing is 6 feet above floor level, so that in the event of breakage or stoppage of the pump, repairs can be carried through before flooding takes place. The lodge room is constructed in the shaft pillar to the dip side of the shaft, as shown in the diagram. Cross-sections of the lodge room with dimensions are included in the sketch.

The 2 feet of soft shale should be removed in the lodge room so as to obtain a hard roof and a hard floor. The part to the full dip is supported by H-beams and concrete. The lodge room on the strike is supported by steel arches and reinforced concrete, to resist side pressure from the beds inclined at 1 in 3. The lodge room has a capacity sufficient to allow the pump to stand for 14 hours before the water rises to the level of the shaft landing. The connecting road, 200 feet long, is arranged to dip at a low grade towards the point *X*, so that the sump may be cleaned out from the lodgment near the pump room.

JULY 1940 EXAMINATION

(Six questions only to be answered.)

Working a Banded Seam of Coal by Machinery

1. A level seam of coal 4 feet thick is divided in the middle by a 4-inch band of dirt. The two portions of the seam are of different quality, and it is desired to keep them separate. Describe with a sketch the method of working you would adopt on a longwall machine-cut conveyor face to achieve this object. (50)

A. At first sight the given conditions appear to be suitable for the application of bottom-loading belt conveyors, so arranged as to allow the bottom part of the belt to deal with the coal from the lower part of the seam, and the return part of the belt to deal with the coal from the top part of the seam. This method of underground coal conveying on faces has been carried out in actual practice in England. However, when several important factors are taken into consideration, such as stoppages at the face by breakage, the increased number of trunks required, and the increased number of loading stations, a simpler and more reliable method of conveying the coal is recommended, as follows :—

The 4-inch dirt band referred to could be cut to a depth of 4 feet 6 inches, in one shift, by utilising a chain coal-cutter of the *Anderson Boyes* 15-*inch type* of 50 B.H.P., using alternating-current electricity and arranged with an overcutting gearhead.

Such a machine has a good reserve of power and a range of cutting-chain speeds to suit hard, medium, or soft holings. It is 8 feet 6 inches long by 3 feet 2 inches wide by 2 feet 3 inches high. In view of the fact that a large amount of cuttings have to be moved over the conveyor, in addition to handling an output of 450 tons per shift, two machines might be installed on the face of 250 yards in length. There would be the valuable safeguard that if one

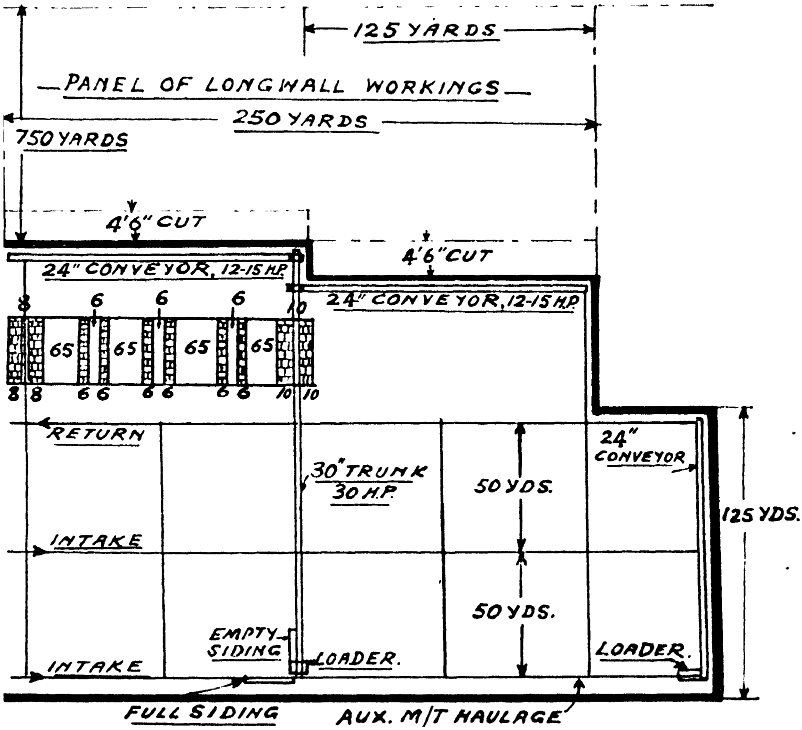

FIG. 39.—Plan of Longwall Workings for Machine Mining.

machine breaks down, the whole face could be cut with the sound machine until the other machine is repaired. In this way the output of coal per shift would be maintained.

Bottom-loading belts, 24 inches wide, could be used on the double-unit face, and these belts would deliver the coal directly into a trunk belt, 30 inches wide, installed in the centre road. Fig. 39 is a sketch plan showing the general layout of the workings in panels 250 yards wide by 750 yards long. The central, or trunk, road should be at least 10 feet wide by 8 feet high, and it should be supported by arch girders 5 inches by 4½ inches with 3-foot centres. The double-unit face is shown staggered by 4 feet 6 inches, the depth of the undercut. If it is desired to have a straight face throughout for a 24-hour cycle of operations, the heads of the face conveyors could be kept a short distance back from the gate conveyor. This would keep the respective bows of the two conveyors clear of each other, the thickness of the seam under consideration being sufficient for this purpose. The discharge ends of the face conveyors should be fitted with loading shutes to discharge the coal to the gate conveyor. The coal discharged in this manner could be properly controlled by ploughs shod with pieces of old belting, to give gentle loading conditions and equal delivery space on the gate conveyor for the coals from the respective face belts. The details of roads and packs constructed for the haulage of the coals and for ventilation are shown in Fig. 39, the sizes being marked in feet. There are three escape roads leading from each face.

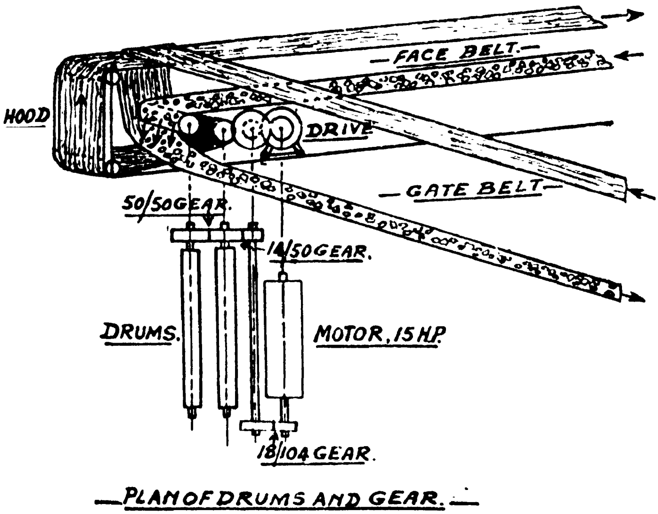

FIG. 40.—Diagram of Delivery Head of Bottom-loading Belts.

Fig. 40 shows the arrangement of the delivery head of a bottom-loading belt conveyor. The coals on the lower part of the face belt pass to the gate belt, or delivery shute, before the lower belt passes to the underside of the delivery drum. The gate belt runs parallel with the main gate, and it passes within the hood or bow which conveys the return belt to the face. The sketch shows also the position of the driving drums and motor for operating the conveyor.

The output of coal from a double-unit face would be $\frac{4 \text{ ft.} \times 4\frac{1}{2} \text{ ft.} \times 750 \text{ ft.}}{30 \text{ cu. ft. per ton}}$ or 450 tons per shift. This output could be dealt with by a modern trunk conveyor in a single shift. The top coal should be stripped off first, during the first half of the

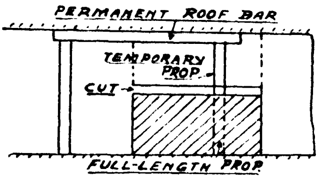

FIG. 41.—Stripping Coal and Supporting the Face.

filling shift, thus allowing of the permanent roof bars being erected and supported as shown in Fig. 41. This would tend to make the working safer, as there would be no overhanging coal requiring temporary supports. The bottom coal could be stripped during the second half of the filling shift, full-length props being set under the bar as soon as the coal is removed.

Diamond Method of Boring

2. Sketch and describe the machinery and appliances used for boring through coal-measures strata to a depth of 600 yards by the diamond method. (40)

A. In describing the diamond method of boring, special attention should be given to the two important parts of the plant required for carrying out the boring operation, viz., the surface plant for rotating the rods and flushing them continuously with water, and the boring crown in the borehole with all its auxiliary attachments for making the hole and for extracting a solid core.

Fig. 42 shows end and side elevations of the mechanism required at the surface by which the hollow boring-rods are given a rapid rotary motion transmitted from an engine through bevel wheels at the entrance to the borehole. Water from a force pump operated by the surface mechanism is forced, at the rate of 20 to 30 gallons per minute, down the hollow rods to keep the boring crown cool and to wash away the cuttings to the sediment tube for surface examination afterwards.

The end elevation shows the belt-driven wheel and bevels, incorporated in a strong frame to which suitable wheels are fixed for the purpose of removing the frame from the borehole position when cores are being extracted. The water union and ball race are used to make a joint between the fixed and rotating rods. A sliding key is used to make the rods revolve with the top bevel wheel and at the same time allow them to move downward as boring proceeds. The side elevation shows shear legs, boring frame, rope drum, belt pulley and counterbalancing arrangement. For shallow boring the rods are weighted, and as the depth increases these weights are replaced by a counterbalance. An engine or motor of about 10 H.P. would be necessary to do the work required, and it is fitted with suitable gearing for driving the rope drum and belt pulley.

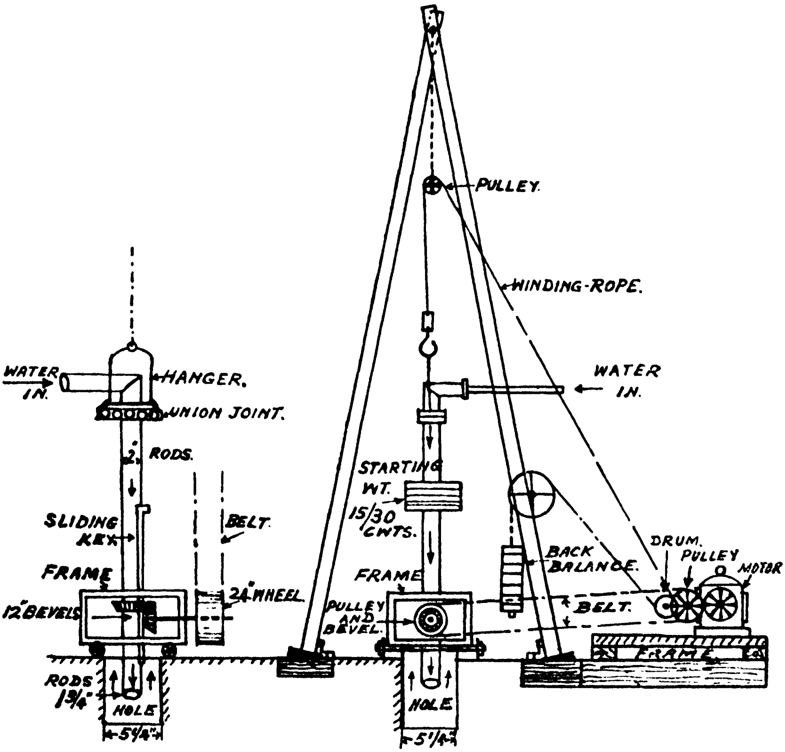

FIG. 42.—End and Side Elevations of Diamond Boring Plant.

Diamond Boring Drill.—Fig. 43 is a section of the diamond boring drill, showing the boring crown with horizontal and vertical water passages, the core trap and clip for wedging and breaking off the core when the rods are lifted, and the core tube 20 feet long with the sediment tube 3 feet long. The black diamonds, fixed in the crown, are set in it to cut an annular ring or passage in the strata, thus forming a solid core inside of the boring tool. When the rods are lifted from the hole by the engine, the core clip becomes wedged between the core and the slightly tapered core trap ; the core is thus broken off and carried with the boring tool to the surface.

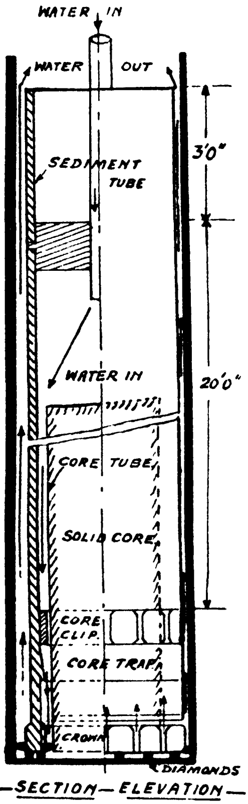

Fig. 43.—Diamond Borer.

To put a hole down to a depth of 600 yards by the diamond method of boring, it would be necessary to tube it as boring proceeds, and when this is done in successive stages each stage reduces the size of the core obtained. If the hole is 6 inches in diameter at the top it might be reduced to about 2 inches at a depth of 600 yards.

Tank for Winding Water from a Sinking Pit

3. Water is met with in a sinking pit and it is decided to wind it in tanks. Describe, with sketch, the kind of tank you would use for getting out the water quickly, and without manual labour in filling. State what provision you would make to empty the tank expeditiously at the surface. (40)

A. Fig. 44 shows in section and elevation the arrangement of the *Galloway pneumatic water-barrel* for winding water from a sinking pit. It consists of a closed cylinder 4 feet in diameter and 7 feet deep, containing water-gauge, vacuum pipe, and a leather-faced inlet valve 18 inches in diameter. A pipe 3 inches in diameter extends from a receiver at the surface down the side of the shaft, and at the end of this pipe about 40 feet of hose is

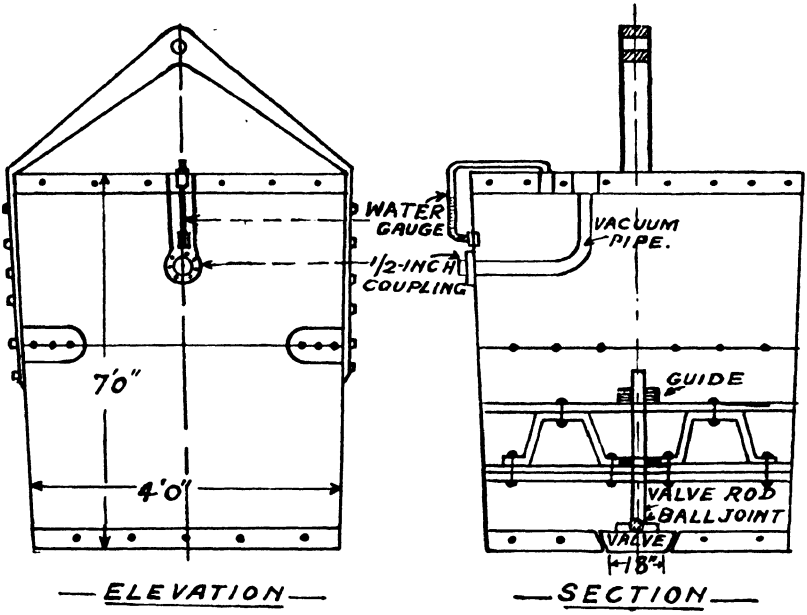

FIG. 44.—Galloway Pneumatic Water-Barrel.

connected, the free end of the hose being fitted with half of an instantaneous coupling and a stopcock. A steam blower or air-pump at the surface is used to produce a vacuum in the receiver. When the barrel comes to rest at the bottom of the shaft, the hose is connected to the vacuum pipe and a vacuum is produced inside the barrel. Water under atmospheric pressure thus lifts the inlet valve in the base of the tank and fills the tank with water to within an inch of the top. The stopcock is then closed and the hose detached, leaving the water in the tank under atmospheric pressure and ready for discharge. The barrel is finally raised to the surface by the winding-engine. Such a barrel has a capacity of about 550 gallons, and might make 30 trips per hour to a depth of 250 yards, thus dealing with 16,000 gallons of water per hour.

Fig. 45 shows how the barrel is emptied at the surface by means of a water trolley. When the barrel reaches the surface it is lowered into a specially constructed trolley which contains a projection for lifting the valve and allowing the water to run away. With this arrangement the barrel is emptied at the surface at about the same rate at which it is filled in the sinking pit.

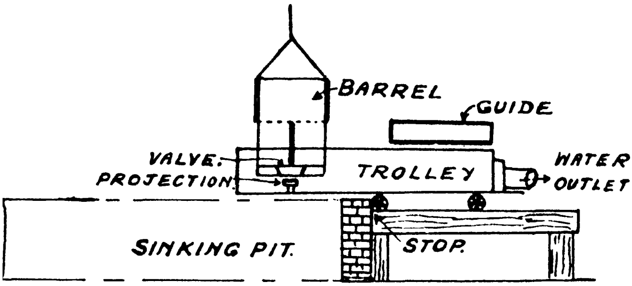

FIG. 45.—Trolley Arrangement to empty Water-Barrel.

Rock Formations

4. Describe aqueous, metamorphic, arenaceous and calcareous rocks. In which rocks do you find stratification ? (40)

A. Aqueous rocks, metamorphic rocks and stratification have been described in the answers to previous questions (see pages 56 and 57).

Arenaceous rocks are those composed mostly of sand, grit, pebbles and small pieces of other rocks. They belong to the mechanically-formed type of aqueous rock. The materials of which they are composed have been obtained by the grinding down and destruction of older rocks by agents of denudation. Through the action of water, these materials have been deposited into beds and cemented together by solutions of silica and iron. Examples of such " rocks " are silt, sandstone in layers, sandstone masses, grits, conglomerates, flints and deep-sea oozes.

Calcareous rocks are those composed mostly of carbonate of lime. They have been formed by plants and shells extracting lime from deep water during their growth or formation, and then forming deposits when dying off. They belong to the organically and chemically formed types of aqueous rock. Examples are chalk, Carboniferous or mountain limestone, oölitic, encrinital and magnesian limestones.

Roadhead Supports

5. Describe, with sketches, the method of supporting the roof at roadheads with which you are acquainted. By roadhead is meant the area around the junction of the gate road with the face. (40)

A. Fig. 46 shows the method of supporting the roof strata at roadheads as practised in the Crawfordstone seam at Mossblown

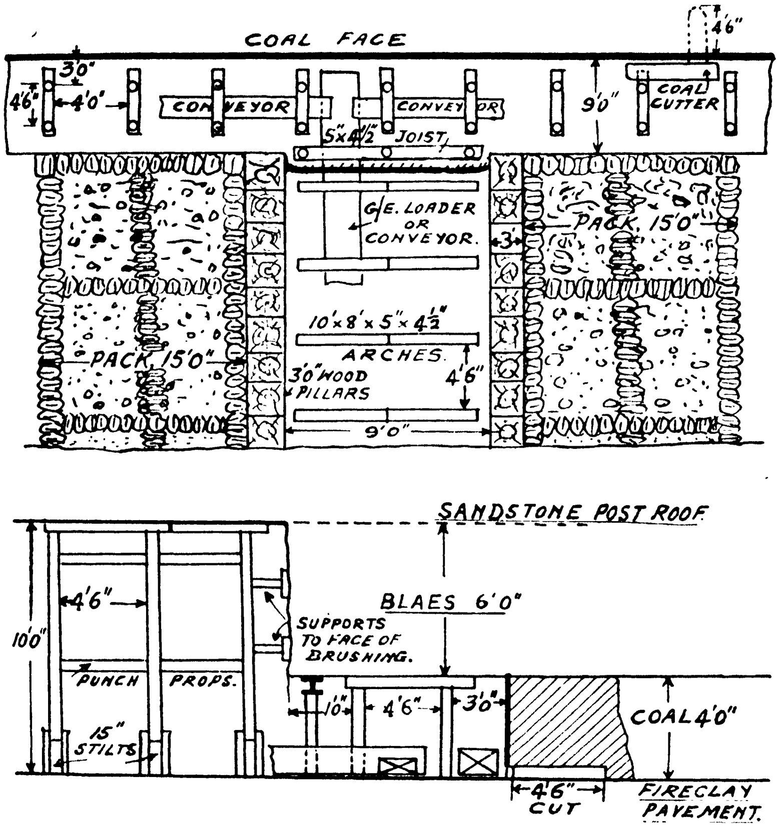

FIG. 46.—Plan and Section showing Method of supporting the Roof Strata at Roadheads.

Colliery, Ayrshire. The coal seam is 4 feet thick and the roadside buildings are 15 feet wide, made of material obtained by brushing the blaes. The brushing face is supported on the underside by a steel joist 5 inches by 4½ inches, and along the face by wood struts and runners. All details of supports are included in the sketch.

Building of Packs

6. What are the essential features of a good pack, and what do you consider to be the best method of building it? (40)

A. The following are the essential features of a good pack. Packs required for the support of the roof in roads of longwall workings should be carefully constructed to give good results. They should be of sufficient size, not less than 6 feet in width, for the width of road and height of seam under consideration. Strength is an important factor, and strong stones of a good size from the roadway ripping should be used in forming the outside and interior walls. The latter, about 4 feet to 6 feet apart, give increased strength to the pack and therefore tend to reduce roof-movement. All interior spaces between walls should be rammed tight with loose material from the ripping or goaf, so as to fulfil the requirements of strength already referred to. The careful building of

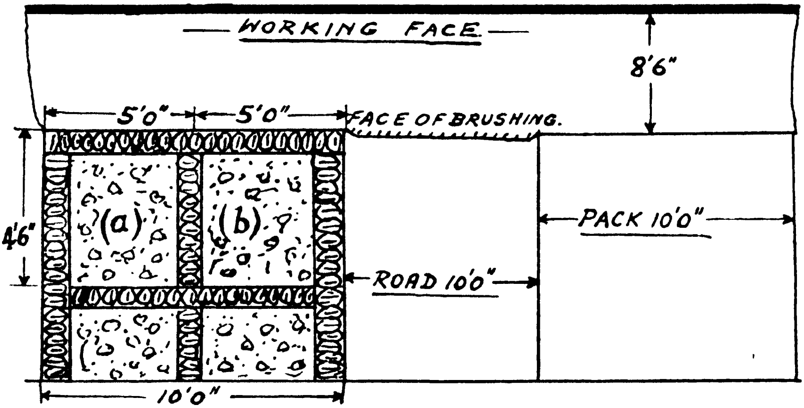

FIG. 47.—Plan showing Method of Building Pack.

packs, to make them stable and tight with the roof and pavement, is important to prevent collapse and bursting out under excessive roof pressure. It is likewise essential that packs should be built on a good solid foundation and as near to the working face as circumstances will permit. Packs on the face, where caving or semi-caving takes place, are built from the material in the waste or goaf.

In building a pack, the following points have been referred to by investigators on longwall working, viz. :—

(i) The boundary of each pack should be marked with chalk on the roof by an official.

(ii) After testing the roof, the floor is properly cleaned up to ensure a good solid foundation.

(iii) Fig. 47 is a plan showing a pack in the course of construction, the part (*a*) 5 feet by 4 feet 6 inches being constructed first, followed by part (*b*) to complete the extension required.

(iv) A layer of stones is placed on the floor to form walls, the stones being as long as possible and laid to run into, rather than parallel with, the walls.

(v) The interior is then filled with small material which is spread level and bedded well amongst the wall stones. The front wall should be built so that it can be continued into an adjoining section without leaving a vertical joint.

(vi) Layer by layer this procedure is followed until the pack reaches the roof. The interior is then rammed tightly to the roof with pick or rammer. The walls are also finished off tight with the roof. The outer walls of the pack should slope in towards the top about one inch per foot of height.

(vii) The roof is tested in the part (*b*) space shown in the diagram, and the roof supports in this area are withdrawn.

(viii) The second part of the pack is then built up in a similar way, care being taken to use good long stones in building the walls so as to make them stable and less liable to burst out under pressure.

(ix) The exterior walls of the pack, facing the roadway, should be built so that the stones do not project into the roadway.

Switchgear required for Underground Electric-driven Plant

7. In a double-unit conveyor district the following electrically-driven units are in use, the system being 550 volts A.C.; longwall coal-cutter, coal-drilling machine, face-belt conveyors and gate-belt conveyor. The loading point is lit electrically. Enumerate the switchgear you would require to control these units and state where you would place it. (40)

A. Fig. 48 is a plan showing the positions of the switches referred to in the above question. Where the district cable branches off from the main cable, say at the point *A*, there should be a panel containing the district isolating switch, or master switch, of the triple-pole air-break type in a flameproof casing.

At the loading point, where electric light is used, there should be a box or panel *B* containing a double-pole switch of the air-break type and an oil-immersed transformer to reduce the voltage to 125 volts. The lighting main from the transformer would be fitted with double-pole switches and fuses for operating the lights, all in a flameproof casing. The whole of the apparatus would be earthed to a common earth bar.

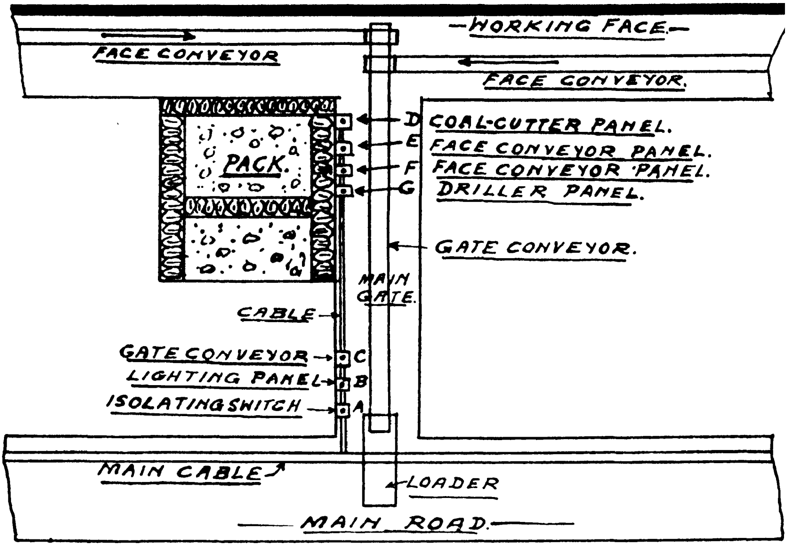

FIG. 48.—Plan showing Positions of Switchgear.

The gate-belt conveyor motor might be placed quite near to the loading-point and a box or panel would be required, say at the point *C*, for this motor. The box would contain an air-break flameproof three-pole automatic circuit-breaker, fitted with a no-voltage release coil and three overload trip coils with time-lag device. A system of earth-circuit protection should likewise be included in the box to ensure that the motor is not operated unless properly earthed. At the motor a suitable starting switch of the flameproof and enclosed type would be required, say a direct-line starting switch for the squirrel-cage motor generally used for this class of work.

Similar boxes or panels to the above, say at the positions marked *D*, *E* and *F*, not less than 20 yards from the face, would be required for the coal-cutting machine motor and for the two motors of the face conveyors. Starting switches of the type already referred to would be necessary for controlling the respective motors. The trailing cable for the coal-cutter would be of the C.T.S. type with copper braiding, having three current cores, one pilot core, and an earth conductor. At each end of the cable should be fitted properly constructed B.S.I. plugs, so that they could not be fixed or withdrawn with the switch in the "on" position.

For drilling purposes a box or panel, convenient to the face, say at the point *G*, would be required, and it would be similar to that already explained for the electric light at the loading point. The voltage should be reduced to 125 volts, and the leads to the drilling machine should terminate in an enclosed switch for operating the drilling motor.

JULY 1941 EXAMINATION

(*Six questions only to be answered.*)

Working a Level Seam of Coal by Machinery

1. COMPULSORY QUESTION.—A level seam, 2 ft. 6 in. thick, is being worked by two conveyors delivering direct into tubs on one main gateway. Each conveyor is 100 yards long. The depth from the surface is 600 feet, the roof is of shale or blaes and the floor of dry fireclay. The seam is not liable to spontaneous combustion and is not very gassy. Show, with sketches, the general layout of these faces. Give the finished dimensions of roadways and the thickness and the number of rippings generally necessary. The coal is machine-cut to a depth of 4 feet 6 inches. Give the approximate daily output to be expected and describe how you would arrange the shifts of the men to carry out the various operations necessary. (50)

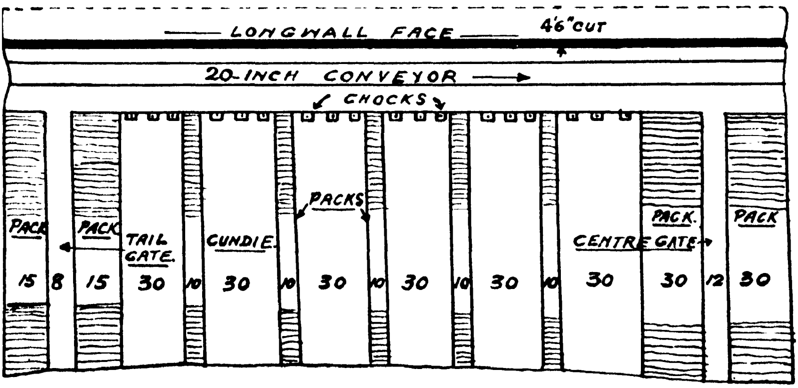

FIG. 49.—Plan of Longwall Face, showing Roads and Packs.

A. Fig. 49 is a plan showing the arrangement and dimensions of the various roads required on a face 100 yards long. The centre gate is 12 feet wide by $8\frac{1}{2}$ feet high and has roadside packs 30 feet wide. The tail gate is 8 feet wide by 7 feet high and has roadside packs 15 feet wide. The semi-caving system of roof control might be applied with main and tail gates only. The intervening space should be supported by 10-foot packs erected from fallen waste material, leaving 30-foot-wide cundies supported behind the conveyor by chocks of hard wood $2\frac{1}{2}'' \times 5'' \times 5''$, fitted with releases. The output per shift from the double-unit face would be

$$\frac{2\frac{1}{2}\text{ ft.} \times 4\frac{1}{2}\text{ ft.} \times 600\text{ ft.}}{30\text{ cu. ft. per ton}} \text{ or } 225 \text{ tons.}$$

Loading direct into tubs at the face of the centre gate of a double-unit system might be carried out by using two top-loading belts

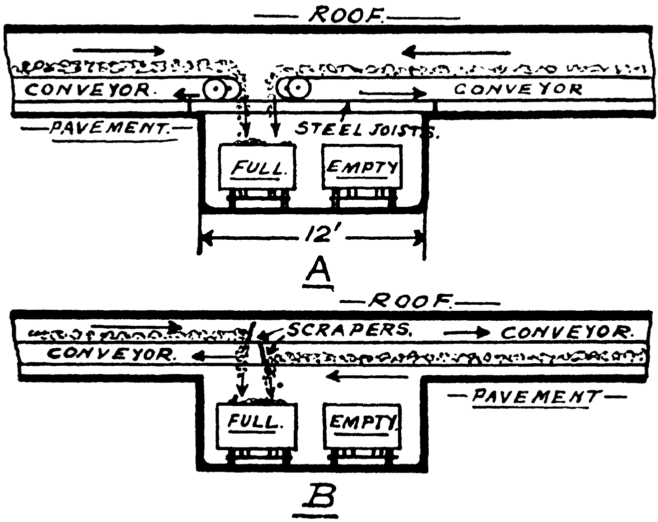

FIG. 50.—Sections showing Coal-filling Arrangements on Centre Road of Longwall Workings.

discharging into a common tub, as shown by Fig. 50A. Alternatively, use might be made of one bottom-loading belt running the complete length of the double-unit face, one side of the face loading coal on to the underside of the belt, and the other half of the face loading on to the top side of the belt, the coal being removed from the belts by scrapers and delivered to a common tub as shown by Fig. 50B. Either of these installations would require a road of about 12 feet wide with 4 feet of pavement ripping carried right up to the face.

Fig. 51 is a plan of the main or centre gate, showing the details of the arrangement suggested in Fig. 50A. The total rippings are 4 feet from the pavement and 2 feet from the roof over a width of

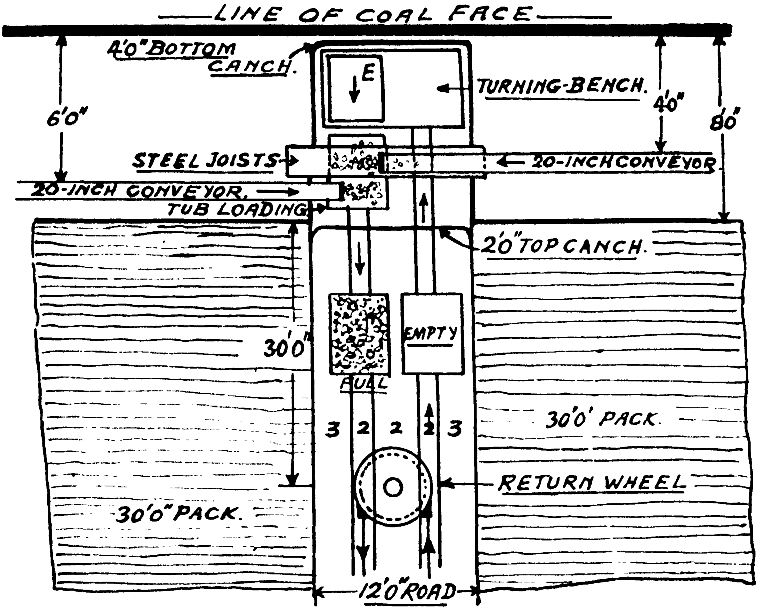

FIG. 51.—Plan showing Coal-filling Arrangements on Centre Road of Longwall Workings.

12 feet, thus giving sufficient stowage material to build roadside packs 30 feet wide. The tubs at the roadhead are moved on plates or on a turning-bench 12 feet by 8 feet, moved forward with the face. The gearhead of one conveyor of the top-loading type could be carried on two steel joists across the roadway to suit the loading arrangement suggested. The return wheel of the auxiliary endless-rope haulage is situated 30 feet from the plates and it is moved forward weekly. This return wheel is 4 feet in diameter and is enclosed in a steel plate casing having rope gaps for the passage of the rope. It is fixed on the pavement and clamped in position, and rails are laid on top of it for the passage of tubs. Useful dimensions are shown in feet on the diagram.

The cycle of operations, over a 24-hour period, in connection with the above methods, is as follows : The coal should be stripped off the face to the depth of the undercut during the first shift of the day, and supports should be set up immediately there is room for them. During the second, or afternoon shift of the day, the coal conveyors should be moved forward to a suitable position for the next stripping shift. The ripping required from roof and pavement should be carried out during this shift, and the packs should be extended forward by $4\frac{1}{2}$ feet. Later on, chocks and props should be withdrawn near the goaf edge. The coal-cutting machine should be turned, flitted and picked ready for operations. In the night shift, or final shift of the 24-hour cycle, shot-holes should be bored at specified distances along the face, say 10 to 15 feet, by using an electric borer. Following up this operation at a safe distance, a chain coal-cutting machine of the Anderson Boyes 15-inch type could cut the entire length of the face, in the fireclay, to a depth of 4 feet 6 inches in one shift. Any further timbering, chock setting and packing required should be completed during this shift. Finally the plates or bench should be placed in position ready for the coal-getting shift which follows on again next day. The workmen required in the various shifts to carry out the above operations might be as follows :—

First Shift.		*Second Shift.*		*Third Shift.*	
Stripping and filling Coal	16	Stonemen	10	Coal cutting	4
Shot-firing	2	Moving Conveyors	4	Cleaners	2
Material	2	Timbering	2	Borers	2
Loading bench	2	Deputy	1	Clearing tracks	2
Greaser	1			Moving bench	2
Deputy	1			Deputy	1
Total	24	Total	17	Total	13

Details of Roof Supports on a Longwall Face

2. Describe in detail, with sketches, the method of roof supports on the faces referred to in Question 1. Give the distances apart of such supports. (40)

A. The method of supporting the roof at the longwall face where two top-loading belts are installed is shown in Fig. 52 in plan and sections. Corrugated steel crowns 6 feet long by $5\frac{1}{2}$ inches wide by $\frac{5}{8}$ inch thick are set in systematic fashion along the face at a distance of 3 feet 6 inches apart. The props used at the face for the given conditions are of the composite tubular mild steel type, $4\frac{1}{4}$ inches diameter and $\frac{1}{4}$ inch thick, with soft core and hardwood ends, the latter projecting $2\frac{1}{2}$ inches from the ends of the tube. These props should be 2 feet 10 inches long with a yield strength of about 12 tons.

The section on the line *AB* shows the position of the supports immediately before the coal stripping starts on the left-hand side of the face. The face props are set 3 feet from the coal, thus leaving ample space for the coal-cutter to operate, the end of the

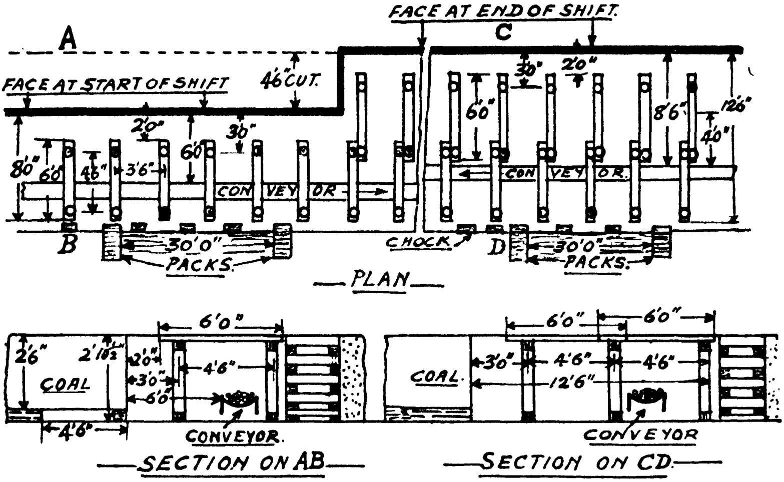

FIG. 52.—Plan and Sections showing Longwall Face Supports.

bar being 2 feet from the coal. The conveyor is 6 feet from the face, and the width from face to row of chocks is 8 feet.

The section on the line *CD* shows the position of the supports after the coal has been stripped off at the end of the shift on the right-hand side of the face. The support dimensions are the same as in the previous case. The conveyor, originally 4 feet from the face, is now 8 feet 6 inches from it. The width from face to row of chocks is 12 feet 6 inches. The hardwood chocks break off the roof in regular fashion as the face advances, and they are moved towards the face with each machine cut.

Auxiliary Main-and-Tail Rope Haulage Arrangement

3. An inbye landing receives tubs from the shaft by a main-and-tail hauler. An auxiliary main-and-tail hauler is to be installed to distribute the tubs inbye. Sketch and describe briefly such a landing, showing the auxiliary haulage house and all terminus rope wheels and leads on both haulages. (40)

A. Fig. 53 is a plan illustrating the details asked for above. The inbye landing of the principal main-and-tail hauler has full and empty sidings to hold 40 tubs. The return wheel for the tail rope is placed below rail level and is covered by protecting plates over which the rail track is laid. The empty set of 40 tubs has

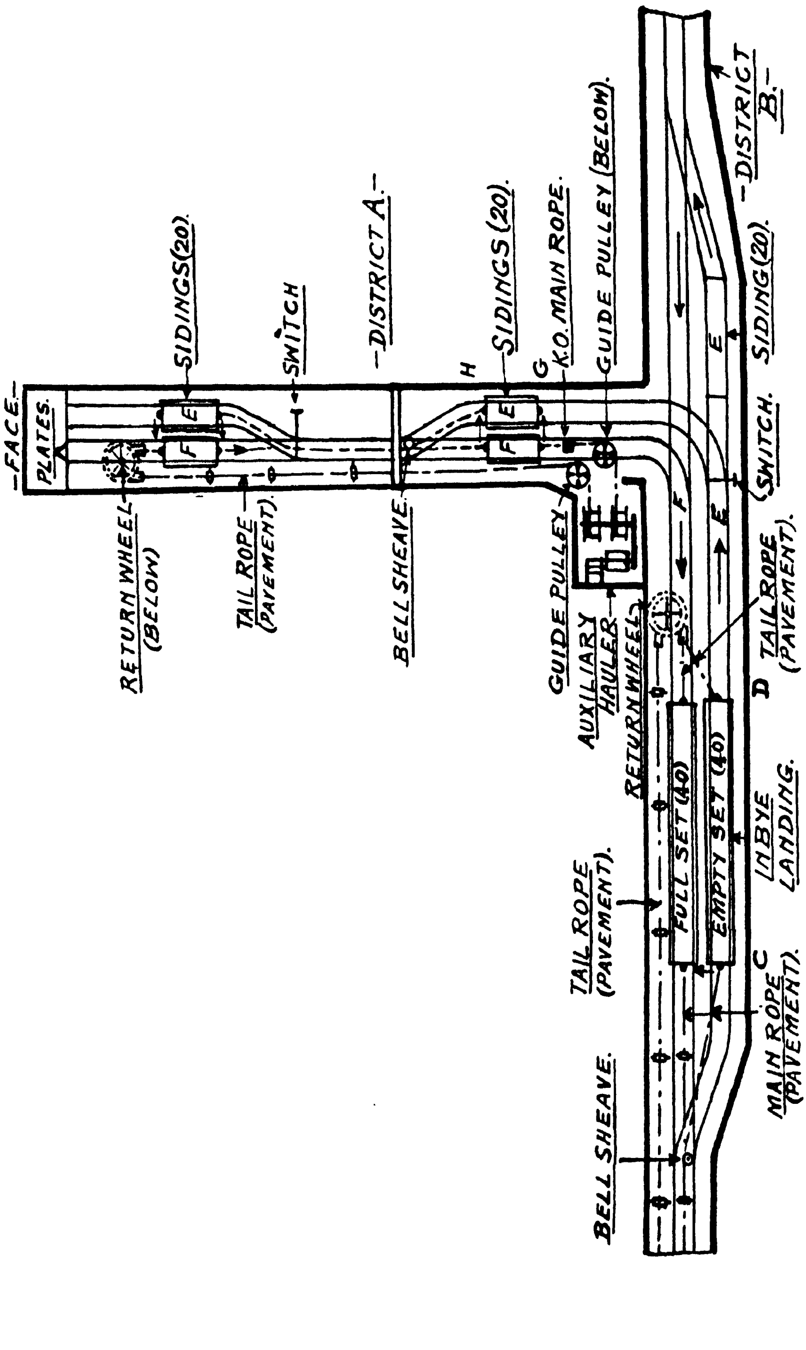

Fig. 53.—Inbye Landing and Auxiliary Main-and-Tail Hauler.

just arrived at the inbye landing, and the tail rope used for hauling it in has been detached from the front of the set and attached to the back of the full set standing ready for hauling out, as shown at *D*. The main rope is detached from the back of the empty set and is fixed to the front of the full set as shown at *C*.

The auxiliary main-and-tail hauler is shown on the left-hand side of the district road *A*, the ropes from this hauler being guided into the proper lead by two pulleys placed at pavement and below pavement level, respectively. The full tubs from district *A* run by gravity to the full siding of the inbye landing, the auxiliary main rope having been detached from the set at the point *KO* in the diagram. This rope is changed over to the back of the empty set in the siding, as indicated by *G*. The empty tubs from the inbye landing run by gravity round the curve to the empty siding in district *A*, from which they are taken to the face of the district by the auxiliary hauler after changing the tail rope from the back of the full set to the front of the empty set as indicated by *H*. Sidings and return wheel near the face are clearly shown on the plan, the return wheel being placed horizontally below pavement level. By having suitable off-take links in the auxiliary rope system, the district *B* might be worked by the same haulage gear.

Freezing and Cementation Methods of Shaft-sinking

4. Describe briefly *either* the " Freezing " *or* the " Cementation " method of sinking shafts through wet ground. When is the method described applicable, and what are its advantages and disadvantages ? (40)

A. The freezing system of sinking shafts consists in freezing water-bearing beds into a hard mass before the actual sinking starts, thus allowing operations to proceed in the usual way as regards the sinking and lining of the shafts. It has been applied very successfully on several occasions in England where difficult beds were met with, both as regards water and loose beds at considerable depths.

Fig. 54 is a plan showing how bore-holes, say 18 to 30 in number according to the size of the shaft, are equally spaced round the selected position of the shaft, say in a circle 30 feet in diameter for a shaft of 20 feet finished diameter. The holes are bored right through the water-bearing beds, depths up to 2,000 feet having been accomplished on the Continent. The holes are lined with wrought-iron lining tubes to support the sides until the freezing-

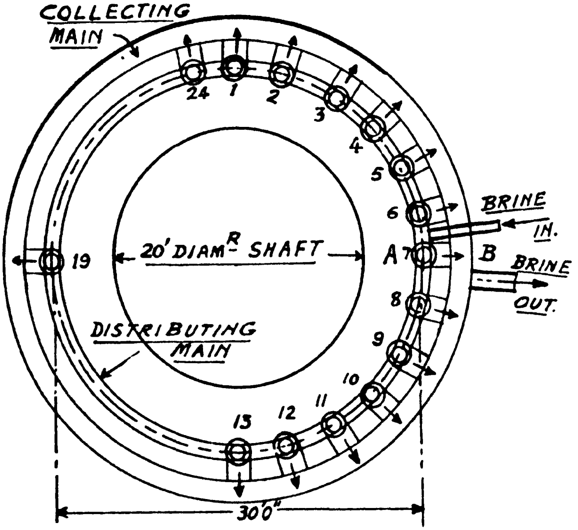

Fig. 54.—Plan showing Position of Holes and Tubes in Freezing Method of Shaft-sinking.

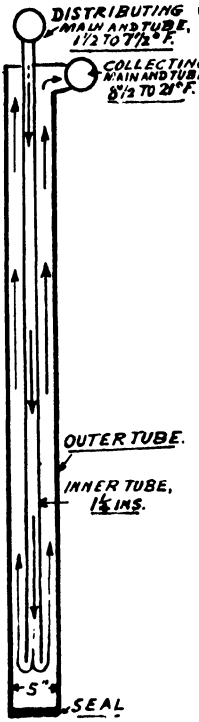

Fig. 54a.—Section on line AB, showing Freezing-tubes.

tubes are inserted, when they are withdrawn prior to freezing being started. The brine mains are also shown in the drawing.

Fig. 54a shows how the inner freezing-tubes, say 1¼ inches diameter, are connected to a distributing brine main arranged round the circumference of the circle. It also shows how the outer freezing-tubes, say 5 inches diameter, are connected to a collecting brine main also arranged around the shaft position. The beds are frozen by a brine solution of about 26 per cent. strength of magnesium chloride, which descends the inner tubes from the distributing main and returns slowly by the outer tubes to the collecting main. The brine extracts heat from the strata as it passes up the outer tubes, amounting to 7° to 14° F. increase in temperature of the brine leaving the distributing main and entering the collecting main. The outer tubes must be perfectly sealed at the base to prevent any escape of the brine, and the joints of the tubes are tested for leakage to a pressure of 30 atmospheres.

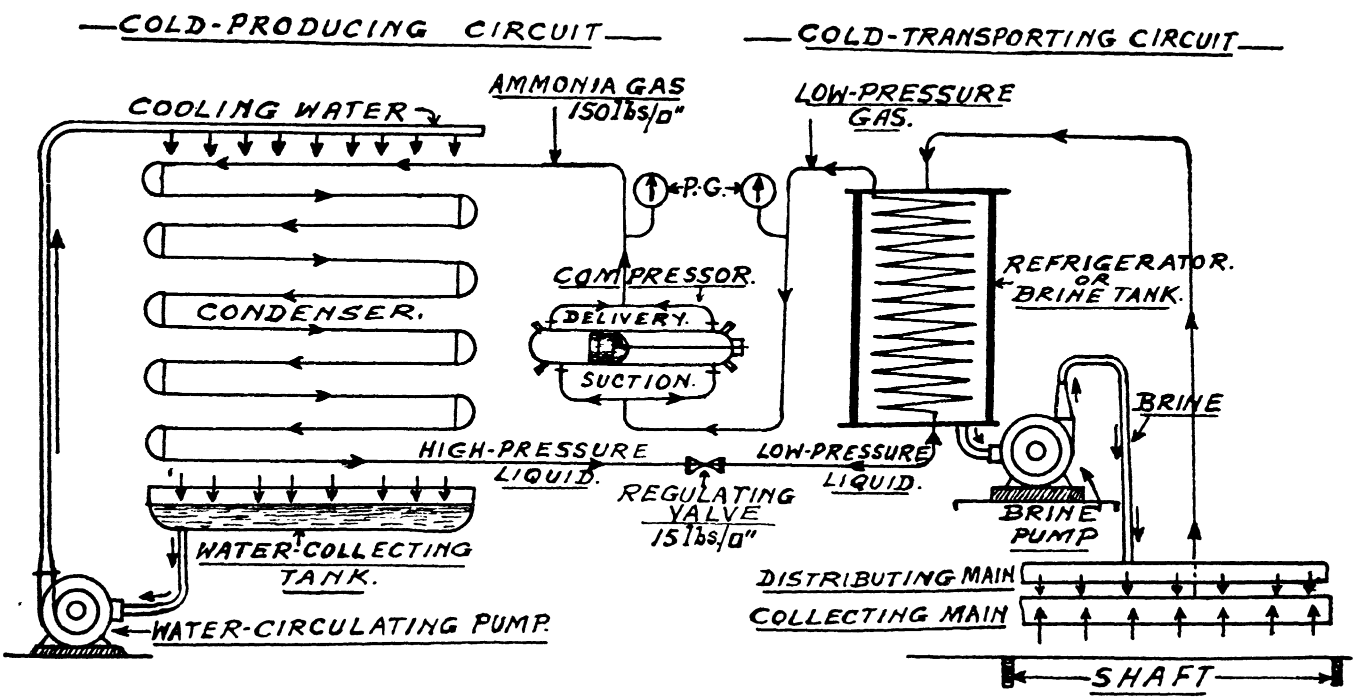

Fig. 55.—General Arrangement of Surface Plant for Freezing.

Surface Freezing Plant.—The general arrangement of the surface plant required for freezing is shown diagrammatically in Fig. 55. There are two separate circuits operating in the surface plant, viz. (*a*) the cold-producing circuit of ammonia NH_3, or carbon dioxide CO_2, and (*b*) the cold-transporting circuit, or brine circuit, of magnesium chloride $MgCl_2$. Compressed ammonia gas from the compressor, at 150 lbs. per sq. inch, passes through a condensing plant in which it is cooled by the application of cold water from the water-circulating pump and becomes liquid at high pressure. After leaving the condenser, the pressure of the ammonia liquid is reduced by a regulating valve to a pressure of 15 lbs. per sq. inch, thus further reducing its temperature and causing an intense degree of cold. In this state it passes through the refrigerator situated inside the brine tank, to cool the brine, and finally passes back again to the compressor in a gaseous state. The brine is taken from the base of the tank by means of a pump, and is circulated through the mains and tubes already referred to until it reaches the top of the tank. It will be noted that the ammonia and brine are thus in continuous circulation in their respective circuits, the former cooling the brine and the latter cooling the strata.

The application of the freezing system might include the sinking of shafts through water-bearing beds of almost any description, hard or soft, at both shallow and great depths, to reduce the quantity of water during sinking operations. It is particularly useful in dealing with beds of running sand containing water. The beds which in actual practice have been sunk through by freezing might be summed up as follows :—

(i) Sand beds exceeding 90 feet in thickness and giving off much water.

(ii) Hard beds of limestone with large cavities giving off large volumes of water.

(iii) Hard and soft beds giving off too much water for pumps to deal with efficiently.

(iv) Beds where the shaft sides would be unsafe with temporary lining at a great depth.

(v) Shafts required near the sea with feeders of salt water, and when the water in the shafts rises and falls with the tides.

The advantages of the freezing method of sinking might be tabulated as follows :—

(i) It is a very reliable method under expert supervision.

(ii) It can be applied to beds that could not be sunk through by any other method.

(iii) After other methods of sinking have failed, it is generally applied as a last resort, but reliable firms must carry out the work.

The disadvantages of the freezing system are :—

(i) It is a very costly method and it is also very slow, thus retarding development and coal production.

(ii) Failures have occurred owing to the ice-wall not being properly formed. The beds are usually frozen solid right to the centre of the shaft, and such occurrences are rare. These failures are usually corrected by the further application of the same system.

(iii) Much expensive plant is required, though in recent years the plant has been hired.

(iv) Expert supervision is required at all times to overcome difficulties which are often met with.

(v) Good watertight linings to keep back water, or strong linings to resist high pressures, are required when the freezing is discontinued and the tubes are withdrawn.

After the shaft is sunk and the lining is completed, the freezing-tubes are withdrawn when the beds have been thawed by circulating steam. The holes are filled with gravel, which is cemented to make a solid plug.

Alternative Answer.—The Cementation method of shaft-sinking has already been described: see pages 37 and 38 and diagrams 25 and 26.

The use of the Cementation method might have reference to practically every kind of water-bearing strata, including many types of gravel, to eliminate the cost of pumping during sinking operations. Such strata might include stratified non-porous fissured rock. Fissured porous rock, such as New Red Sandstone, might be sunk through by cementation, provided a preliminary injection of silicates or of caustic soda is carried out. Cementation is not considered a practical process for dealing with running sand. In addition to the above, it might be applied for the recovery of inundated shafts, to stiffen up brick linings of shafts that are old and wet, for reconditioning old shafts lined with tubbing segments, for underground drifts, dams, fire stoppings to ensure airtightness, and for foundations in strata disturbed by subsidence. Where beds are unstable, with boulders and large feeders of water, it might be applied as a preliminary to freezing in order to reduce the volume of water and to stiffen the beds.

Advantages.—The following advantages are claimed for this method of shaft-sinking :—

(i) It is a quick method at a very reasonable cost.

(ii) A very small plant is required for the work.

(iii) There is only slight risk of partial failure.

(iv) The shaft sides are strengthened permanently by cementation and weaker artificial linings might be used.

(v) The bulk of the water is sealed off permanently from the shaft.

(vi) It has a wide scope of application.

The disadvantages of the system are :—

(i) There might be considerable leakage of cement into cavities and fissures extending to the surface.

(ii) Beds very near the surface might be difficult to deal with.

(iii) The unexpected presence of running sand might result in failure.

(iv) Stoppages might be frequent if the cement pump, pipes and valves are not kept in good working order. These parts are liable to wear quickly owing to cement deposits and high pressures.

Concealed Coalfield

5. Explain the term "Concealed Coalfield" and give one example in Great Britain, with a sketch showing a vertical section of the strata. (40)

A. A concealed coalfield may be defined as being composed of coal-bearing strata lying in a trough or depression of the earth's crust and entirely covered by newer rock formations. Its boundaries do not crop out at any point on the surface of the earth. Most British coalfields lie in trough-shaped depressions only partly concealed under newer rocks. The measures often crop out at the surface at the extremities of the trough.

A good example of a concealed coalfield in Great Britain is the Kent coalfield, composed of a Carboniferous trough of strata completely covered by newer formations. Its boundaries are not determined at the present time owing to the thick covering of newer rocks. Workings in the coal-bearing strata exist in the Dover area in the south, where the strata dip under the Channel. Workings also exist in the north, towards the city of Canterbury.

A vertical section showing the succession of strata at Dover is annexed (Fig. 56).

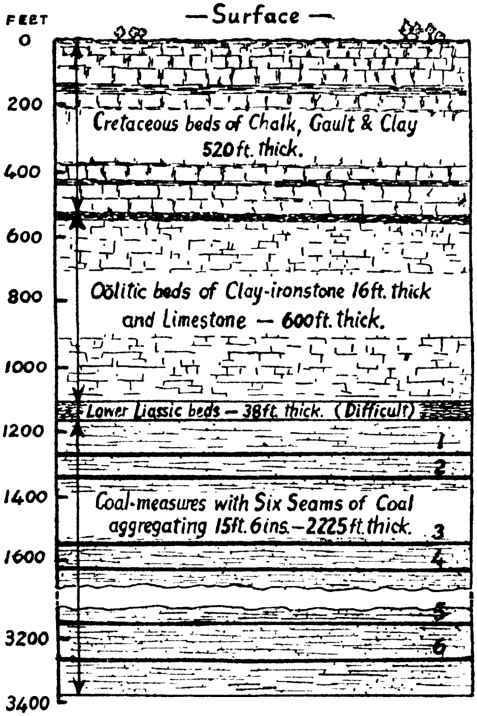

FIG. 56.—Vertical Section of Strata in the Kent Coalfield at Dover.

Conditions for Application of Coal-Cutters and Pneumatic Picks

6. In planning the extraction of a seam of coal by the longwall method, what factors would you bear in mind in recommending the adoption of (*a*) coal-cutting machines; (*b*) pneumatic picks; and (*c*) a combination of both? (40)

A. The factors to be considered in recommending the use of **coal-cutting machines** might be as follows:—

(i) What is the nature of the roof and pavement of the seam? A fairly good roof and an even pavement are both desirable. Are the conditions bad enough to rule out the use of machines?

(ii) Is it possible to work on a straight line of face, as in longwall, or is the seam to be worked on the pillar-and-stall system? These conditions would be considered in selecting the type of machine to be used.

(iii) Do any faults exist at the proposed faces, and can they be overcome if machines are used ?

(iv) What is the cutting material composed of, and is it hard or soft ? What is its position in relation to the coal seam ?

(v) Is it proposed to cut in the seam itself ?

(vi) Will the coal be of sufficient strength to remain in position during undercutting operations ?

(vii) Will packing material be available and suitable from roof or pavement for the building of roadside packs ?

(viii) If the longwall method of working cannot be applied, is the seam of coal suitable for pillar-and-stall work ?

The following factors might be considered when the application of **pneumatic picks** is being investigated :—

(i) What is the nature of the coal and the roof ? Will the coal respond better to the use of pneumatic picks as compared with hand picks ? Will the roof remain safe and unbroken during the operation of coal-getting ? Are the workings to be longwall or pillar-and-stall ?

(ii) What is the type of power available at the face ? Will compressed air be available at the face for the pneumatic picks ?

(iii) Does the seam of coal give off firedamp freely during working operations, thus making the use of electricity at the face and blasting-down of the coal dangerous ?

(iv) Is it necessary to have the coal as large and clean as possible ?

(v) Would the workmen be likely to entertain the use of pneumatic picks instead of hand picks, and will facilities be given for workmen to have instruction in the use of these picks ?

(vi) Will proper consideration be given to the size and type of pick to be used to reduce vibration and the risk of stoppage by breakages ?

(vii) Where there is a bad parting between the coal and the roof, better results are obtained by using pneumatic picks than by using explosives.

The factors to be considered when the use of **pneumatic picks and coal-cutting machines combined** is under investigation are as follows :—

(i) Is the use of explosives considered unsafe after undercutting owing to the presence of firedamp or the possibility of producing a bad roof ? Will the use of pneumatic picks give greater safety ?

(ii) Will it be an advantage to have larger and cleaner coal produced ?

(iii) Is the coal-cutting machine worked by compressed air ?

(iv) Will there be sufficient space available at the face and in the road for compressed-air pipes ?

Sheathed Explosives

7. Describe a cartridge of "Sheathed Explosive" and explain the purpose for which it is designed. (40)

A. Sheathed explosives are those in which the ordinary explosive cartridge is surrounded by about ⅛-inch thickness of sodium bicarbonate $NaHCO_3$, in a finely-divided state. It is separated from the explosive compound by waxed rubber, and is held in position by a strong wrapping of paper, which will withstand a certain amount of rough usage before breaking. The ends of the cartridges are not sheathed.

The purpose of this design is to give increased safety in shot-firing operations where there are breaks in or near the shot-hole containing firedamp. The finely-divided sheathing powder absorbs heat after firing, and the CO_2 generated from the powder gives an extinctive blanket, or quenching effect.

The limiting charge of explosive might be increased without increasing the risk of blown-out and over-charged shots. If the diameter of the sheathed cartridge is increased from 1¼ inches to $1\frac{7}{16}$ inches by sheathing, there is no loss of strength. If the same diameter is maintained at 1¼ inches after sheathing, and the length of the cartridge is increased to provide the same amount of explosive, there is a corresponding loss in efficiency. Sheathing of explosives increases the cost of production as compared with ordinary high explosives in cartridges, but it gives greater safety ; moreover, sheathed explosives can be used under either dry or wet conditions, and they do not deteriorate in storage.

Although sheathing has increased the safety of explosives as used in mining operations, it has recently been pointed out by experts on the subject that complete freedom from explosions of firedamp has not been attained. When a break runs through a shot-hole and firedamp issues therefrom, it is not to be supposed that the use of a sheathed explosive will definitely prevent ignition of the firedamp in all such cases.

PART II.—THEORY AND PRACTICE OF VENTILATION

MAY 1931 EXAMINATION

Ventilation of Workings

1. On the plan of a mine accompanying this paper, show how you would ventilate the working places. Mark the direction of each air-current; also indicate clearly the position of air-crossings, doors, stoppings, regulators and sheets, using the signs laid down in the Regulations. Sheets to be shown thus ∫. (50)

A. The plan referred to in the above question is shown in Fig. 57, all details of the ventilation of the longwall workings being included on this plan. Regulators are not required.

Steart or Multiple-Propeller Ventilating Fan

2. Describe, with sketches, the Steart (or multiple-propeller) ventilating fan, giving details of its construction and the principles upon which it operates, and enumerate what, in your opinion, are the advantages and disadvantages of this fan. (30)

A. Fig. 58 shows, in sectional elevation, the general arrangement of the Steart ventilating fan. This fan is constructed with a number of propellers of the two-bladed or four-bladed types, similar to aero propellers. The blades are built up of wooden laminations and are covered from the tip to within a few inches of the centre-boss with a waterproofed fabric material. The propellers are mounted in banks of four on the driving shaft, which is supported by roller bearings, and they rotate in a steel casing which is supported at one end by the brickwork continuation of the fan drift, and at the other end by the brickwork of the evasée, a brick pillar supporting the centre of the casing. The drive is of

Note.—Under Examination conditions, candidates are required to answer *six* questions from this Section. The figures in brackets indicate the maximum marks allotted to each question.

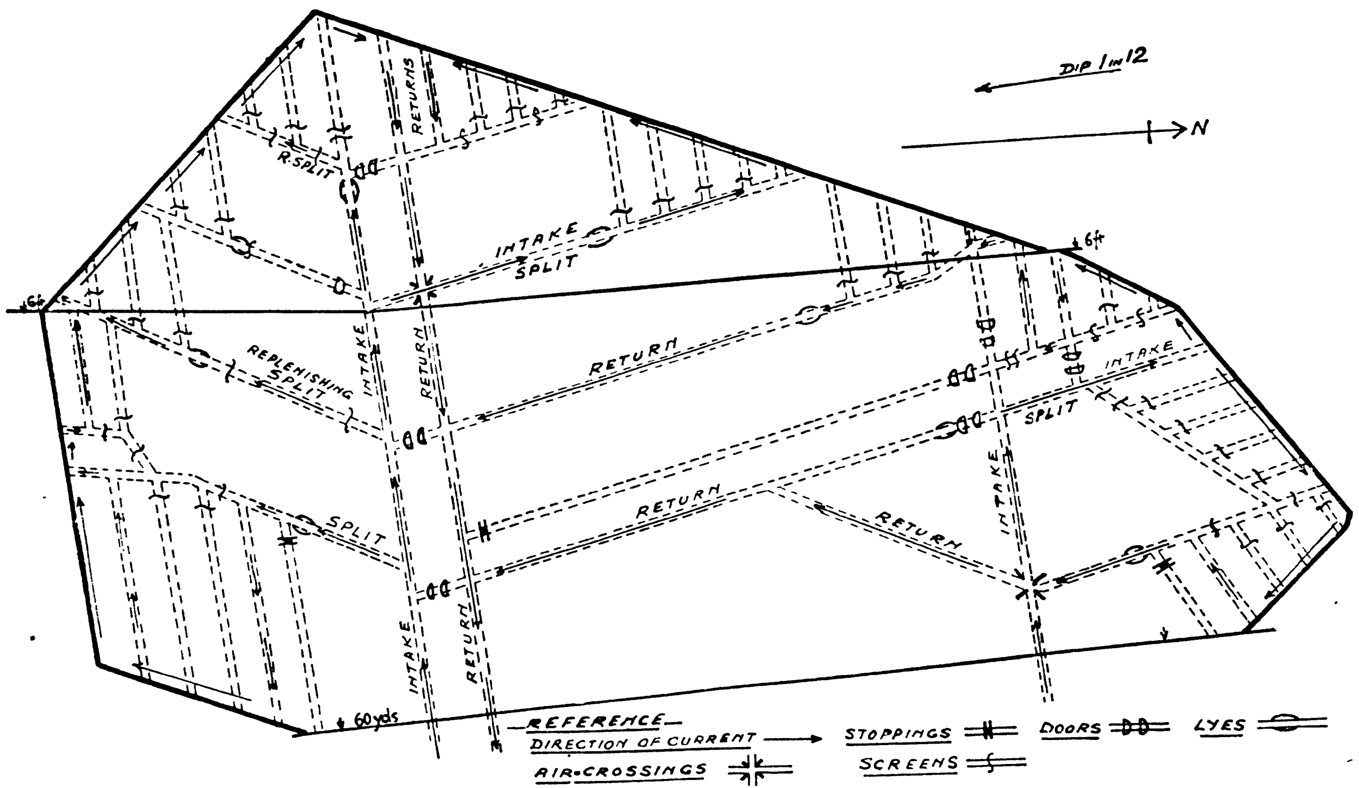

FIG. 57.—Ventilation of Longwall Workings.

the short Lenix belt type, with about 7-foot 6-inch centres, and a jockey pulley for tensioning.

The propellers are rotated by a motor and belt at 300 to 800 revolutions per minute and cause a thrust on the air to force it through the fan parallel to the axis.

The following are examples of plant at present in use :—

(*a*) Two 8-ft. diameter four-bladed propellers ; 730 r.p.m., 75 H.P. motor, 93,000 cu. ft. per min., 1·2 inch water-gauge.

(*b*) Four 9-ft. diameter four-bladed propellers ; 730 r.p.m., 200 H.P. motor, 190,000 cu. ft. per min., 3·4 inch water-gauge.

The advantages claimed for this type of ventilator are : low cost of installation, about one-half that for centrifugal fans ; high efficiency of 70 per cent. upwards ; the working parts are easily replaced when worn ; the fan may be adapted to meet the requirements of developing mines by varying the number of blades and

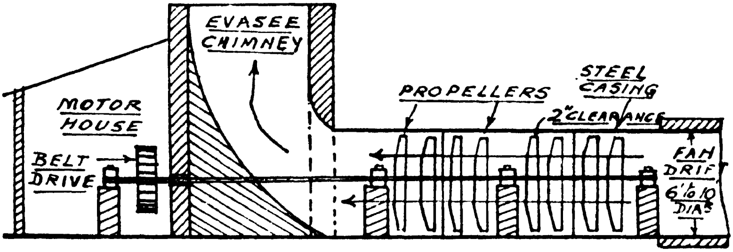

FIG. 58.—Steart Ventilating Fan.

the speed, a feature which is absent in centrifugal fans ; clean-running conditions are obtained, and no deposits appear on the blades ; reversing the direction of rotation of the impellers changes the direction of the air-current, and gives about two-thirds of the original quantity, the reversing process therefore being quick and effective.

The disadvantages of the Steart fan might consist of difficulties caused by the great length of the propeller shaft, and the possibility of an excessive air-slip. Other difficulties might be detected after the fan had been in service beyond the experimental stage.

Lighting Arrangements in a Large and Fiery Mine

3. At a large, deep new colliery, working a fiery seam, what arrangements for lighting do you consider should be adopted : (*a*) near the shafts ; (*b*) on the roadways ; (*c*) at the working faces ? Describe briefly the various lamps you would use. (30)

A. According to the Mines Act Regulations, permanent lighting of roads is allowed as follows :—

(i) From the downcast pit to within 300 yards of the first working face.

(ii) From the downcast pit to within 50 yards of the first working face where electricity is used at the face and where notice has been given.

(iii) From the downcast to within 100 yards of the first working face (where permission has been granted by the Mines Inspector).

(iv) From the upcast for a distance not exceeding 300 yards, where men are drawn in the upcast pit.

(v) The voltage must not exceed 125 volts where newly installed.

Lighting near the Shaft.—Approved electric lamps of special make and of the fixed type might be installed 30 yards apart, each having a capacity of 100 to 300 watts. They should have thick glass covers to protect the bulbs, also suitable wire guards to prevent damage to the cover-glasses. Where compressed air is available, use might be made of M.L. Pneumatic Electric Lamps of 60 watts, 25 volts, and 100 candle-power. In this lamp there is an air-driven turbo-generator, and the lamp and power mains are protected throughout by air under pressure. Any loss in pressure due to rupture of the casing causes an immediate cessation in the current. The general arrangement of the M.L. lamp is shown in Fig. 59.

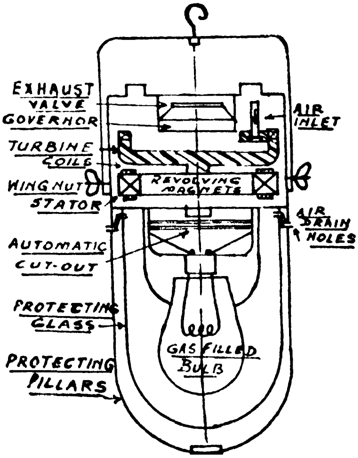

Fig. 59.—M.L. Pneumatic Electric Lamp.

Lighting on Roads.—Approved lamps of the same design as stated above might be used, having a capacity of 30 to 60 watts, and being placed 50 yards apart. If compressed air is available, the smaller type of M.L. lamp might be installed (18 watts, 6 volts, 25 candle-power).

Lighting at the Working Face.—(i) Electric lamps of the hand or cap types might be used at the face, and to allow of the detection of firedamp, one oil-flame safety-lamp must be installed for every eight men employed in longwall workings, and one for every four men employed in other workings. Cap lamps of the self-contained type are considered better for face work than hand lamps of similar construction. The Wolf Alkaline Cap Lamp, of about 9 candle-power, might be used. Fig. 60 shows the general

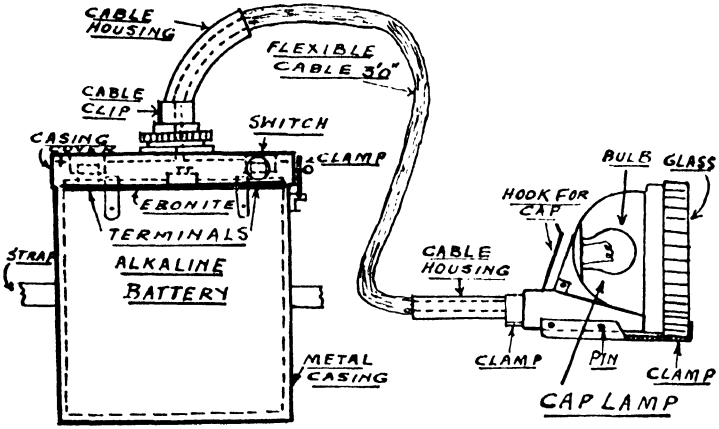

FIG. 60.—Wolf Alkaline Cap Lamp.

arrangement of this lamp. The alkaline battery is contained in a steel casing which is strapped to the waist of the workman. It has an ebonite top covering and contains two terminals. The lid of the casing is detachable and holds the contact clips, cable leads and switch. The cap lamp is connected to the casing lid by 3 feet of flexible cable, and contains the lighting bulb, which is protected by a strong casing containing the glass. The lamp is provided with a suitable hook for the cap of the workman. A type of Schedule A flame safety-lamp might be applied if much firedamp is given off at the faces.

(ii) During recent years many excellent systems of *fixed lighting* for longwall faces have been put into operation, and one of these might be applied, viz. :—

(*a*) *Siemens-Schuckert System.*—This system is greatly used in British mines at loading stations and on longwall faces. An air-cooled transformer is used to take H.T. three-phase current from the mains at 625, 550, 500 or 400 volts, and to step down to 125 volts on the L.T. side. The transformer and lighting circuit are specially protected against short-circuit and overload, and an isolating switch on the transformer can be used to make apparatus dead for repair work. The face cable is of the flexible cab-tyre sheathed type, having five conductors lying symmetrically round a hard rubber centre. The cable is in sections to hold 3 to 6 lamps, the sections being joined by flameproof plug-and-socket connections.

The face cable is tee'd off at intervals of 5 to 8 yards, and lamps are connected through 2 to 3 yards of wander cable. The lamps are of 40 to 60 watts capacity and are provided with well glass and strong wire guard. A cable gland is used for the wander cable. To ensure safety in fiery mines, the well glass and lamp fittings are filled with CO_2 and air under pressure, and the circuit is completed by a pressure-operated switch in the base of the lamp. In the event of breakage, the leakage of CO_2 operates the switch and cuts off the current, while at the same time the CO_2 keeps the mine air from the filament until it has cooled down.

(*b*) *General Electric Co. Installation, in conjunction with S.M.R.B.*—In this system the lamp bulbs are made for 110 volts, being gas-filled, and having a small pressure-operated switch inside. The switch operates if the bulb is broken, and the inert gas around the filament keeps away the mine atmosphere until the temperature of the filament is below the ignition temperature of firedamp and air. The flameproof lighting fitting is designed for 40–60 watt lamps and is attached to a short wander cable plugged into the fitting and held by bolts. There is no possibility of open sparking from the plugs. The cable is of the Maynard double-screened type, containing three conductors, an earth, and inner and outer screens, both screens being connected by a trip controlling switch in the main circuit. In the event of mechanical damage to the cable, the screen circuit is affected before injury can extend to the cores, and the main switch is opened instantly before dangerous conditions arise. The screen system also affords protection against faults between phases, or between phase and earth.

(*c*) *Lighting by Magnetic Induction.*—In this system there is no electrical connection between the lamps and the supply mains. The principle is that each lamp is connected permanently to the secondary of a small transformer, the primary of which is fed from the mains. When the lamp is to be used, its secondary is rendered active by being placed close to its primary coil, so that it is magnetically coupled thereto. When the lamp is to be extinguished, it is simply removed from the neighbourhood of its primary, no switch being necessary and no circuit being broken. The cable voltage is 125, and this is stepped down to 25 volts at the lamps, which are 5 yards apart. Fig. 61 shows the general arrangement of the lamp. The standard base is fitted with a gland for a twin armoured cable. The primary coil is mounted on the core and is protected by the steel casing. The secondary coil is mounted upon a projection from the yoke which forms part of the casing. By means of wing nuts the lamp can be removed, taking with it the

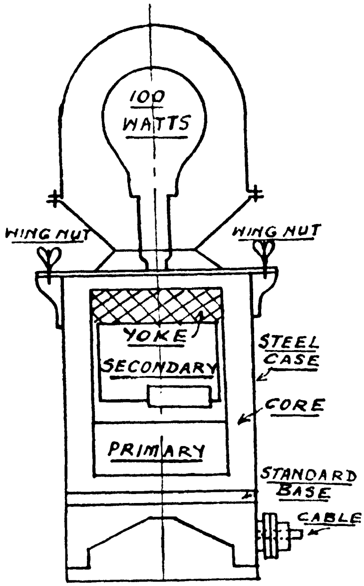

FIG. 61.—Magnetic Induction Lamp.

secondary coil and yoke ; this action at once extinguishes the lamp. The system is in the experimental stage.

(*d*) *The Reyrolle Equipment.*—Current of the three-phase type at 44 volts is taken from a transformer in the gate-end panel, and lamps are connected direct to a face cable. The cable is five-core, with a pilot interlock circuit. The fittings are flameproof, and in addition a wire gauze is placed inside the prismatic well glass.

Surface Arrangements for Winding Coal at both Shafts

4. It is proposed to wind large quantities of coal at both the downcast and upcast shafts. Sketch the arrangement you would adopt at the surface to enable this to be done without interfering with the ventilation. (30)

A. Fig. 62 shows a plan view of the arrangement of a pithead for a large output from two shafts. The winding-cages have three decks for three trams on each deck, and simultaneous decking is accomplished by the use of gravity drop-cages. The full trams, after running by gravity from the winding-cages and drop-cages, are raised by means of electrically-driven hoists, situated outside the pithead buildings, to a level which is sufficient to enable them

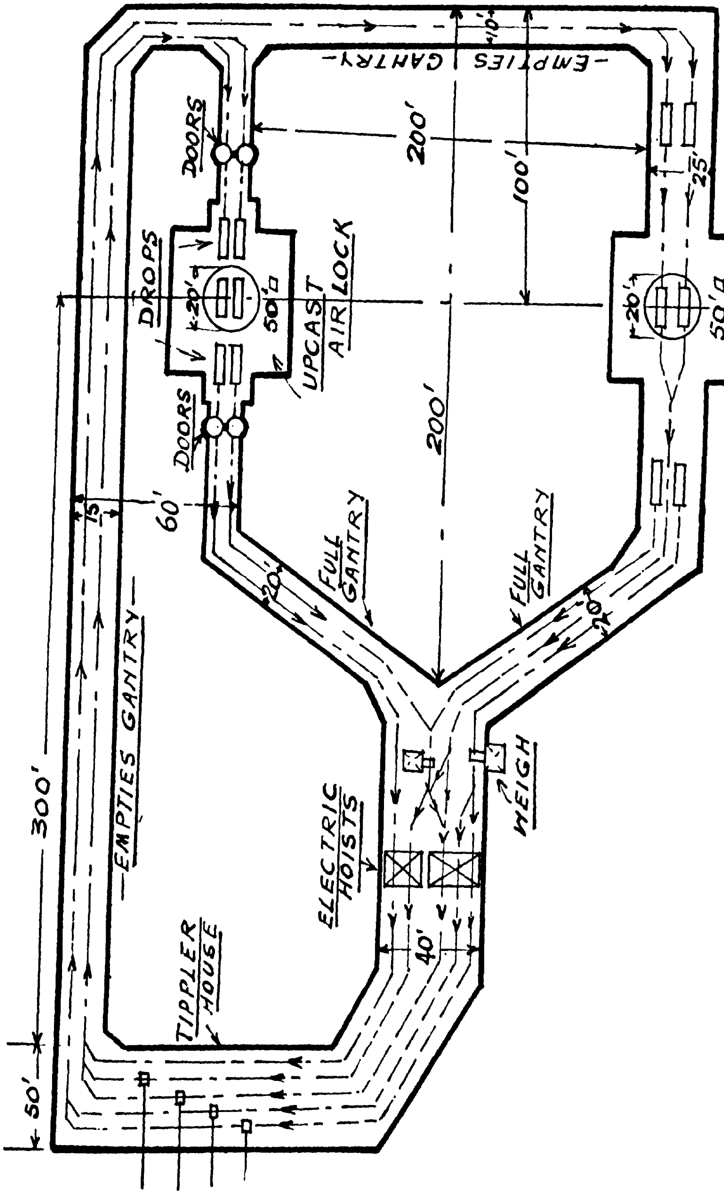

FIG. 62.—Plan View of Pit-head Arrangement with Drops, Gantries and Tipplers for Gravity Working.

to run to the tipplers and back again to the empty-side banking level, without the use of creepers.

The decks are all changed simultaneously by the use of drop-cages on each side of the shaft. Those on the empty side lower six trams to the two bottom decks, while three trams for the top deck are run over the drop-cage. The full-side drop-cages work independently for the two top decks. The empty trams are pushed into the cages by hydraulic rams, while the full trams run out at the opposite side, the whole cycle of decking operations being automatically accomplished by the use of six-stop star wheels and pneumatic controllers.

The raising and lowering operations of the drop-cages are controlled by a cataract oil cylinder situated immediately under each cage, with its piston tube directly connected to the cage. Balance weights are also provided equal to the weight of the drop-cage itself plus half the weight of the trams, thus allowing the cages to return automatically to the loading position. Similar arrangements to the above are made at the pit bottom inset, to allow of simultaneous loading of cages.

To prevent any interference with the ventilation, the downcast shaft top is open to the atmosphere, while the upcast top is entirely enclosed by a suitable air-lock of ample dimensions, as shown in

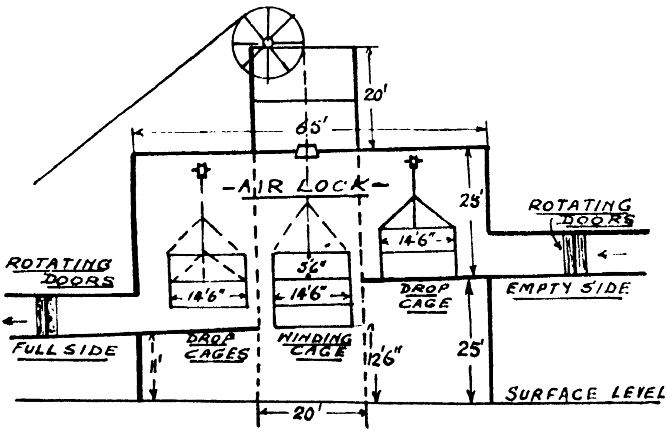

FIG. 63.—Elevation of Upcast Air-lock.

Fig. 63. The full and empty trams pass through rotating doors when entering and leaving the air-lock. Ordinary doors are provided in the air-lock casing for the passage of workmen.

Good Proportion of Air-current at the Working Face

5. State fully what means you would adopt to ensure that a good proportion of the air-current produced reaches the working face. What would you consider a satisfactory proportion: (*a*) in advancing longwall where the faces average a mile from the shaft bottom; (*b*) in retreating longwall where the faces average half a mile from the shaft; (*c*) in pillar-and-stall workings a mile from the shaft? (30)

A. To attain the results required by the above question, the ventilating appliances should be of the best construction in the case of air-crossings, wood doors and permanent stoppings. The latter should be advanced right up to the lyes of the various sections. In longwall workings, good solid packs should be built to prevent air leaking through waste workings into the return airway. Sufficient canvas sheets and temporary stoppings should be fitted up in the various working sections to reduce air leakage to the minimum.

The airways should be made of ample dimensions and be so arranged as to have the intakes and returns as widely separated as possible. Where the airways are used as haulage roads, their dimensions should be as large as possible, especially if the haulage is quick-running. A sketch plan containing all roads and appliances might be made periodically to detect any openings between intake and return airways, while at the same time anemometer readings might be taken to detect leakages.

In the case of the advancing longwall workings the leakage might be from 25 to 40 per cent. In the retreating longwall the leakage might be slightly less, say 20 to 30 per cent. Pillar-and-stall workings usually have big air leakage, and this might reach 50 to 70 per cent. if stooping is being carried on.

Air-Crossings

6. Sketch an air-crossing. What are the two principal objects aimed at in the construction, and how would you ensure their achievement? (30)

A. Fig. 64 shows in plan and section the construction of an air-crossing. The materials of construction are concrete, straight girders and semi-circular girders.

The two principal objects aimed at are strength to resist crush and the force of an explosion, and tightness to prevent leakage of air from intake to return airways. The openings at the crossing should not be less than at the connecting roads. The air-crossing shown in the drawing should achieve these results if constructed in a proper manner.

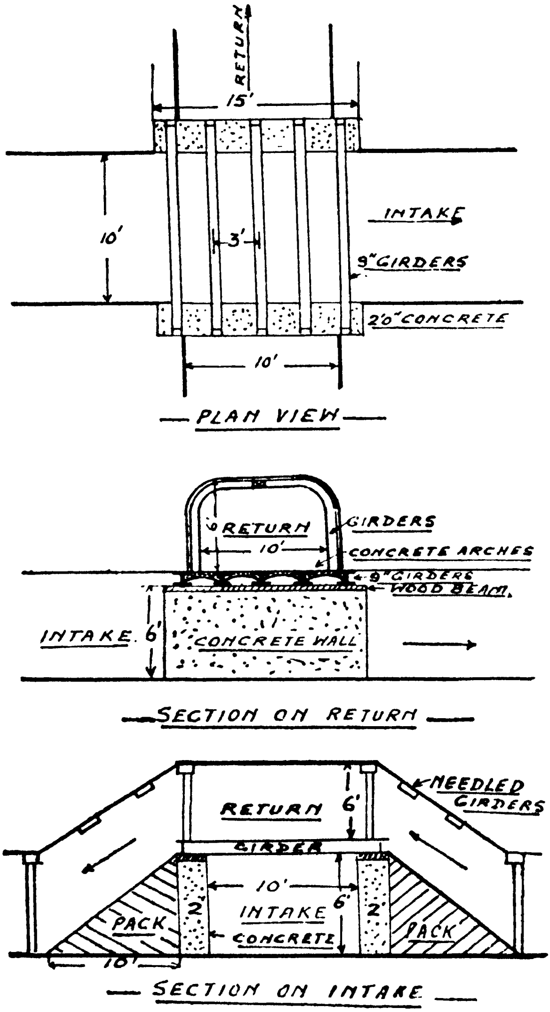

Fig. 64.—Plan View and Sections of an Air-Crossing.

Power and Water-Gauge of Fan

7. A fan produces 300,000 cu. ft. of air per minute with a 4-inch water-gauge. Calculate the horse-power of ventilation, and state

the power you would install to ensure the fan giving this quantity. If the quantity is required to be increased to 400,000 cu. ft. in the same mine, what would be the new horse-power and water-gauge ? (30)

A. Horse-power of ventilation (output) $= \dfrac{300{,}000 \times 4 \times 5{\cdot}2}{33{,}000}$

$$= \frac{2080}{11}$$

$$= \underline{190} \text{ (almost)} \quad . \quad (1)$$

Allowing for an overall efficiency of 75 per cent. in the plant :

Indicated or Electrical horse-power (input) $= \dfrac{190 \times 100}{75}$

$$= \frac{760}{3}$$

$$= \underline{253} \quad . \quad . \quad . \quad (2)$$

Quantity increased to 400,000 cu. ft. per min. with same fan :

Horse-power of ventilation (output) $= 190 \times \left(\dfrac{400{,}000}{300{,}000}\right)^3$

$$= 190 \times \frac{64}{27}$$

$$= \underline{450} \quad . \quad . \quad . \quad (3)$$

Water-gauge $= 4 \times \left(\dfrac{400{,}000}{300{,}000}\right)^2$

$$= 4 \times \frac{16}{9}$$

$$= \underline{7 \text{ inches}} \quad . \quad . \quad (4)$$

Indicated or Electrical horse-power (input) $= 253 \times \left(\dfrac{400{,}000}{300{,}000}\right)^3$

$$= 253 \times \frac{64}{27}$$

$$= \underline{600} \quad . \quad . \quad . \quad (5)$$

NOVEMBER 1931 EXAMINATION

Ventilation of Workings

1. On the accompanying plan of a mine show how you would ventilate the working places. Mark the direction of each air-

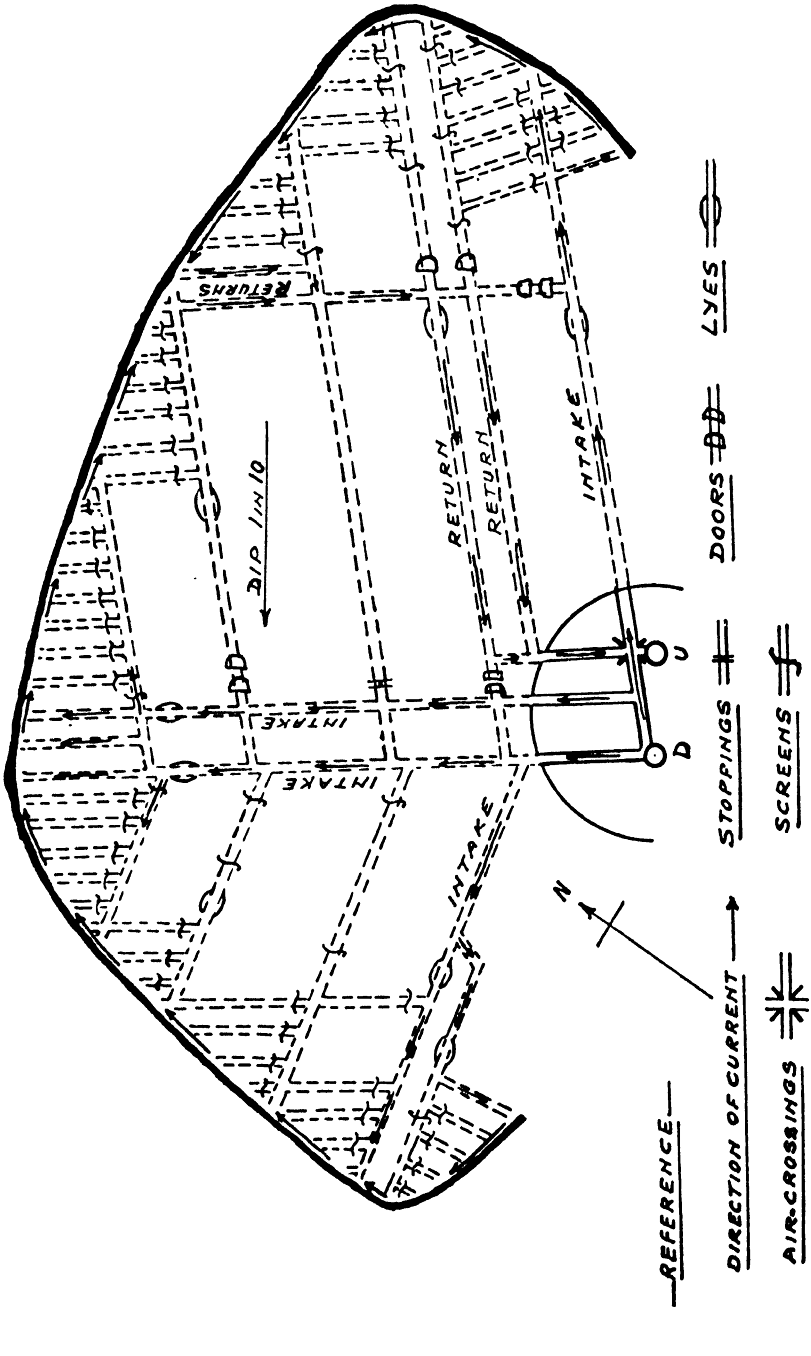

FIG. 65.—Ventilation of Longwall Workings.

current; also indicate clearly the position of air-crossings, doors, stoppings, regulators and sheets, using the signs laid down in the Regulations. Sheets to be shown thus ∱. (50)

A. Fig. 65 shows the plan referred to by the above question. Full details of the ventilation are included on the same. Regulators are not required.

Properties and Composition of Mine Gases—Diffusion of Gases

2. Give the properties and composition of gases usually met with in mines. State what you know about the diffusion of gases. (30)

A. (a) **Firedamp.**—This gas is an impure hydrocarbon gas composed chiefly of methane and varying in its composition according to the percentage of nitrogen, carbon dioxide, sulphuretted hydrogen, oxygen, hydrogen, and other hydrocarbon gases such as ethane, which it may contain.

Chemical Properties: Methane is a chemical compound with the approximate formula CH_4, and is the result of the chemical combination of one atomic proportion of carbon with two volumes of hydrogen. This combination results in the formation of one volume of the gas. It will not support combustion, and lights are extinguished in air containing 15 to 20 per cent., but it is combustible and explosive in air and oxygen when 6 to 15 per cent. is present. Small percentages of Firedamp, say 1·5 to 6 per cent., can be detected by the blue cap formed on the reduced flame of a safety-lamp.

Physical Properties: Firedamp has no colour or taste, but may have a slight smell if H_2S is included in its composition. It is a very light gas and tends to accumulate in cavities, faces of ripping, and faces of rise workings, above the general flow of the air-current. Firedamp has a molecular weight of 16, a relative weight of 8 (hydrogen = 1), and a specific gravity of 0·56 (air = 1).

Physiological Properties: Firedamp is a suffocating gas, and when present in high percentages, exceeding 25 per cent., death results from lack of oxygen.

(b) **Blackdamp.**—This is a gas which is chiefly composed of 4 to 14 per cent. of carbon dioxide and 86 to 96 per cent. of nitrogen. It may also contain carbon monoxide, depending chiefly on the method of its formation in the mine.

Chemical Properties : Blackdamp is a mixture of gases and contains no free oxygen. It will not support combustion and is not combustible. Lights are extinguished in air containing about 15 per cent. of this gas.

Physical Properties : This gas has no colour or smell, but it may have a slightly acid taste owing to the presence of CO_2 in its composition. Its density is greater or less than air according to the percentage of CO_2 it contains.

Physiological Properties : Blackdamp is chiefly a suffocating gas, death generally resulting from lack of oxygen in percentages of 40 to 60. If it is breathed for long periods, very small percentages of blackdamp might result in poisoning owing to the presence of CO_2 and possibly CO.

(c) **Whitedamp.**—This gas is composed of carbon and oxygen chemically combined and contains mostly carbon monoxide.

Chemical Properties : If two atomic proportions of carbon be chemically combined with one volume of oxygen, the result will be two volumes of CO. It is a non-supporter of life and combustion, but is combustible in air and oxygen.

Physical Properties : Whitedamp has no colour, taste or smell. Its molecular weight is 28, relative weight 14 and specific gravity 0·97.

Physiological Properties : It is a highly poisonous gas which is difficult to detect in the ordinary way in mines. It acts as an oxygen robber to the blood of the human body, by combining chemically with the hæmoglobin of the blood to form a carboxy-compound. Its affinity for blood is up to 300 times as great as the affinity of the latter for oxygen. A low percentage, say 0·1 per cent., will render a workman helpless in about one hour. The usual symptoms of poisoning by CO are giddiness after exertion, palpitation of the heart, pounding of the head, loss of leg power or eyesight or interest. This gas is usually detected in mines by the use of mice or small birds, as in small percentages they are affected more quickly than human beings, thus giving time for observation.

(d) **Stinkdamp.**—This is a gas composed of hydrogen and sulphur chemically combined to form sulphuretted hydrogen, or H_2S.

Chemical Properties : This is a compound gas, and results from the chemical combination of sulphur with one volume of hydrogen to form one volume of the gas. It is a non-supporter of life and combustion, but is combustible in air and oxygen. Its presence in air turns lead-acetate paper a black colour.

Physical Properties : Stinkdamp has no colour or taste, but a very disagreeable smell, which is easily perceptible long before the air contains dangerous percentages. Its molecular weight is 34, relative weight 17 and specific gravity 1·18.

Physiological Properties: Stinkdamp is a highly poisonous gas, said to be even more toxic, if less dangerous, than carbon monoxide. It produces pains in the eyes. Low percentages, such as 0·005 per cent., are dangerous, whilst 0·05 to 0·07 per cent. is sufficient to cause death in an hour.

(e) **Afterdamp.**—This gas has the same properties as blackdamp when formed by normal firedamp explosions (such as up to 10 per cent. CH_4), but when formed from explosions with higher percentages of firedamp or with coal-dust as fuel, it contains also **CO** and hydrogen. In the latter case firedamp and other hydrocarbons, such as ethane, might be present in the mixture.

Chemical Properties: Normal afterdamp does not support life and combustion, and it is not combustible in air or oxygen. The percentages of **CO**, CH_4 and oxygen available might, however, result in an abnormal afterdamp being combustible, while at the same time a light might burn in it as usual, but life would be in grave danger.

Physical Properties: Afterdamp has no colour, taste or smell, and its density varies with its composition.

Physiological Properties: Afterdamp is a suffocating gas when present in the form of blackdamp, but it is highly poisonous when **CO** is present in its composition.

Diffusion of Gases.—If two gases be arranged in layers and in contact with each other, they will gradually mix or diffuse into each other until a uniform mixture of the two is obtained. This action is known as gaseous diffusion, and it is in this way that dangerous gases are removed from mines by diffusing into the flowing air-current. When gases are thus diffused they can be separated only by chemical processes.

The rate of diffusion of one gas into another is inversely proportional to the square root of their relative densities, thus:

$$\text{Rate of diffusion of } CH_4 \text{ into air} = \sqrt{\frac{14 \cdot 4}{8}} = 1 \cdot 34$$

$$\text{Rate of diffusion of air into } CH_4 = \sqrt{\frac{8}{14 \cdot 4}} = 0 \cdot 75$$

In a given time 1·34 volumes of CH_4 would diffuse into air, while in the same time 0·75 of a volume of air would diffuse into CH_4. The lighter the gas the greater is its rate of diffusion into air.

Energy Supplied to Ventilating Plant—Power of Velocity of Efflux

3. A fan produces 200,000 cu. ft. of air per minute in a mine with a water-gauge of 3 inches. Assuming an efficiency of 65 per cent. in the fan and motor, calculate what energy is being supplied to the motor, expressed in H.P. If this air leaves the evasée of the fan with a velocity of 500 ft. per minute, what proportion of the power supplied is being used up in this velocity of efflux ? Assume your own figures for the temperature and density of the air. (30)

A. Percentage of combined efficiency $= \dfrac{\text{Output} \times 100}{\text{Input}}$

$$= \frac{\text{H.P. in air} \times 100}{\text{E.H.P. of motor}}$$

$$\therefore \text{ E.H.P. of motor} = \frac{\text{H.P. in air} \times 100}{\text{Combined efficiency}}$$

$$= \frac{200{,}000 \times 3 \times 5{\cdot}2 \times 100}{33{,}000 \times 65}$$

$$= \frac{94{\cdot}5 \times 100}{65} = \underline{145{\cdot}4} \quad . \quad . \quad . \quad (1)$$

Motive column (m) due to velocity of efflux must include inertia and friction :—

$$m \text{ (inertia)} = \frac{(\text{Velocity per sec.})^2}{2g} = \frac{(500 \div 60)^2}{64} = 1{\cdot}08 \text{ ft.}$$

$$m \text{ (friction)} = \left(\frac{\text{Quantity}}{312 \times a}\right)^2 = \left(\frac{200{,}000}{312 \times 400}\right)^2 = 2{\cdot}56 \text{ ft.}$$

(a = Area at efflux.)

Pressure at efflux $= (1{\cdot}08 + 2{\cdot}56) \times 0{\cdot}075$ (lb. weight per cu. ft.)
$= 0{\cdot}273$ lb. per sq. ft.

$$\text{Power at efflux} = \frac{200{,}000 \times 0{\cdot}273}{33{,}000} = 1{\cdot}65 \text{ H.P.} \quad . \quad . \quad (2)$$

$$\text{Proportion of power used at efflux} = \frac{1{\cdot}65 \times 100}{94{\cdot}5}$$

$$= \underline{1{\cdot}75 \text{ per cent.}} \quad . \quad . \quad (3)$$

Sirocco Mine Ventilating Fan

4. Sketch, and describe briefly, a Sirocco mine ventilating fan, giving its characteristic features. (30)

A. Fig. 66 shows in side elevation and plan the general arrangement of the Sirocco fan, as used for exhausting air from mines. The fan wheel is constructed with 64 short blades which are curved forward in the direction of rotation and are secured to circular-

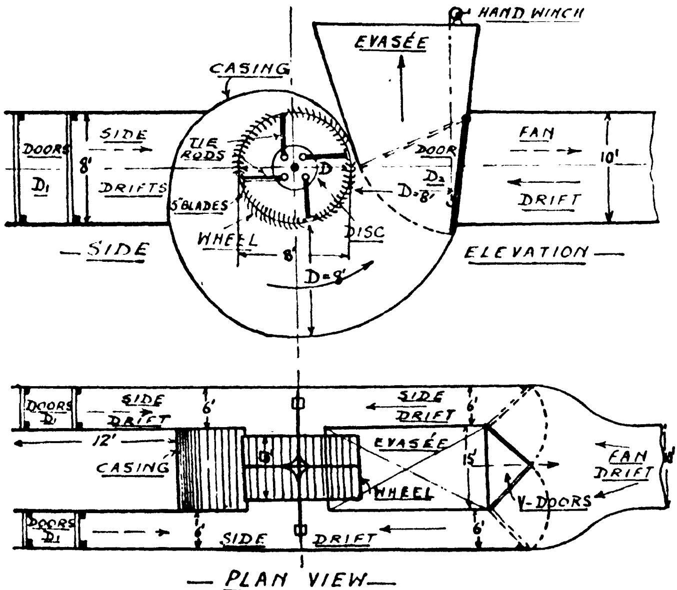

FIG. 66.—Sirocco Mine Ventilating Fan.

shaped rims at each side. These rims are secured to a central disc on the fan shaft by strong tie rods. The length of the blade is about $\frac{1}{20}$ the diameter of the wheel, and its width $\frac{4}{5}$ the diameter. The wheel thus constructed is rotated inside the fixed fan casing, so that air is withdrawn from the fan drift and upcast shaft and expelled into the atmosphere through the evasée chimney of the casing. The fan wheel is driven by belt or ropes from a steam engine, or directly by an electric motor. The sketches show how the air-current might be reversed by lifting the door D_2, opening doors D_1 at the side drift entrance, and opening the V-doors in the fan drift, which close the side drifts on the upcast shaft side at the same time. The arrows indicate the direction of the air-current.

The characteristic features of the Sirocco fan are as follows :—

(i) The fan and casing are constructed of steel, which gives to the plant strength, durability and reliability.

(ii) The plant is small in size, smooth-running, and very suitable for direct or geared electric-motor drive.

(iii) The cost of installation and cost of upkeep of the plant are low owing to its moderate size.

(iv) Good manometric and mechanical efficiencies are obtained and maintained.

(v) The reversing arrangement of the air-current is both simple and effective.

(vi) The air-current takes a short and direct route through the fan casing, thus causing the internal resistance of the fan and eddy currents to be reduced to a minimum.

Testing a Mine for Firedamp

5. How would you test in a mine for firedamp with a flame safety-lamp ? Give the heights of cap for each percentage in the lamp you make use of. Describe any other methods in use for testing for gas in a mine. (30)

A. *The general method* adopted when testing for the presence of firedamp in a mine by using a safety-lamp, is to lower the wick until a very small and non-luminous flame is obtained, say $\frac{1}{10}$ inch high. This is easily accomplished when the lamp is fitted with a screw for adjusting the wick. The lamp should be raised slowly towards the roof or cavity with both hands, one hand holding the lamp at the top ring and the other holding the screw or wick adjustment. If firedamp is present a blue cap will appear on the reduced flame, the height of which cap gives a good estimation of the percentage of firedamp present in the air. It is not wise to have the lamp at any time during a test above the level of the eyes. If a high percentage is met with unexpectedly, the flame might be retained by lowering the lamp steadily and screwing up the wick sharply.

Fig. 67 shows the heights of the firedamp caps as obtained for the various percentages of firedamp in air when using a bonneted Clanny lamp with a flat wick and operating screw.

Other Tests for Firedamp: (*a*) *McLuckie Indicator.*—Fig. 68 shows in diagrammatic fashion the arrangement of the McLuckie firedamp indicator, by means of which a successful test can be carried out in a mine in 6 minutes.

The operating handle is placed at position 1 and the bulb is pressed 12 to 14 times so as to take in a sample of air. The handle is then turned to position 2 for 2 minutes so that the temperature

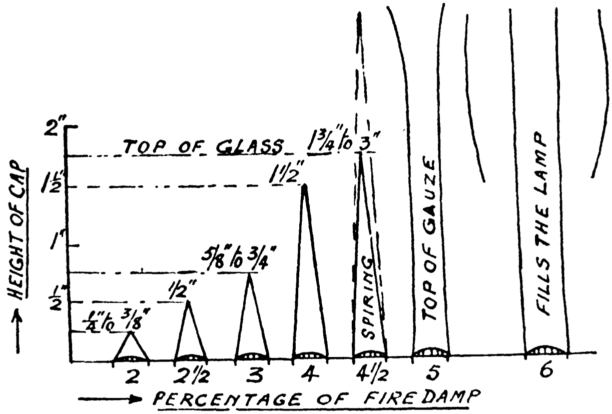

FIG. 67.—Height of Firedamp Caps (bonneted Clanny lamp).

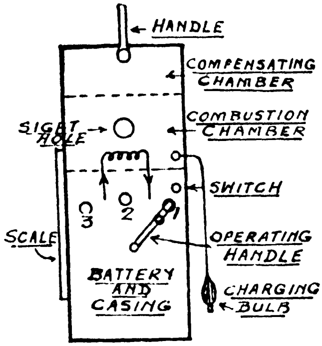

FIG. 68.—McLuckie Firedamp Indicator.

of the sample will be the same as that in the chamber. The handle is then turned back to position 1 for 5 seconds to allow of the sample being at atmospheric pressure. The sample is now burned in the combustion chamber during 2 minutes by turning the handle to position 2 and pulling out the switch to make the filament glow. The progress can be observed through the sight hole provided in the chamber. After allowing the chamber to cool for 2 minutes, during which steam is condensed and a vacuum produced, the handle is turned to position 3, when the percentage of firedamp is indicated on the manometer scale.

This indicator is useful for testing the air for firedamp in main roadways and in the general body of the air-current, rather than at the face during the customary fireman's inspection. The principle upon which the indicator works might be explained by the following equation showing the combustion of firedamp in oxygen :—

$$CH_4 + 2O_2 = CO_2 + 2H_2O \text{ (steam).}$$

(*b*) *Ringrose Alarm Lamp.*—Fig. 69 shows diagrammatically the arrangement of the " Ringrose " firedamp alarm lamp. This lamp

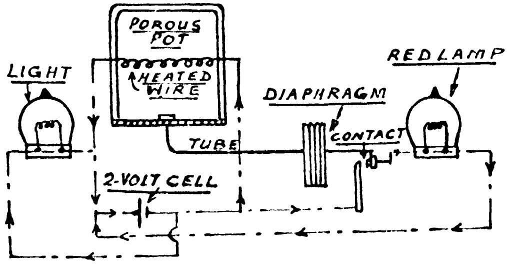

FIG. 69.—Circuit Diagram of Ringrose Firedamp Alarm Lamp.

is set up by the makers to glow red at $\frac{1}{4}$, $1\frac{1}{4}$ or $2\frac{1}{2}$ per cent. of firedamp, as required. Enclosed in the lamp is a porous pot, within which is a wire filament mounted on two studs fixed to a removable carrier, the filament being heated by a 2-volt battery, which also supplies current for the ordinary lighting bulb. Any firedamp present in the air percolates through the double-gauze protected openings to the interior of the porous pot, where it is slowly burned by the heated wire. The above action creates a partial vacuum in the pot, depending on the percentage of firedamp present in the air, and this vacuum is transmitted by a connecting-tube to the aneroid diaphragm, which in turn makes contact with the battery terminals and the red lamp.

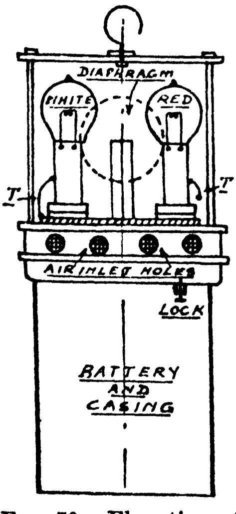

FIG. 70.—Elevation of Ringrose Alarm Lamp.

Fig. 70 shows in elevation the general arrangement of the " Ringrose " lamp. This alarm lamp is very useful for giving indication of the presence of firedamp at fixed points where electric machinery is in operation. It might also be used for testing purposes in main airways and in the general body of the air-current.

The Mueseler Safety-Lamp

6. What is the characteristic principle of the Mueseler type of safety-lamp? Sketch a lamp using this principle. (30)

A. Fig. 71 shows the arrangement of a Mueseler safety-lamp, which incorporates in its construction two important features: firstly, a metal funnel or chimney inside the gauzes to separate the inlet air and the products of combustion and thus give a better light; and, secondly, the fitting of an extinguisher to prevent the taking-off of the lamp bottom with the flame burning. The latter is effected by the wick-tube having to pass through a bobbin-shaped extinguisher which is securely held in the glass ring of the lamp by a lock pin. After the lamp bottom is taken off by unscrewing, the extinguisher can be released from the lamp by withdrawing the lock pin. The extinguisher is inserted on the wick-tube, and after re-lighting the wick, the bottom can be attached again to the lamp and the lock pin pressed into position.

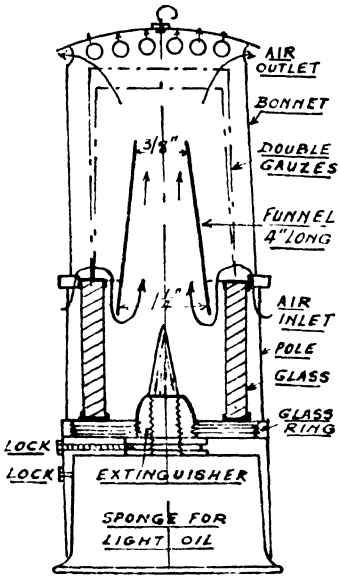

FIG. 71.—Mueseler Safety-lamp.

This lamp gives a fairly good light, but requires careful handling to retain the light.

Calculation of Height of Motive Column*

7. A pair of shafts are each 500 yards deep and 21 feet diameter. Taking the surface barometric pressure as 30 inches, the average temperature in the upcast as 75° F., and that of the downcast as 55° F., calculate the height of the motive column in this case. (30)

A. Barometric pressure at the bottom of shafts $= 30 + \frac{1500}{900}$

$= 31\frac{2}{3}$ inches.

* For explanation of the term "motive column," see p. 165.

$$\text{Average barometric pressure in the shafts} = \frac{30'' + 31\tfrac{2}{3}''}{2}$$
$$= 30{\cdot}83 \text{ inches.}$$
$$\text{Weight of one cubic foot of downcast air} = \frac{1{\cdot}3253 \times 30{\cdot}83}{459 + 55}$$
$$= 0{\cdot}0795 \text{ lb.} \quad . \quad . \ (w_1)$$
$$\text{Weight of one cubic foot of upcast air} = \frac{1{\cdot}3253 \times 30{\cdot}83}{459 + 75}$$
$$= 0{\cdot}0765 \text{ lb.} \quad . \quad . \ (w_2)$$

[*Note.*—459 cu. ft. of air at 0° F. and 30 ins. barometer weigh 39·76 lb.
and 459 ,, ,, ,, ,, ,, 0° F. ,, 1 in. ,, ,, 1·3253 lb.]

$$\text{Motive column} = \left(\frac{w_1 - w_2}{w_1}\right) \times d$$

d = depth of upcast shaft in feet

$$= \frac{0{\cdot}0795 - 0{\cdot}0765}{0{\cdot}0795} \times 1500$$
$$= \underline{56{\cdot}6 \text{ feet}}$$

MAY 1932 EXAMINATION

Comparison of Marsaut and Mueseler Safety-Lamps

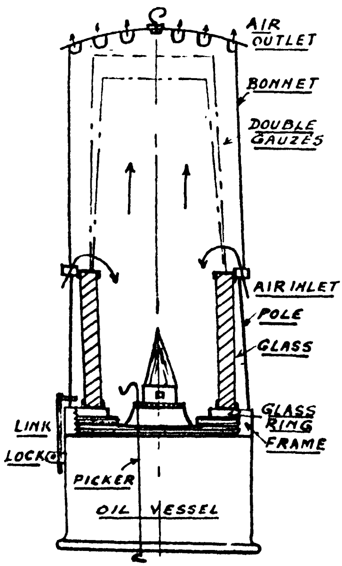

Fig. 72.—Marsaut Safety-lamp.

1. Describe, with sketches, the essential difference between the Marsaut and Mueseler types of safety-lamps. (30)

A. The Marsaut safety-lamp is an improved Clanny lamp, the improvements consisting of double gauzes, a bonnet, and a shorter glass cylinder about 3 inches long by 2 inches diameter, so that the inlet air is taken into the lamp nearer the flame than with the Clanny lamp. Fig. 72 is a sketch showing the chief parts of the lamp.

The Mueseler safety-lamp is also an improved Clanny lamp, the improvements being: shorter glass for better light; internal funnel to separate the inlet air from the products of combustion; light burning oil used from a round wick-tube connected to a saturated sponge in the lamp bottom; and a patent extinguisher to prevent the lamp bottom being taken off with the flame burning. Fig. 71, page 107, shows the chief parts of this lamp.

Horseshoe, Straight-sided and Splayed Steel Arches

2. Discuss the use of steel arches on main ventilating roadways, stating what type of arch you prefer (horseshoe, straight-sided or splayed) and what steps can be taken to minimise friction. A roadway of this kind, lined with splayed steel arches, is 9 feet 6 inches high in the clear, 12 feet wide at springing of arch, and the splay is 7 inches each side (that is, the width of arch is 13 feet 2 inches at floor). When the velocity of the air in this roadway is 450 feet per minute, what is the approximate quantity passing? (30)

A. The introduction of steel arches for supporting main airways in mines has resulted in better airways being maintained. Generally the roads are larger and more uniform in area throughout their length. This has been brought about by the increased strength afforded by the steel arches and the systematic methods applied for erecting the same. Falls of roof are for the most part avoided, and the air-current is not greatly impeded in its passage through these roadways.

The horseshoe type of arch is costly to install in any mine and is introduced only when necessary to resist combined roof and side pressures. From a ventilation point of view it is excellent, giving a large area with the minimum of resistance to the passage of air.

The straight-sided steel arch is much used in main airways to resist top pressure and bad roof. It gives much better conditions for the passage of air-currents than ordinary timbered roads, owing to the larger area obtained and the reduction of resistance.

The splayed type of steel arch is generally preferred to the straight-sided arch where the sides of the road are strong and only roof pressure has to be dealt with. It gives a larger area than the latter type with about the same resistance, and is therefore better from a ventilation point of view.

To minimise friction, the space between the various arches should be filled in with brickwork or concrete so as to present a more regular and smoother rubbing surface to the passage of the air-current.

$$
\begin{aligned}
\text{Area of roadway} &= \text{⏢ part} + \text{◠ part} \\
&= \left(\frac{13\frac{1}{6} + 12}{2}\right) \times 3\tfrac{1}{2} + \frac{12^2 \times \frac{11}{14}}{2} \\
&= 44 + 57 \\
&= 101 \text{ sq. ft.} \\
\text{Quantity of air} &= \text{area} \times \text{velocity} \\
&= 101 \times 450 \\
&= \underline{45{,}450 \text{ cu. ft. per min.}}
\end{aligned}
$$

How Firedamp is given off in Mines—Safety Precautions

3. State the various ways in which firedamp is given off in mines, giving a description of the phenomena connected therewith. At what periods would you expect the quantity to vary, and in view of this, what precautions should be taken to ensure safety ? (30)

A. Firedamp might be given off in a general way during the process of working a seam of coal ; the gas might issue from the broken and exposed face of the coal seam, or from the roof and floor of the seam. It might be liberated quietly from such strata, or it might be accompanied by a cracking and hissing sound. The gas might escape from the coal seam during undercutting and stripping processes respectively, and from the roof and pavement during ripping operations. It might be given off quietly at a regular rate or intermittently from breaks in the strata. A sudden outburst might appear in the form of a " blower " of gas, accompanied by loud reports or bumps.

Firedamp is often met with when faults and dykes intersect the coal and strata, the gas appearing when the workings are approaching near these interruptions and usually for a period after they have been penetrated. In numerous cases the great pressure of the gas in the vicinity of the fault or dyke has resulted in much coal being blown out, and large volumes of gas being liberated in the form of a blower.

When coal pillars are being extracted and the roof of the seam allowed to fall, and in longwall workings when the roof takes a break, firedamp might be liberated in large volumes into the waste working and the actual working area.

The quantity of firedamp given off from faces and strata may vary according to the nature of the coal and associated strata met with. The coal might change to a softer nature and give off more gas, while the strata might change in texture and present more breaks for the issue of the gas. When there is a fall of atmospheric pressure, firedamp might issue freely from waste workings into the working area. The firing of shots to bring down coal and other strata often results in the opening-out of breaks from which firedamp may issue in large volumes.

Precautions against Firedamp.—To ensure safety to the workmen and to prevent accidents, the following precautions are necessary. First and foremost must be the circulation of a good volume of air to render the gas harmless and to carry it away from the mine. This might be possible in all ordinary cases by having a good ventilating pressure and all ventilating appliances in good condition, but in the event of blowers of gas the danger is grave and may be difficult to deal with. Secondly, good safety-lamps should be provided to give ready indication of the presence of gas, while at the same time reliable officials should be employed to keep a watching brief on all workings subject to the presence of firedamp, and to deal with it in a safe and satisfactory manner. Finally, great care must also be exercised to prevent roof fractures as far as possible when firing shots to bring down undercut coal, while at the same time the requirements of the Coal Mines Act should be strictly enforced. Additional care might also be necessary to prevent ignitions of firedamp during the process of undercutting the coal.

Ventilating Fan and Housing

4. Sketch and describe any type of main mine ventilating fan and housing to produce 200,000 cubic feet of air per minute, giving only the principal features and dimensions. (30)

A. Fig. 73 shows the general arrangement of a *Capell* fan suitable for a main mine ventilating plant and capable of the duty required in the above question. Such a fan has 18 curved blades arranged in three sizes, as shown in the diagram. This fan might have 7-foot diameter double inlets, a 15-foot diameter wheel running at 150 revs. per min., and capable of producing 200,000 cubic feet of air per minute with 4 inches water-gauge. Other dimensions are given on the diagram.

The driving motor might be of the three-phase alternating-current type with 6 poles working at 50 cycles and 960 revs. per min. Its electrical horse-power would be 180 for a duty of 127 horse-power in air. The gearing between motor and fan might be accomplished by the use of 10 ropes, 4 inches circumference. The housing for

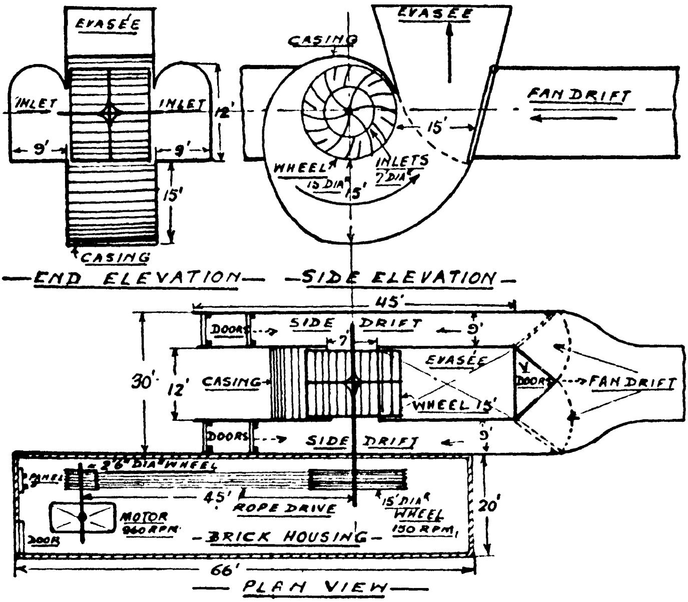

FIG. 73.—Capell Ventilating Fan and Housing.

the motor and driving wheels is shown in the diagram, and might be of brickwork or concrete of the dimensions stated on the drawing.

Calculation of Motive Column, Quantity and Horse-power in Air

5. A seam of coal has been reached by sinking at a depth of 1,200 feet in two adjoining shafts each 18 feet diameter. After these are connected in the seam, and the sinking fans are stopped, it is found after a time that the temperature in the shaft which became down-cast averages 62° F., and that in the other shaft 76° F. The barometer being 30 inches, calculate the height of the motive column. Also make your estimate of the quantity of air produced, and the horse-power of this natural ventilation, showing the steps by which you arrive at this result. If it were decided to place a temporary partition across this air-current's path so as to obtain a water-gauge reading, where would you place it so as to obtain a maximum result ? (30)

A. Assuming the average barometer reading for these shafts at 30⅔ inches or 30·7 inches :—

$$\text{Weight of one cubic foot of downcast air} = \frac{1\cdot3253 \times 30\cdot7}{459 + 62}$$
$$= 0\cdot078 \text{ lb.} \quad . \quad . \quad (w_1)$$

$$\text{Weight of one cubic foot of upcast air} = \frac{1\cdot3253 \times 30\cdot7}{459 + 76}$$
$$= 0\cdot076 \text{ lb.} \quad . \quad . \quad (w_2)$$

$$\text{Motive column} = \left(\frac{w_1 - w_2}{w_1}\right) \times d$$
$$= \left(\frac{0\cdot078 - 0\cdot076}{0\cdot078}\right) \times 1200$$
$$= \underline{31 \text{ feet}} \text{ (almost)} \quad . \quad . \quad . \quad (1)$$

$$\text{Ventilating pressure} = (w_1 - w_2) \times d$$
$$= (0\cdot078 - 0\cdot076) \times d$$
$$= 0\cdot002 \times 1200$$
$$= 2\cdot4 \text{ lb. per sq. ft.} \quad . \quad . \quad . \quad (2)$$

$$\text{Water-gauge} = \frac{2\cdot4}{5\cdot2} = \underline{0\cdot46 \text{ inch}} \quad . \quad . \quad . \quad (3)$$

By using the ordinary formula for friction of air-currents the air velocity can be determined :

$$ha = cplv^2 \times 2 \text{ shafts}$$
$$\text{and } v = \sqrt{\frac{ha}{2cpl}}$$

[h=2·4. Area of shaft (a)=254 sq. ft. Coefficient of friction (c)= 0·005. Perimeter (p)= 56 ft. l= 1200 ft.]

$$= \sqrt{\frac{2\cdot4 \times 254}{2 \times 0\cdot005 \times 56 \times 1200}}$$
$$= \sqrt{0\cdot907}$$

= 0·95 thousands per min. or 950 ft. per min.

$$\text{Quantity} = 254 \times 950$$
$$= \underline{240{,}000 \text{ cu. ft. per min.}} \quad . \quad . \quad . \quad (4)$$

$$\text{Horse-power in air} = \frac{240{,}000 \times 2\cdot4}{33{,}000}$$
$$= \underline{17\cdot45} \quad . \quad . \quad . \quad . \quad (5)$$

To obtain the maximum water-gauge reading, the partition should be placed at the lowest possible point in the road connecting the shafts in the coal seam.

Coefficient of Friction

6. In ventilation what is meant by the term "Coefficient of friction"? State briefly what you know of the method of arriving at the proper figure to be used. (30)

A. Air flowing through the airways of a mine is retarded to a great extent by rubbing against the surfaces of the same, these surfaces usually being known as the rubbing surface, the extent of which depends on the perimeter and the length of the airways. To overcome the resistance thus set up, a ventilating pressure has to be maintained to give the current velocity or movement. In order to calculate the ventilating pressure to overcome friction at desired velocities, a factor known as the "coefficient of friction" must be used, appropriate to the nature of the rubbing surface. The values suggested by Murgue, and which are usually taken, are as follows :—

Roads at working faces, 0·015 lb.	Values per sq. ft. of rubbing surface for a velocity of 1,000 feet per min.
Timbered roads, 0·008 lb.	
Untimbered roads, 0·005 lb.	
Arched roads and modern airways, 0·002 lb.	

To arrive at a proper figure for any mine or road, a simple experiment might be carried through by selecting two parallel airways of good length and uniform area, as shown in Fig. 74. By means of water-gauges, as shown, and by other collected details such as area, perimeter, quantity of air and length, the coefficient of friction can be calculated. The airways should be as flat as possible to reduce the error caused by natural ventilating pressure.

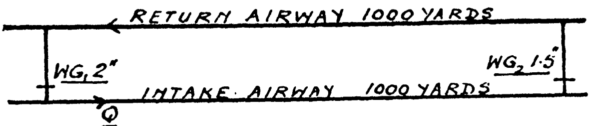

FIG. 74.—Illustrating Investigation of Coefficient of Friction.

In the test referred to above, suppose that the following data were collected: W.G.$_1$ = 2 ins., W.G.$_2$ = 1·5 ins., mean area 90 sq. ft., perimeter 40 ft., quantity 90,000 cu. ft. per min., length of two roads 6,000 ft.

Then water-gauge absorbed by frictional resistance is :—

2 ins. − 1·5 ins., or $\frac{1}{2}$ inch.

By use of the formula $ha = cplv^2$,

$$c = \frac{ha}{plv^2}$$

$$= \frac{\frac{1}{2} \times 5{\cdot}2 \times 90}{40 \times 6000 \times 1^2}$$

$$= \underline{\underline{0{\cdot}01}}$$

NOVEMBER 1932 EXAMINATION

Ventilation of Workings

1. On the plan of a mine which accompanies this paper, show how you would ventilate the working places. Indicate clearly the direction of each air-current; also the positions of air-crossings, doors, stoppings, regulators and sheets, using the signs laid down in the Regulations. Sheets to be shown thus ∫. (50)

A. Fig. 75 is the plan referred to in the above question. The faces have been ventilated by having four splits of air, which are ample for the size of the workings. All details of the ventilation are included on the plan. Regulators are not required.

The Hygrometer and Improvements

2. Describe the hygrometer, mentioning its essential features. Give an example of the readings obtained in an actual case, and show what use can be made of them. Briefly describe any improvements developed of recent years. (30)

A. The hygrometer is an instrument used for measuring the humidity of an air-current, or its condition with reference to moisture. It consists of two thermometers placed side by side on a suitable stand, one thermometer having its bulb exposed to the air-current and termed the "dry bulb," while the other, termed the "wet bulb," has its bulb covered by a silk fabric which is kept moist by an attached wick dipping into a vessel containing water. When the two readings are almost the same the air is humid or damp, but when they are widely separated the air is dry.

Supposing the dry-bulb temperature to be 55° F. and the wet-bulb 50° F., the following information can be obtained :—

$$\text{Relative humidity} = 100 - (\text{difference in readings} \times 5)$$
$$= 100 - (5 \times 5)$$
$$= 75 \text{ per cent.} \qquad (1)$$

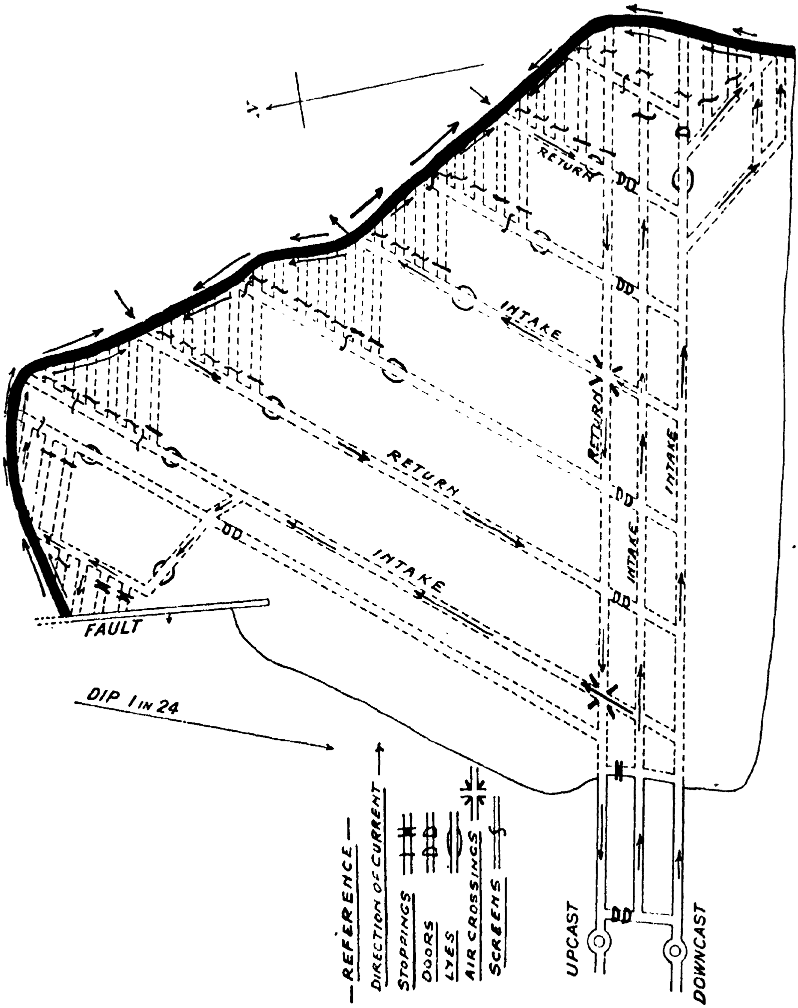

FIG. 75.—Ventilation of Longwall Workings.

Dew-point temperature = Dry-bulb temperature − Difference in readings × k (*see* Glaisher's Tables *below*)

$$= 55 - (5 \times 1{\cdot}96)$$
$$= 45^\circ \text{ F.} \quad . \quad . \quad . \quad . \quad . \quad (2)$$

Grains of moisture per cubic foot = 3·5 (*see below*) . . (3)

GLAISHER'S CONSTANT k.

Dry-bulb temp. ° F.	30	35	40	45	50	55	60	65	70	75	80
k . . .	4·15	2·6	2·29	2·16	2·06	1·96	1·88	1·82	1·77	1·72	1·68

GLAISHER'S TABLE OF DEW-POINT HUMIDITY.

Dew-point temp. ° F.	15	20	25	30	35	40	45
Grains moisture per cu. ft. . .	1·0	1·3	1·5	1·8	2·3	2·8	3·5
Dew-point temp. ° F.	50	55	60	65	70	75	80
Grains moisture per cu. ft. . .	4·1	5·0	5·9	6·8	7·9	9·4	10·9

Improved Hygrometers.—Fig. 76 shows in front elevation the arrangement of *Lloyd's Hygrodeik,* which is a convenient form of hygrometer. Results are obtained from the thermometer readings by the use of the attached chart, index hand and sliding pointer. Tables are incorporated on the chart, and calculations, as given above, are entirely dispensed with.

Example :

Dry-bulb reading 68° F.
Wet- „ „ 60° F.

Set the sliding pointer of the index hand to correspond with 60° F. wet-bulb on left side of instrument ; then swing the index hand over towards the right until the pointer intersects the curved line coming from 68° F. on the dry-bulb scale. Result : Relative humidity 65 per cent. The dotted curved line indicates about 4·8 grains per cubic foot on the upper scale, and 55° F. dew-point temperature on the dry-bulb scale for the pointer position.

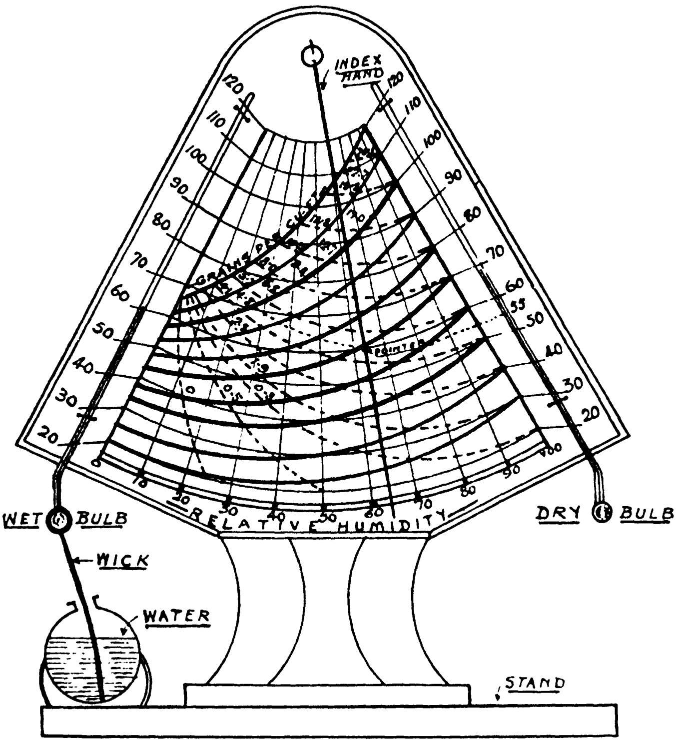

FIG. 76.—Lloyd's Hygrodeik.

Fig. 77 shows in front elevation the arrangement of the *Storrow Whirling Hygrometer*, which might be used in mine airways. The thermometer stand is made to rotate round a fixed handle, held in

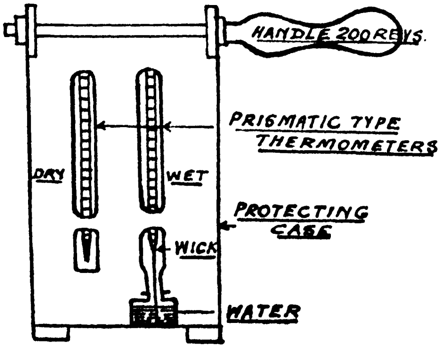

FIG. 77.—Storrow Whirling Hygrometer.

the centre of the roadway to obtain reliable readings, and about 200 revolutions are made before the readings are taken. Results are calculated from Marvin's Psychrometric Tables, specially prepared. This instrument is portable and simple in construction.

Weight of Air in Shafts and Natural Water-Gauge

3. Calculate the weight of air in a downcast shaft 20 feet diameter and 450 yards deep, the barometer reading at the surface being 31 inches, and the average temperature in the shaft being 50° F. Also the weight of air in the adjoining upcast shaft of similar dimensions, having an average temperature of 82° F. Assuming the barometer on the following day to be at 29 inches, make similar calculations for these shafts ; and estimate the water-gauge at the shaft bottom due to the natural conditions on each day. (30)

A. The above results can be obtained by calculating the weight of 1 cubic foot of air in each of the shafts, without going to the unnecessary trouble of determining the weight of air in the respective shafts.

(I.)

Barometric pressure at surface = 31 inches (First day)
Barometric pressure at shaft bottom = $32\frac{1}{2}$ inches (see p. 107)
∴ Average barometric pressure = $31\frac{3}{4}$ inches . . (1)

$$\text{Weight of 1 cubic foot of air} = \frac{1{\cdot}3253 \times \text{barometer reading}}{459 + \text{temperature }^\circ\text{ F.}}$$

$$\text{Weight of 1 cubic foot of downcast air} = \frac{1{\cdot}3253 \times 31\frac{3}{4}}{459 + 50}$$

$$= 0{\cdot}08267 \text{ lb. } (w_1) \qquad . \quad (2)$$

$$\text{Weight of 1 cubic foot of upcast air} = \frac{1{\cdot}3253 \times 31\frac{3}{4}}{459 + 82}$$

$$= 0{\cdot}07778 \text{ lb. } (w_2) \qquad . \quad (3)$$

$$\text{Natural water-gauge} = \frac{(w_1 - w_2) \times \text{depth in feet}}{5{\cdot}2}$$

$$= \frac{(0{\cdot}08267 - 0{\cdot}07778) \times 1350}{5{\cdot}2}$$

$$= 1{\cdot}27 \text{ inches} \qquad . \quad . \quad . \quad . \quad (4)$$

ALTERNATIVE SOLUTION.

Weight of downcast air $= 20^2 \times \frac{11}{14} \times 1350 \times 0{\cdot}08267$ lb.
$= 35{,}080$ lb. . . . . . (1)

Weight of upcast air $= 20^2 \times \frac{11}{14} \times 1350 \times 0{\cdot}07778$ lb.
$= 33{,}000$ lb. . . . . . (2)

$$\text{Natural water-gauge} = \frac{35{,}080 - 33{,}000}{314 \text{ sq. ft. area} \times 5{\cdot}2}$$
$= \underline{1{\cdot}27 \text{ inches}}$ . . . . . (3)

(II.)

Barometric pressure at surface $=$ 29 inches. (Second day)
Barometric pressure at shaft bottom $= 30\frac{1}{2}$ inches
$\therefore$ Average barometric pressure $= 29\frac{3}{4}$ inches . . (1)

$$\text{Weight of 1 cubic foot of downcast air} = \frac{1{\cdot}3253 \times 29\frac{3}{4}}{459 + 50}$$
$= 0{\cdot}07746$ lb. . . (2)

$$\text{Weight of 1 cubic foot of upcast air} = \frac{1{\cdot}3253 \times 29\frac{3}{4}}{459 + 82}$$
$= 0{\cdot}07288$ lb. . . (3)

$$\text{Natural water-gauge} = \frac{(0{\cdot}07746 - 0{\cdot}07288) \times 1350}{5{\cdot}2}$$
$= \underline{1{\cdot}19 \text{ inches}}$ . . . . . (4)

ALTERNATIVE SOLUTION.

Weight of downcast air $= 20^2 \times \frac{11}{14} \times 1350 \times 0{\cdot}07746$
$= 32{,}865$ lb. . . . . . (1)

Weight of upcast air $= 20^2 \times \frac{11}{14} \times 1350 \times 0{\cdot}07288$
$= 30{,}920$ lb. . . . . . (2)

$$\text{Natural water-gauge} = \frac{32{,}865 - 30{,}920}{314 \times 5{\cdot}2}$$
$= \underline{1{\cdot}19 \text{ inches}}$ . . . . . (3)

Main Ventilating Fan—Saving in Power by Evasée

4. Sketch a main mine ventilating fan fitted with an evasée or expanding chimney for the outlet of air. Assuming that a quantity of 200,000 cu. ft. per minute is produced, estimate the saving of power effected by the use of this evasée when the area at the smaller end is 10 feet square and that at the other end 20 feet square. (30)

A. For the first part of the above question see Figs. 66 (Sirocco fan) and 73 (Capell fan).

Velocity of air at base of evasée

$$= \frac{200{,}000}{10 \times 10 \times 60} = 33{\cdot}33 \text{ ft. per sec.} \quad . \quad . \quad (1)$$

Velocity of air at top of evasée

$$= \frac{200{,}000}{20 \times 20 \times 60} = 8{\cdot}33 \text{ ft. per sec.} \quad . \quad . \quad . \quad (2)$$

∴ Gain in pressure per square foot

$$= \frac{(\text{Velocity at base})^2 - (\text{Velocity at top})^2}{2g} \times \text{weight of air per cu. ft.}$$

$$= \frac{33{\cdot}33^2 - 8{\cdot}33^2}{64{\cdot}4} \times 0{\cdot}075$$

$$= \frac{1111 - 69}{64{\cdot}4} \times 0{\cdot}075$$

$$= 1{\cdot}21 \text{ lb.} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

$$\text{Saving in power} = \frac{200{,}000 \times 1{\cdot}21}{33{,}000} = \underline{7{\cdot}33 \text{ H.P.}} \quad . \quad . \quad . \quad (4)$$

Appliances required for Splitting an Air-Current

5. Assume a colliery ventilated by a single current of 40,000 cu. ft. per minute, with a water-gauge of 3 inches at the fan drift. It is desired to split this air into three currents. Taking your own example of conditions, give a list of all the appliances and arrangements you consider necessary, and a brief account of the operations. When the change has been accomplished, how would you proceed to ascertain the results ?

A. When an extensive change in the ventilation is about to be made at a colliery, it is safer and more convenient to accomplish this by having a good system or organisation for carrying through the work.

A detailed plan should be carefully prepared from actual observations, showing all ventilating appliances and haulage arrangements, such as screens, doors, regulators, temporary and permanent stoppings, lyes or flats, haulage roads and air-crossings. With this plan at hand, the new system for three air-splits could be properly designed so as to have separate intakes and returns for each split from convenient spots in main intakes and main returns. New fittings would be carefully indicated on the plan in red, while unnecessary fittings would be cross-marked. When complete, this plan should be handed over to a competent official to have the new appliances installed ready for the change-over. Old appliances should also be removed as far as possible without reducing the efficiency of the ventilation.

When all the above work is completed, arrangements must be made for the change-over. Officials and workmen would be detailed for this work, say at the week-end. They would be posted at all points where changes are required and instructed to start operations at a definite time. These might include the breaking down or completing of stoppings and sheets, opening and closing doors, and fixing regulator shutters. Finishing details might be carried out afterwards.

After the change is completed and the air is circulating as required, an examination of each split should be carried out to detect the presence of firedamp, while at the same time the air-current should be measured in each split. These details will determine whether further alterations are required or not.

How Horse-power or Energy in Air-Current is Absorbed

6. What becomes of the horse-power or energy applied to produce the ventilating current in a mine ? Enumerate and shortly describe the successive losses as the air travels from the open atmosphere through the workings to the outlet of the fan. (30)

A. As the air travels through a system of mine workings the energy applied to it is used up mostly in overcoming friction and inertia at the various parts as follows :—

(*a*) Downcast Shaft : Friction and inertia. Impact due to impediments such as cages, buntons and guides. Eddy currents due to the above.

(*b*) Mine Airways and Faces : Friction and inertia. Impact at bends in the roads and timbers. Impact and eddy currents by impediments such as hutches, machinery and canches. Regeneration of velocity at contractions of roads and at regulators. Internal air circuits.

(*c*) Upcast Shaft : The same conditions might exist as given under the heading of downcast shaft.

(*d*) Ventilating Fan : Friction, impact and eddy currents inside the fan.

MAY 1933 EXAMINATION

Ventilation of Workings

1. On the accompanying plan (see page 124), show how you would arrange the ventilation. This is a plan of a pit bottom of a large modern colliery working a fiery seam and employing about 1,200 men in four large districts of nearly equal size. Winding is carried on at both shafts. Show how you would arrange the ventilation of this colliery, marking the direction of each air-current, and the positions of all air-crossings, doors, stoppings, regulators and sheets, using only the signs laid down in the Regulations. Colour the path of the return-air red and that of the intake blue. The seam is level. (50)

A. Fig. 78 shows the plan referred to in the above question. The ventilation has been arranged to suit the four large districts and to allow of winding at both shafts. The intake airways are shown open, and the return airways are shaded to show a distinction between the two. Ventilation appliances are included by signs laid down in the Regulations. Regulators and sheets are not required.

Horse-power and Efficiency of Ventilation

2. A ventilating fan delivers 200,000 cu. ft. of air per minute from its evasée at a velocity of 1,000 ft. per minute. The water-gauge in the fan drift is 2 inches. Calculate the horse-power of ventilation. Assuming the air to weigh 0·08 lb. per cubic foot, what proportion of the energy applied to the air is absorbed in imparting the exit velocity to the air-current ? If the combined efficiency of the fan and motor is 60 per cent., what horse-power would have to be supplied to the motor to produce the above quantity ? (30)

A. Horse-power in ventilation

$$= \frac{200{,}000 \times 2 \times 5{\cdot}2}{33{,}000} = \frac{200 \times 10{\cdot}4}{33} = \underline{63 \text{ H.P.}} \qquad . \qquad . \quad (1)$$

Energy absorbed in imparting exit velocity = Inertia + friction.

$$\text{Inertia motive column} = \frac{v^2}{2g} = \frac{\left(\frac{1{,}000}{60}\right)^2}{2 \times 32} = 4{\cdot}3 \text{ ft.}$$

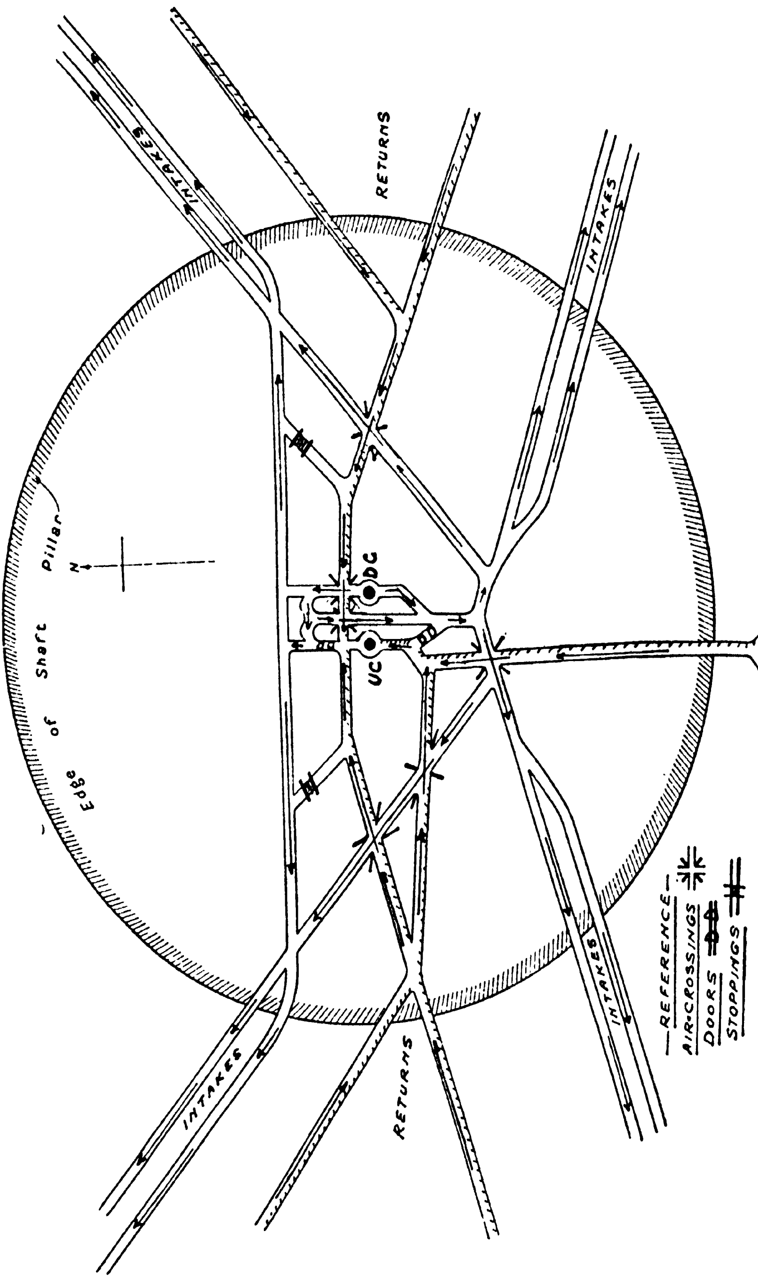

FIG. 78.—Ventilation of Main Roads near Shafts.

$$\text{Friction motive column} = \left(\frac{q}{312 \times a}\right)^2 = \left(\frac{200{,}000}{312 \times 200}\right)^2 = 10{\cdot}3 \text{ ft.}$$

$$\text{Pressure} = (4{\cdot}3 + 10{\cdot}3) \times 0{\cdot}08 = 1{\cdot}17 \text{ lb. per sq. foot.}$$

$$\therefore \text{Energy absorbed} = 200{,}000 \times 1{\cdot}17 = \underline{234{,}000 \text{ ft.-lb. per min.}}$$

$$\underline{\text{or } 11 \text{ per cent.}} \quad . \quad . \quad . \quad (2)$$

$$\text{Input to motor} = \frac{63 \times 100}{60} = \underline{105 \text{ H.P.}} \quad . \quad . \quad . \quad (3)$$

Effect of Heated Atmosphere

3. Describe fully how you would ascertain the moisture present in the air in the fan drift of a mine, giving the results of an actual case, and explain whence the moisture is derived. Under what conditions is moisture in the air of great importance, and what effect has it on the health of the workmen ? (30)

A. See Figs. 76 and 77 for details of the Hygrodeik and Whirling Hygrometers.

Figures of Actual Test :	Dry-bulb temp. 70° F.	Shaft 1,300 feet deep.
	Wet „ „ 64° F.	
Results :	Relative humidity .	. 70 per cent.
	Dew-point temp. .	. 61° F.
	Grains per cubic foot	. 5·9

Whence the Moisture is Derived.—In passing from the downcast shaft through the main intake airways of a mine, the air-current is increased in temperature, and it is therefore capable of taking in moisture at a quick rate from road surfaces and dust accumulations on such roads. Moisture is also produced by oxidation and heating in the mine ; also naturally from the strata; and by the breathing of workmen and the burning of lights. At the working faces the air reaches a high temperature, and it becomes heavily charged with moisture by the evaporation of sweat from the bodies of the workmen.

Importance of Moisture in the Air.—A humid atmosphere assists in preventing ignitions of coal-dust at the working face. Such an atmosphere, however, also causes rotting of timber supports in return airways, and the danger of falls of roof strata arises. Strata are generally affected by moisture, being loosened, and falls of roof and sides occur. In the presence of fine dust, a humid atmosphere causes a deposit of dust on the fan blades and thus reduces the efficiency of the fan. One per cent. increase in the moisture content of an air-current reduces the candle-power of flame lamps by 6 per cent.

Effect on the Health of the Workmen.—There is less evaporation of sweat from the bodies of the workmen in an atmosphere greatly charged with moisture, and healthy respiration is impossible; thus fatigue is produced, and health is affected. A temperature of 80° F. wet bulb is considered dangerous for the workmen in mines, and under these conditions a strong ventilating current must be constantly maintained.

Calculations of Weight and Volume of Air

4. Calculate the volume, in cubic feet, of a ton of normal air at a temperature of 55° F. and a barometric pressure of 31 inches. Also the volume of the same weight of air at 75° F. and a barometric pressure of 28 inches. What would be the weight of 1,000 cubic feet of air at 60° F. and 30 inches barometric pressure? (30)

A. By experiment, the weight of 459 cubic feet of air at 0° F. and at 30 inches barometric pressure is 39·76 lb. The weight of 459 cubic feet of air at 0° F. and 1 inch barometric pressure is therefore 1·3253 lb.

Weight of x_1 cu. ft. of air in first case

$$= \frac{x_1 \times 1{\cdot}3253 \times 31}{459 + 55} = 2240 \text{ lb.}$$

$$\therefore x_1 = \frac{2240 \times 514}{1{\cdot}3253 \times 31} = \underline{28{,}030 \text{ cu. ft.}} \quad . \quad (1)$$

$$\text{Similarly in second case } x_2 = \frac{2240 \times 534}{1{\cdot}3253 \times 28} = \underline{32{,}230 \text{ cu. ft.}} \quad . \quad (2)$$

Weight of 1000 cu. ft. of air

$$= \frac{1000 \times 1{\cdot}3253 \times 30}{459 + 60} = \underline{76{\cdot}61 \text{ lb.}} \quad (3)$$

Modern Flame Safety-Lamp

5. Give a full description, with sketches, of a modern flame safety-lamp. (30)

A. A modern flame safety-lamp of the *Protector* type is shown in Fig. 79. This lamp is of the two-piece design, marked *A* and *B* in the sketch. The part *A*, consisting of the bonnet, can be

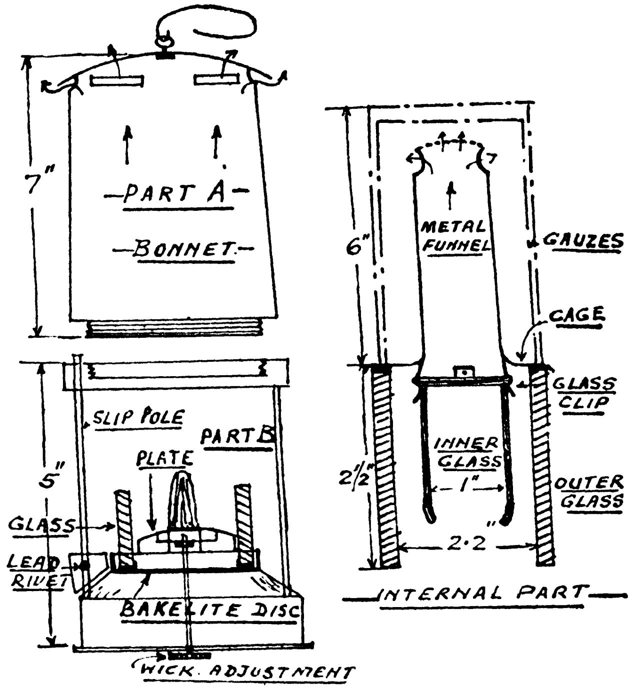

FIG. 79.—Protector Improved Combustion-tube Safety-lamp.

unscrewed when the slip pole drops after removing the lead rivet. The internal parts, comprising outer glass, gauzes and combustion tube, are then easily withdrawn for cleaning purposes. The part *B* includes the frame and oil vessel.

The Protector lamp is of simple construction and can be dismantled easily for cleaning and examination, while at the same time it is robust and gives a good light. It burns mineral colza oil and has a fine screw-down wick adjustment for gas testing. The combustion-tube holder is of the clip type. Tilting of the lamp does not damage the glass or extinguish the flame. The lamp bottom is fitted with a patent diffusing and reflecting ignition plate which gives easy re-lighting of the wick. The feed air enters the lamp at the base of the bonnet and passes down between the two glasses to the flame. The inner gauze is 20-mesh, and the outer

28-mesh. This lamp is 12¼ inches high and weighs 3¾ lb. It gives 3 to 4 candle-power with a clear glass, 5 to 6 candle-power with a silvered glass, and 10 candle-power with an outside reflector.

Ventilation of Sinking Shafts

6. It is proposed to sink a pair of shafts, each 18 feet diameter, to a depth of 600 yards. Give full particulars of the plant you would install for ventilating these shafts during sinking. (30)

A. Fig. 80 shows the arrangement which might be adopted for

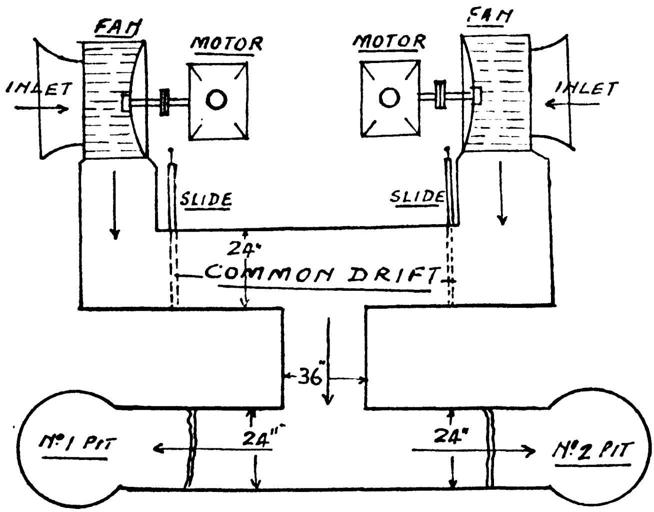

FIG. 80.—Ventilating a Pair of Sinking Pits.

ventilating a pair of deep sinking pits by means of duplicate fans. Each fan is of the *Sirocco* forcing type, 42 inches in diameter, single inlet, 600 revs. per minute, and operated by an electric motor. The quantity of air produced is 10,000 cubic feet per minute by each fan, with a water-gauge of 2 inches. The driving motor is 5 to 6 E.H.P. The air from the fans is delivered into a common drift, from which it is conducted to the respective pits by suitable branches. In the event of one fan breaking down, the other fan would continue to ventilate both pits, suitable sliding doors being fitted in the common drift to isolate the idle fan from the system. One fan might be sufficient to ventilate the pits until sinking has reached a depth of 300 yards.

Fig. 81 shows the method of conducting the air down the shafts by means of green canvas tubes 24 inches in diameter and in 50-foot lengths. The joint in the tubes is made by using a galvanised sheet-steel socket 15 inches long, having beaded edges. The ends of the tubes are put over the beaded edges and are then secured by straps

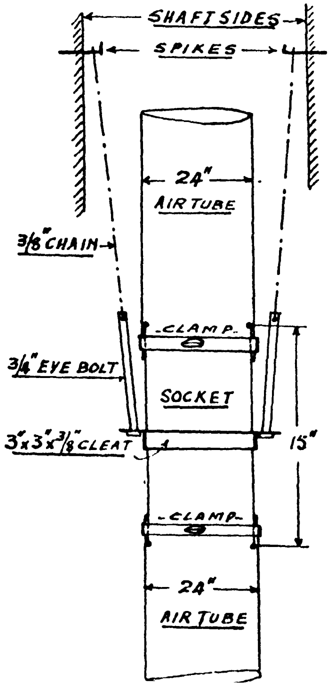

FIG. 81.—Securing Ventilating Pipes in a Sinking Pit.

and clamps. The sockets are secured to the shaft sides by cleats, chains and spikes as shown in the sketch.

By this method, and the use of a length of flexible hose at the bottom of the pipe-line, a good current of air is obtained at the shaft bottom to displace firedamp and shot-firing fumes.

Coefficient of Friction in Underground Roads

7. What is meant by "Coefficient of friction of ventilation"? Give its value as determined by Atkinson. Discuss the variations proposed by recent authorities, giving their views. Also state what,

in your opinion, can be done in practice to reduce this factor to its lowest possible value. (30)

A. The term "Coefficient of friction" has already been dealt with : see pages 114–15.

The value given by Atkinson is 0·0217 lb. per sq. foot of rubbing surface for a velocity of 1,000 feet per minute. This figure was determined by practical experiments in mines over 50 years ago, when the air passages were small, irregular in shape, twisted, and of short length. It is not suitable for the conditions existing at the present day with large airways, steel supports, better designed workings, and better return airways.

Recent Figures for Variations in Coefficient :—

Murgue :	0·00175 to 0·00275 for arched roadways.	
	0·005 for unlined roads.	
	0·008 ,, lined roads.	
Fairley :	0·01 ,, timbered roads.	
Cooke and Statham :	0·004 ,, arched roads.	
	0·01 ,, timbered roads.	

The views expressed in giving the above figures include the following. The coefficient depends greatly on the nature of the rubbing surface of the airways ; sometimes this is smooth owing to improved linings, and in some instances it is rough and unlined. The value depends upon the method of erecting roof supports, especially timbers, and the method of building roadside packs. Bends in airways produce eddy currents and turbulent motion of the air, and these must also be taken into account. Restrictions in roadways caused by falls of roof and sides, brattices, hutches or tubs, manholes, and road crossings which alter the cross-section of roads, also greatly affect the coefficient.

Reducing the Coefficient of Friction :—

(i) The use of circular shafts with wire rope guides and no buntons reduces friction.

(ii) Have main roads of large and uniform area, with steel supports, smooth sides, and as few bends as possible.

(iii) Have ventilating splits of uniform area and length, if possible.

(iv) The face workings should have uniform air passage, the packs should be built in a uniform way, restrictions in area avoided, brattices arranged to give uniform areas, and tubs should not remain in small areas for long periods.

(v) Return airways should be supported by steel supports or packs to avoid falls caused by rotting timbers. The same precautions should be taken as with main roads.

(vi) The upcast shaft should be arranged in the same way as the downcast to reduce friction.

(vii) The fan drift should be smooth-lined and of ample area.

(viii) The ventilating fan should have a good manometric efficiency and a low internal resistance.

NOVEMBER 1933 EXAMINATION

Ventilation of Workings

1. The accompanying plan (see page 132), is that of a colliery working a fiery seam and employing about 400 workers in the principal shift. Coal is wound only in the downcast shaft. Show how you would arrange the ventilation, marking the direction of each air-current and the position of all air-crossings, doors, stoppings, regulators and sheets. Use the signs laid down in the Regulations, and colour the path of the intake air blue and that of the return red. (50)

A. Fig. 82 shows the plan referred to in the above question. All details of the method of ventilating are shown on it. There are four splits and four air-crossings. Regulators are not required.

Choice of Safety-Lamps for Fiery Colliery

2. Assume that you are in charge of a fiery colliery and have to select safety-lamps for use therein. What considerations would you take into account in making your choice ? What lamps would you select ? And in the case of a flame safety-lamp, describe its construction and state the precautions necessary in assembling each component part, so that, when complete, the lamp shall be safe for use in the mine. (30)

A. If the seam to be worked is not subject to sudden outbursts of firedamp, and is worked longwall, electric lamps of the hand or cap type might be used, with the addition of one flame safety-lamp for every eight workmen employed at the face. In the event of the seam being subject to outbursts of firedamp and worked by driving narrow roads, as in bord-and-pillar workings, flame safety-lamps only should be used at the working face. Electric hand or cap

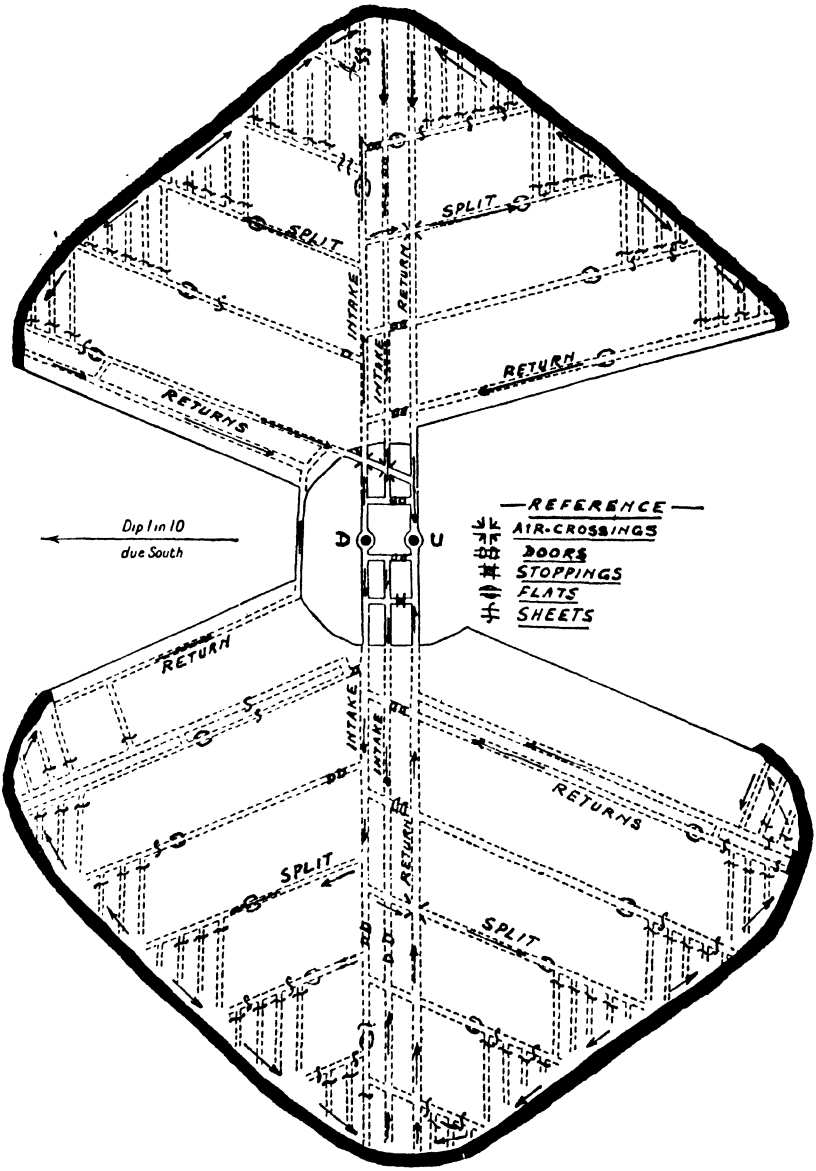

FIG. 82.—Ventilation of Longwall Workings.

lamps could be used in all other parts of the mine to within 100 yards of the working faces.

The flame safety-lamp selected should satisfy the following conditions :—

(i) Be simple but strong in construction.

(ii) Be easily dismantled, examined and cleaned.

(iii) Be fitted with two gauzes, a good glass, and an efficient magnetic lock.

(iv) Have a reliable re-lighting arrangement without unlocking the lamp.

(v) Show good firedamp caps and be safe in high percentages of firedamp.

(vi) Show a candle-power of at least 4 during the whole shift.

A safety-lamp suitable for a fiery mine and fulfilling the above conditions has already been described; see pages 127–8 and Fig. 79.

Precautions necessary in assembling a safety-lamp should include the following :—

(i) The bonnet should be closely examined for punctures after cleaning.

(ii) The gauzes should be properly brushed clean and then examined for broken wires before being placed inside the bonnet.

(iii) The glass should be cleaned and inserted into the lamp with washers of asbestos at the top and bottom after being examined for cracks and chips. The glass should be fitted into the lamp by a ring screwed up tight into the frame. Finally, this part of the lamp should be tested for the proper fitting of the glass by the use of a " Universal Tester " in the lamp-room.

(iv) The lamp bottom should be charged with oil, then screwed into the lamp frame, and securely locked by a magnetic lock.

(v) The wick should be lighted by a re-lighting apparatus, and the lamp then tested for safety in an explosive atmosphere, by means of a " Universal Tester," before being handed out of the lamp-room.

Gases met with in Mines—the Atmosphere

3. State the composition of ordinary atmospheric air (omitting argon and other rare gases) and that of the gases met with in mines, giving the properties and characteristics of each component. In dealing with these gases, state the precautions necessary to ensure safety. What limits are placed to the impurities allowable in the return air of a mine ? (30)

A. *Composition of Atmosphere :*

Oxygen	20·9 per cent. or $\frac{1}{5}$ by volume.
Nitrogen	78·0 " " $\frac{4}{5}$ " "
Carbon dioxide	0·03 to 0·04 per cent.
Water vapour	up to 1 per cent.

Oxygen has neither colour, taste nor smell, and it is the supporter of all life and combustion. It is a chemically active gas, and has a density of 16, taking hydrogen as unity. It is contained in the compound gases CO and CO_2 as well as in the majority of explosive compounds. The standard of oxygen in mine air must not be less than 19 per cent. to satisfy the conditions of the Coal Mines Act. Air containing only 17 per cent. of oxygen causes lights to be extinguished, whilst air with 14 to 15 per cent. of oxygen has serious effects on workmen in a mine. Air with only 2 per cent. of oxygen causes unconsciousness.

Nitrogen is contained in the atmosphere in mixture with oxygen. It has no colour, taste or smell, and will not support life or combustion. Its function in the atmosphere is to correct the volume of oxygen taken into the lungs at each inhalation. The density of nitrogen is 14, taking hydrogen as unity. It is a very inert gas and not chemically active like oxygen.

The other gases found in a mine have already been described ; see pages 99–101.

In dealing with mine gases in general, it is necessary to circulate a good volume of air in the mine to render them harmless and to carry them out of the mine by the upcast shaft. In this way ignitions of firedamp by lights, shot-firing, electric plant, etc., will be greatly reduced, and the danger of explosions of the gas and of coal-dust will also be reduced. The danger of workmen being suffocated in high percentages of firedamp should not arise in well-ventilated mines. Good supervision, by well-qualified officials, must not be left out of account in dealing with firedamp in a mine to prevent accidents. *Blackdamp* requires special mention as frequently being met with in mines, especially shallow mines, and for safe working conditions a good air-current must be circulated. Men should be withdrawn from all parts of a mine where lights burn badly.

Carbon dioxide (CO_2) is met with at pump sumps and where heating is going on. Frequent treatment of the sumps with lime and fresh water is advisable in the first case, and good ventilation in the mine roads is necessary in the latter case to ensure safe working conditions. *Carbon monoxide* (CO) is not often met with in mines, except where heating has developed. Its presence is detected by using small cage-birds and mice, and by analysis of the air-current. *Sulphuretted hydrogen* (H_2S) is seldom met with in mines in dangerous quantities. It is easily detected by its disagreeable smell, and though very poisonous is seldom a source of danger. Special care is necessary when examining old workings, where this gas is usually found.

The percentage of firedamp in the return airways of a naked-light mine must not exceed 0·5 per cent.; if the percentage is under 0·5 the return can be used for haulage purposes. In the case of a safety-lamp mine the percentage of firedamp should not exceed 1, no figure being stated by the Coal Mines Act.

The percentages of blackdamp and of CO_2, respectively, should not exceed 7·5 and 1·25 in the return airways of any mine, the latter figure only being stated by the Coal Mines Act.

Ventilation of Cross-Measure Drifts

4. It is proposed to drive a pair of long parallel cross-measure drifts in a fiery colliery. State what plant you would install for ventilating the drifts, and how you would arrange to drive the plant, giving sizes and arrangements and quantity of air. State the advantages and disadvantages of making air-tubes carry (*a*) the intake air to, and (*b*) the return air from, the face of the drifts. (30)

A. Fig. 83 shows the general arrangement of plant for ventilating a pair of cross-measure drifts in a fiery colliery. The drifts are 6 feet high by 10 feet wide and are ventilated separately by turbine blowers marked *A* and *B*, each circulating 2,000 cubic feet of air per minute. A connection between the drifts should be made at convenient spots, at say every 100 yards. The return air from one of the drifts is piped to the main return airway, so as to keep the drifts and connecting place free from firedamp. All necessary details are included on the plan.

The inset shows the arrangement of the blowing fan, which is driven by a compressed-air turbine. The turbine wheel and the fan blades are mounted on the central shaft and rotate as one unit, the air driving the turbine wheel and the fan forcing the air into the drift. The compressed air escaping from the turbines mixes with the air circulating in the respective drifts. After the drifts are advanced 100 yards, and are connected, one of them becomes a return airway and the other an intake airway.

Air-tubes as the Intake.—In this case the drifts are charged with

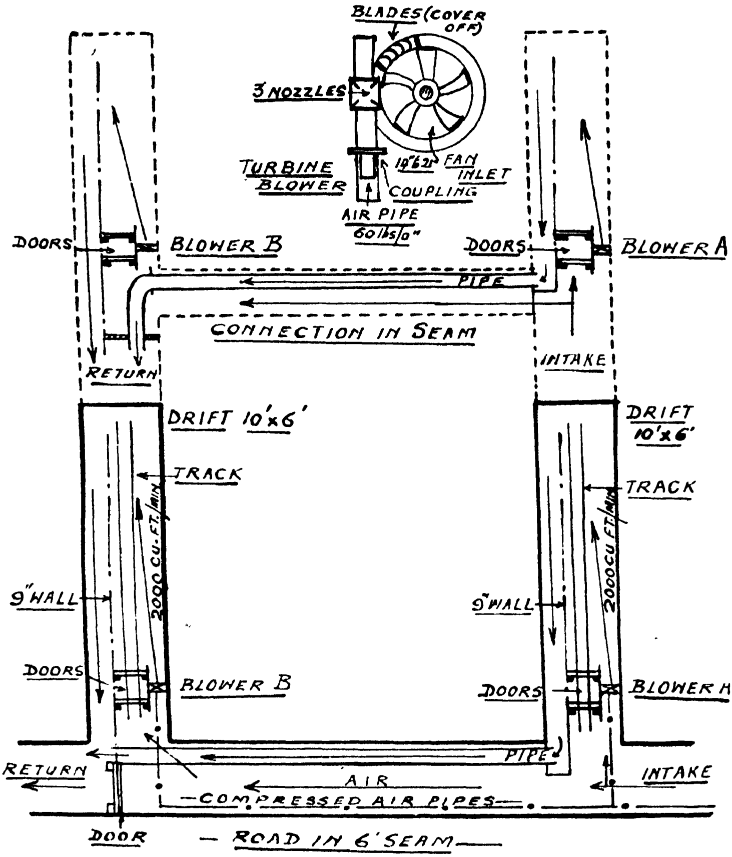

FIG. 83.—Ventilation of Cross-measure Drifts.

return air and they might contain dangerous percentages of firedamp, thus producing dangerous conditions for the workmen engaged in them, and a greater risk of explosion. Firedamp might accumulate at the faces and give much trouble, while at the same time the work at the face would be irregular, and shot-firing would be dangerous.

Air-tubes as the Return.—With this arrangement the drifts are charged with intake air free from firedamp, and dangerous conditions, as in the previous case, would not exist. Such an arrangement would be safer for shot-firing at the faces, and regular work would be obtained. Any firedamp given off from the face strata would pass direct to the return airway by means of the tubes.

Air-Crossings in Mines

5. In what circumstances do you consider it advisable to construct an air-crossing of a temporary character, and in what circumstances should the structure be of a more permanent type ? Sketch and describe the construction of a permanent air-crossing, and give an account of the operations necessary, in proper sequence, in bringing it into use for the first time. (30)

A. *Temporary air-crossings* might be installed in a pit to overcome some difficulty of a temporary character, such as :—

(i) Where narrow headings are advancing beyond the general line of working face, and where large volumes of firedamp are liberated from a blower, a fault, or through a sudden change in the character of the seam being worked and its associated strata. Under such conditions it might be necessary to ventilate the faces by separate air-currents, in order to keep the roads free from firedamp and to have safe working conditions with a regular advance forward. In this case, temporary air-crossings might be installed near the face in preference to ventilation by pipe-lines.

(ii) A temporary air-crossing might be installed near the working faces, in preference to a permanent structure, where the strata have not completely settled down, and there is a danger of damage to a permanent crossing.

(iii) In restoring the ventilation after an explosion, temporary crossings might be applied until the damaged permanent crossings are properly repaired or renewed.

Permanent air-crossings are always preferable to temporary crossings in a mine where the roof strata have completely settled down, where the position is a reasonable distance back from the working faces, and when the crossing will be required for a long period in connection with the ventilation of the mine.

The construction of a permanent air-crossing has already been described ; see pages 95–6 and Fig. 64.

When an air-crossing is completed and ready for use, the arrangements necessary to bring it into commission are as follows :—

(i) A large-scale plan of the ventilating system should have been prepared to include all roads and all ventilating appliances.

(ii) A proper design in red should be made on the plan, to bring the air-crossing into commission and to show all the necessary alterations in the system.

(iii) The work in connection with the change-over should be prepared under the supervision of a responsible official, who would report to the Manager when the work was complete, so that the details could be checked over by the Manager or Under-Manager.

(iv) The change-over should be properly organised by the management, and the work undertaken at the week-end when the pit is not in active working, officials and workmen being posted at all points where alterations are required. The work at each point must start at a specified time.

(v) After the change-over is completed, the districts affected should be properly examined and a report handed to the Manager. The air-current circulating in each district should be measured.

Calculations of Areas and Perimeters of Roadways

6. Compare the areas and perimeters of two main roadways—one lined throughout with timber settings, 7 feet high, 7 feet wide at the top, and 10 feet wide at the floor—all in the clear ; the other lined with standard steel girders of the splayed type, 8 feet high, 8 feet wide at springings, and 9 feet 2 inches wide at floor. With a velocity of 600 feet per minute, calculate the quantity of air passing per minute in each road. Discuss the question of the power required to overcome the resistance of each of these roadways, giving your opinion as to the proportionate resistance in each case, assuming an equal length of, say, 800 yards ; and state the grounds on which you base your opinion. (30)

A. (*a*) *Timbered Roadway* :—

$$\text{Area of road} = \text{Average width} \times \text{height}$$

$$= \frac{7 + 10}{2} \times 7 = \underline{60 \text{ sq. ft.}} \qquad (1)$$

$$\text{Perimeter of road} = 7 + 10 + 7 + 7 = \underline{31 \text{ feet}} \qquad (2)$$

$$\text{Quantity of air} = \text{Area of road} \times \text{Velocity per minute}$$

$$= 60 \times 600 = \underline{36{,}000 \text{ cu. ft. per min.}} \qquad (3)$$

$$\text{Resistance of roadway} = \frac{cplv^2}{a} \text{ (see p. 113).}$$

$$= \frac{0{\cdot}01 \times 31 \times 2400 \times \left(\frac{36{,}000}{60{,}000}\right)^2}{60}$$

$$= \underline{4{\cdot}46 \text{ lb. per sq. foot}} \qquad (4)$$

$$\text{Power required} = \frac{\text{Quantity} \times \text{Resistance}}{33{,}000}$$

$$= \frac{36{,}000 \times 4{\cdot}46}{33{,}000} = \underline{4{\cdot}86 \text{ H.P.}} \qquad (5)$$

(*b*) *Steel-lined Roadway* :—

$$\text{Area of road} = ⏢\ \text{part} + ⌓\ \text{part}$$

$$= \left(\frac{8 + 9\frac{1}{6}}{2}\right) \times 4 + \frac{8^2 \times \frac{11}{14}}{2}$$

$$= 34\tfrac{1}{3} + 25\tfrac{1}{7}$$

$$= \underline{60 \text{ sq. ft.}} \text{ (almost)} \quad . \quad . \quad . \quad (1)$$

$$\text{Perimeter of road} = ⏢\ \text{part} + ⌒\ \text{part}$$

$$= 9\tfrac{1}{6} + 4 + 4 + \frac{8 \times \frac{22}{7}}{2}$$

$$= 17 \text{ ft. } 2 \text{ in.} + 12 \text{ ft. } 7 \text{ in.} = \underline{29 \text{ ft. } 9 \text{ in.}} \quad (2)$$

$$\text{Quantity of air} = \text{Area of road} \times \text{Velocity per minute}$$

$$= 60 \times 600 = \underline{36{,}000 \text{ cu. ft. per min.}} \quad (3)$$

$$\text{Resistance of roadway} = \frac{cplv^2}{a}$$

$$= \frac{0{\cdot}002 \times 29\frac{3}{4} \times 2400 \times \left(\frac{36{,}000}{60{,}000}\right)^2}{60}$$

$$= \underline{0{\cdot}857 \text{ lb. per sq. foot}} \quad . \quad . \quad . \quad (4)$$

$$\text{Power required} = \frac{\text{Quantity} \times \text{Resistance}}{33{,}000}$$

$$= \frac{36{,}000 \times 0{\cdot}857}{33{,}000} = \underline{0{\cdot}935 \text{ H.P.}} \quad . \quad (5)$$

The resistance of the arched road is considerably less than that of the timbered road, being in the ratio of 1 to 5. The reason for this is that the surface of the arched road is of a smooth nature as compared with that of the timbered road. The coefficient of friction for the arched road is therefore much less than that for the timbered road, being in the ratio of 0·002 to 0·01 or 1 to 5. The power required to overcome the resistance also varies with the nature of the rubbing surface and the coefficient of friction. The lined roadway requires only $\frac{1}{5}$ of the power necessary for the timbered road.

Calculation of Weight of Air

7. How would you proceed to calculate the weight of 1 cubic foot of atmospheric air under ordinary conditions? State exactly what conditions you assume for this purpose. Calculate the effect of the increase of temperature observed in passing from the downcast at 50° F. to the upcast at 74° F. respectively. Calculate also

the effect produced by a change of atmospheric pressure of 1-inch increase on that assumed by you for the first part of your answer. What is the effect of the presence of a high percentage of moisture in the air? (30)

A. By experiment the weight of 459 cubic feet of atmospheric air at a pressure of 30 inches barometer and at 0° F. = 39·76 lb. when dry.

∴ Weight of 459 cubic feet at 1 inch barometer and 0° F. = 1·3253 lb. when dry.

∴ Weight of 1 cubic foot of air

$$= \frac{1{\cdot}3253 \times \text{barometer in inches}}{459 + \text{temp. } ^\circ\text{F.}} \quad . \quad . \quad . \quad (1)$$

Ordinary conditions may be taken as represented by 30 inches pressure and 60° F.

Weight of 1 cubic foot of downcast air

$$= \frac{1{\cdot}3253 \times 30 \text{ (assumed)}}{459 + 50} = \underline{0{\cdot}0781 \text{ lb.}} \quad . \quad . \quad (2)$$

Weight of 1 cubic foot of upcast air

$$= \frac{1{\cdot}3253 \times 30 \text{ (assumed)}}{459 + 74} = \underline{0{\cdot}0746 \text{ lb.}} \quad . \quad . \quad (3)$$

Weight of 1 cubic foot of downcast air with 1-inch increase in pressure

$$= \frac{0{\cdot}0781 \times 31}{30} = \underline{0{\cdot}0807 \text{ lb.}} \quad . \quad . \quad . \quad . \quad (4)$$

Weight of 1 cubic foot of upcast air with 1-inch increase in pressure

$$= \frac{0{\cdot}0746 \times 31}{30} = \underline{0{\cdot}0771 \text{ lb.}} \quad . \quad . \quad . \quad . \quad (5)$$

(Note that in the latter two results the weight varies directly with the pressure.)

Effect of Moisture.—Moist air has a lower density than dry air, and in this way produces a smaller weight per cubic foot.

(Mass of 1 litre of aqueous vapour = $\frac{5}{8}$ the mass of 1 litre of dry air.)

Weight of saturated air per cubic foot

$$= \frac{1{\cdot}3253 \times (\text{barometer} - \frac{3}{8}f)}{459 + t^\circ \text{ F.}} \quad . \quad . \quad . \quad (1)$$

(f = vapour pressure in inches of mercury = 0·36 at 50° F. and 0·90 at 74° F.)

Weight of 1 cubic foot of air at 30 inches barometer and 50° F. when saturated

$$= \frac{1{\cdot}3253 \times (30 - \frac{3}{8} \text{ of } 0{\cdot}36)}{459 + 50}$$

$$= \frac{1{\cdot}3253 \times 29{\cdot}865}{509}$$

$$= \underline{0{\cdot}0778 \text{ lb.}} \qquad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

(This result is less than the 0·0781 as already calculated for dry air.)

Weight of 1 cubic foot of air at 30 inches barometer and 74° F. when saturated

$$= \frac{1{\cdot}3253 \times (30 - \frac{3}{8} \text{ of } 0{\cdot}9)}{459 + 74}$$

$$= \frac{1{\cdot}3253 \times 29{\cdot}662}{533}$$

$$= \underline{0{\cdot}07375 \text{ lb.}} \qquad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

(This result is again less than 0·0746 as already calculated for dry air.)

MAY 1934 EXAMINATION

Combustion-Tube Safety-Lamps—Improvements

1. Describe a Combustion-tube Safety-lamp of the type largely used three years ago; and then give an account of improvements introduced into this type recently with the object of giving more light, showing how this has been achieved. (30)

A. Fig. 84 shows the construction and arrangement of the *Hailwood Combustion-tube Safety-lamp* of about 1931 pattern. The combustion tube is suspended by a cage arrangement from the top of the outer glass, and it is made up of a glass of small size, above which is fitted a metal funnel. The feed air enters the lamp at the base of the bonnet and is deflected down between the two glasses to the flame. Other details are marked on the sketch.

Fig. 85 shows the recently introduced *Hailwood ADC*3 *high candle-power safety-lamp* of the Schedule A type, which embodies the following improvements :—

(i) A better light of 4 to 5 candle-power.

(ii) Gauzes of 20-mesh are fitted instead of 28-mesh.

(iii) A floating combustion tube is arranged to seat properly on the lamp dome, and an internal reflector is fitted on the inner glass.

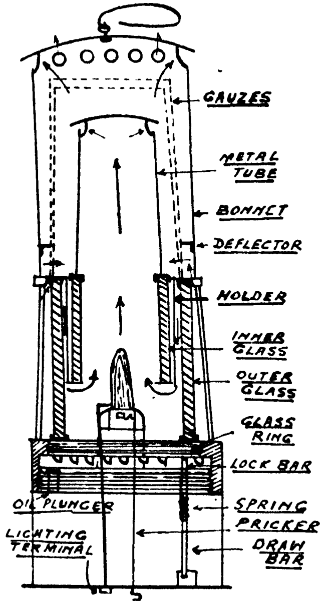

FIG. 84.—Hailwood Combustion-tube Safety-lamp.

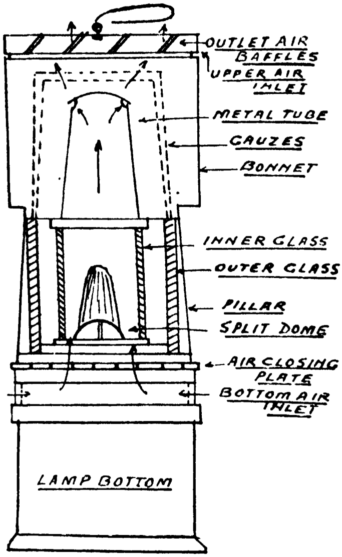

FIG. 85.—Hailwood Improved Combustion-tube Safety-lamp.

(iv) The light and gas caps are not obscured by the framework of the lamp.

(v) A split dome is included, with the front half arranged for lowering when testing for gas.

(vi) The bottom air inlet is regulated by a screw-action plate to give the maximum light and to prevent the flame being extinguished in high velocities.

(vii) By partly closing the bottom air inlet, the feed air enters the lamp by the top inlet near the roof and enables more accurate gas-testing to be carried out.

(viii) The outlets at the lamp top are of the guarded non-return type.

(ix) The lamp is 10½ inches high and weighs only 5 lb.

The lighting power of the Hailwood lamp has been improved by using light mineral colza oil and by introducing an enclosed burner instead of an open one, to give a wide-spreading flame ; by introducing the feed air at the base of the lamp and making provision for regulating the same ; by using 20-mesh gauzes instead of 28-mesh ; and by fitting a reflector to the lamp glass.

Note: In a recent model, the air-chamber at the base has been dispensed with.

Main Ventilating Fans

2. Describe, with sketches, a main ventilating fan of the centrifugal type. State what other types of fan have been recently introduced for ventilation purposes, and enumerate their advantages or disadvantages as compared with the first-named type. (30)

A. The first part of the above question has been answered in dealing with previous questions; see Figs. 66 and 73.

During recent years, the *axial-flow fan* has come into use as a means of ventilating mines. The broad principle of these fans is that of the aircraft propeller, hence the earlier designs, such as the Steart fan shown in Fig. 58, were very noisy in operation. While this defect has not been wholly eliminated it has been appreciably reduced. The reduction in noise and vibration has been brought about by modification of the blade form and pitch, thus permitting a lower tip speed to be used for a given water-gauge.

A modern development on the above lines is the *Aerex* fan, constructed by Messrs. Walker Bros. of Wigan, which has been installed during recent years in British mines to work with single rotor or multiple rotors in stages. Such a fan has rotors and guide vanes made of a special alloy (silicon–aluminium) to give lightness and increased strength. The fan shaft is made of forged steel and is fitted with roller bearings. The fan drift and fan are arranged for parallel air-flow, while the former is concrete-lined and provision is made for removable roof, hole for water-gauge connection, and cleaning facilities in the base. The following are some of the details of a recent installation :—

Single-stage fan, 75 ins. diam. with housing 42 ins. diam. in a drift 81 ins. diam.; 103,000 cu. ft. of air per min. with $4\frac{1}{4}$ ins. total water-gauge at a speed of 850 revs. per min.; 84 per cent. fan efficiency, 88 per cent. motor efficiency, 98 per cent. texrope drive efficiency, 90 E.H.P. of motor, 65·6 H.P. in air, 73 per cent. useful effect.

The *Aeroto* fan, constructed by Messrs. Davidson of Belfast, is another type of modern fan made on the same lines as the above and shown in Fig. 86. In this fan the lower tip speed, and the use of the silicon–aluminium alloy with a high strength–weight factor, are together responsible for a very substantial reduction in the stresses induced in the blades by centrifugal force. Each stage of the fan has a separate casing, while distance-pieces between casings provide access to the fan rotors. Twin roller-bearings carry the shaft, one of which takes the thrust. The fan blades are fitted on a cast-steel hub, and the guide vanes of cast iron are of aerofoil

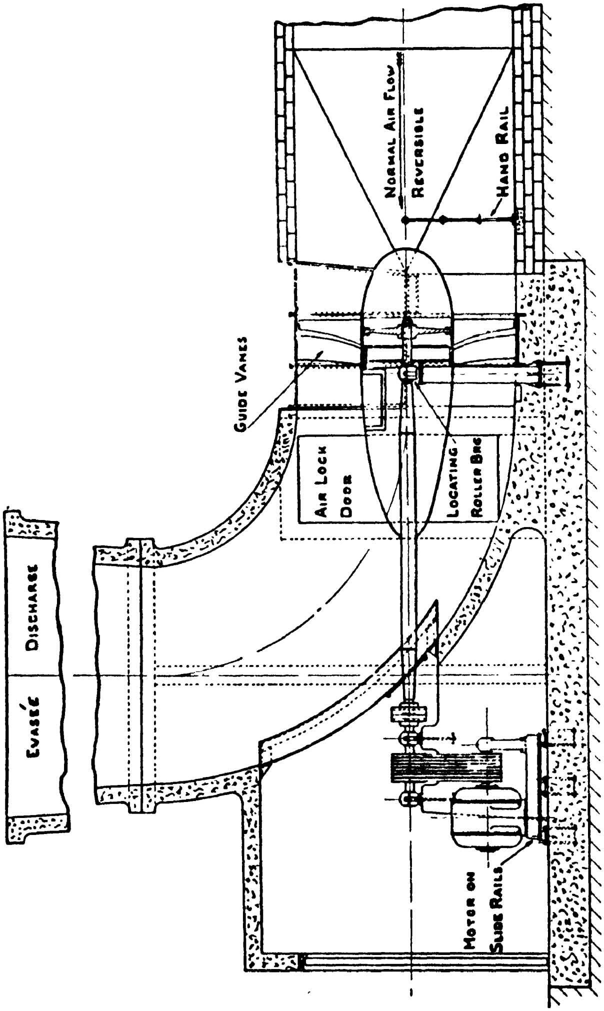

Fig. 86.—Sirocco Aeroto Single-stage Ventilating Fan.

section and are carried on a steel centre ring. All cross-girders in the air-flow are streamlined to offer the minimum of resistance, while fairings fitted at each end of the hubs considerably reduce the central vortex formed by the hubs of the rotors. The following details are of recent erections.

(a) 17·5 ins. diam. fan, 5,000 cu. ft. per min. through 22 ins. diam. duct at 2,880 revs. per min.

(b) 85 ins. diam. fan, 122,000 cu. ft. per min. with 2 ins. of water-gauge at 530 revs. per min.

(c) 165 ins. diam. fan. of three stages exhausting 500,000 cu. ft. of air per min. with 11¼ ins. water-gauge when running at 275 revs. per min. in a fan drift 18 feet square. Each rotor has 20 blades and there are 21 guide vanes between rotors. The fan is driven by an induction motor of 1,000 B.H.P. running at 325 revs. per min., the speed reduction being accomplished by 14 to 25 V-ropes of Dixel Balata type. This plant is 60 feet long by 35 feet high and has a total weight of 60 tons.

The advantages of axial-flow fans as compared with centrifugal fans are as follows:—

(i) Lower installation cost, 40 to 60 per cent. reduction in weight, less space occupied, lower power consumption, and a higher mechanical efficiency of 75 to 85 per cent. over a wide range of duties.

(ii) Simple in construction, robust and smooth-running with less vibration and noise, lower maintenance cost.

(iii) Reversal of the air-current is accomplished by changing the direction of rotation of the driving motor, giving 75 per cent. power and 80 per cent. air-flow.

(iv) For small duties the fan and driving motor are combined.

The disadvantages experienced include air-slip, danger of breakages at high speeds, and maintenance of bearings on the considerable length of shaft.

Temperature of the Air in Deep Mines

3. In deep and hot mines, to what causes do you attribute the high temperature; what effects are observed on the workmen's health; and what precautions and arrangements are desirable in order to reduce these troubles? Suppose the workings in a seam of coal are fairly flat and lie at a depth of 2,400 feet, make an estimate of the temperature of the air at, say, two miles inbye, showing how you arrive at your figure. (30)

A. High temperatures in deep mines are attributed to the increase in temperature of the strata, the geothermic gradient being 1° F. for every 70 feet of depth. Heat is also caused in a mine by the oxidation of coal and strata, by compression and friction of the air-currents, by machinery, and by the grinding of strata causing friction. A certain amount of heat is also produced by men working, by horses, and by lamps. If the ventilation in a mine is inadequate, the heat generated in the mine becomes excessive.

The health of the workmen is affected by heavy perspiration in high temperatures, this causing loss of salt from the blood and gradual weakening. Healthy perspiration is retarded in mines by high wet-bulb temperatures and the workmen are thus more easily fatigued. A wet-bulb temperature of 80° F. is considered to be the maximum for mine workers, while continuous hard work is almost impossible at 78° F. wet-bulb temperature.

Precautions should be taken to reduce the moisture content in the air-current, to keep down the temperature in the mine by circulating large volumes of air, and to have free circulation of the air in the workings to secure healthy respiration for the workmen. The arrangements necessary should include an adequate amount of circulating air by a reliable main ventilating fan, a standby duplicate fan, and the use of auxiliary fans for advanced parts of the workings. The air-current might be cooled on the downcast side by cold-water circulation, by ice, by the release of compressed air, or by a refrigerating plant. In this way the wet-bulb temperature might be reduced as much as 10° F. and the inlet air by as much as 30° F.

The temperature of the air at a depth of 2,400 feet, and at a point two miles inbye, would very nearly approach that of the strata. Assuming the strata to have a constant temperature of 50° F. at a depth of 50 feet, and the geothermic gradient to be 1° F. for each 70 feet of depth,

$$\begin{aligned}\text{Temperature at 2400 ft.} &= \frac{2400 - 50}{70} + 50 \\ &= 33\tfrac{1}{2} + 50 \\ &= \underline{83\tfrac{1}{2}^\circ \text{ F.}}\end{aligned}$$

Ventilating a Pair of Sinking Pits

4. Give a detailed account of the plant and appliances you would install for ventilating a pair of sinking pits 20 feet diameter, expected to reach a depth of 600 yards and passing through fiery coal measures. Show how to ensure a supply of fresh air at the bottom of the pits during sinking. (30)

A. The above question has already been answered in dealing with a previous question; see pages 128–9, Figs. 80 and 81.

If a feeder of gas is met with during sinking, it might necessitate the installation of a small fan of the forcing type to ventilate the feeder separately from the main plant described.

Air Regulation—Calculation of Quantity

5. Sketch a regulator capable of giving an area of 3 square feet for the passage of the current, and calculate the quantity you would expect to pass with water-gauges of 1 inch and 2 inches, respectively, applied to this area. Assume that the air weighs 0·08 lb. per cubic foot. (30)

A. Fig. 87 shows a front elevation of an air-current regulator of robust construction. The regulator frame is built in brickwork,

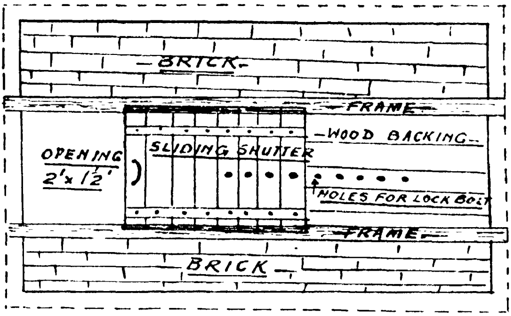

FIG. 87.—Air-current Regulator.

and the regulating shutter is fixed in the required position to a wood backing by means of a locking bolt. The keys required to adjust the opening are carried only by officials of the colliery.

Quantity of air passing with 1-inch water-gauge

$$= 0{\cdot}65A \times 60V \qquad [0{\cdot}65 = \text{Coefficient of the } \textit{Vena contracta},$$
$$= 0{\cdot}65A \times 60\sqrt{2gh} \qquad A = \text{Area of opening},$$
$$= 0{\cdot}65A \times 60 \times 8\sqrt{h} \qquad V = \text{Velocity per sec.},$$
$$= 312 \times A \times \sqrt{h} \qquad h = \text{Motive column in feet.}]$$

$$= 312 \times 3 \times \sqrt{\frac{1 \times 5{\cdot}2}{0{\cdot}08}}$$

$$= 936 \times 8{\cdot}06 = \underline{7{,}544 \text{ cu. ft. per min.}} \qquad (1)$$

Quantity of air passing with 2-inch water-gauge

$$= 936 \times \sqrt{\frac{2 \times 5{\cdot}2}{0{\cdot}08}}$$

$$= 936 \times 11{\cdot}4 = \underline{10{,}670 \text{ cu. ft. per min.}} \quad . \quad . \quad (2)$$

NOVEMBER 1934 EXAMINATION

Ventilation of Workings

1. On the given plan, show how you would arrange for the distribution of the air-currents, using the signs and colours as given in the Regulations. Sheets to be shown thus ⨍. The open working faces are represented by a thick line, roadways in coal pillar by solid lines, and the other roadways by dotted lines. The rest of the space inbye of the pit pillar is stowed. Coal is wound at downcast only. (50)

A. Fig. 88 shows the plan referred to in the above question. All the details of the ventilating system are included. Owing to the lack of return airways, it is necessary to arrange for replenishing air-currents in ventilating the faces *B* and *C*, which are advancing through the fault.

Characteristics of a New Ventilating Fan

2. What investigations would you conduct at a mine to obtain the information necessary to ascertain the characteristics required in a new ventilating fan, about to be installed, so as to ensure that the fan would be suitable for the conditions of the mine ? (30)

A. The investigations should include the determination of the total water-gauge of the mine, and the correct quantity of air passing in the mine with that water-gauge. The total water-gauge includes the water-gauge at the fan drift plus the water-gauge due to natural conditions. The former can be registered, and the latter calculated. From these details the correct area of equivalent orifice of the mine can be calculated, and this is the important factor which determines the dimensions and characteristics of a suitable ventilating fan.

Assuming the following data, the size of fan can be calculated as follows :—

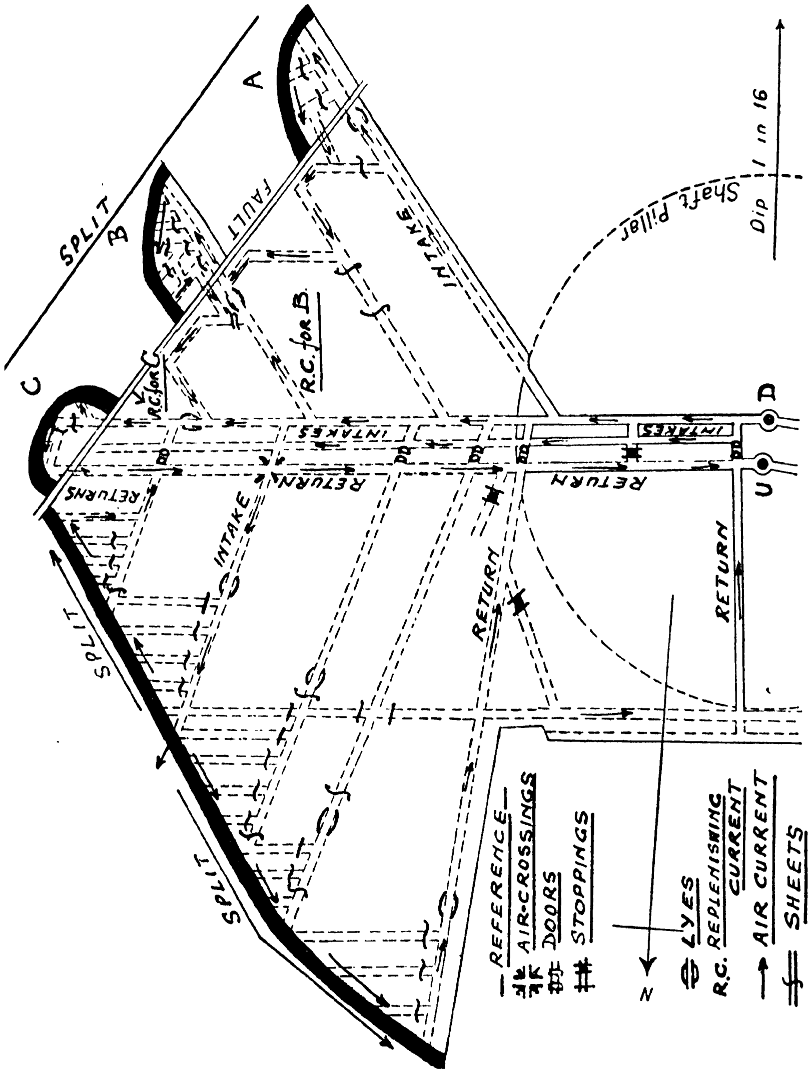

FIG. 88.—Ventilation of Longwall Workings.

Total water-gauge of mine, $2\frac{1}{2}$ inches.
Quantity of air circulating in the mine, 200,000 cu. ft. per min.
Density of downcast air, 0·08 lb. per cu. ft.
Manometric efficiency of fan, 80 per cent.
Useful effect of fan, 70 per cent.

Area of equivalent orifice of mine

$$= \frac{200,000}{312 \times \sqrt{\frac{2\frac{1}{2} \times 5{\cdot}2}{0{\cdot}08}}} = \text{approx. } 50 \text{ sq. ft.}$$

Theoretical water-gauge

$$= \frac{2\frac{1}{2}}{0{\cdot}8} = 3{\cdot}1 \text{ inches.}$$

Orifice of passage of fan

$$= \frac{200,000}{312 \times \sqrt{\frac{0{\cdot}6 \times 5{\cdot}2}{0{\cdot}08}}} = 103 \text{ sq. ft.}$$

Assuming a double-inlet Sirocco fan to be installed, the dimensions are as follows :—

Diameter of inlets

$$= \sqrt{\frac{103}{2} \times \frac{14}{11}} = \underline{8 \text{ feet}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

Diameter of fan $= \underline{8 \text{ feet}}$ (approximately) . . (2)

Length of fan blades

$$= \frac{8 \times 12}{20} = \underline{5 \text{ inches,}} \text{ with } 30° \text{ forward curve} \quad . \quad (3)$$

Width of fan

$$= 8 \times 0{\cdot}8 = 6\tfrac{1}{2} \text{ feet each side, or } \underline{13 \text{ feet total}} \quad . \quad (4)$$

To find the speed of the fan in revs. per min. :

$$V_T(V_T + V_R \tan 30°) \times 0{\cdot}00046 = 3{\cdot}1 \text{ inches of W.G.}$$

$$8\times\frac{22}{7}\times R\left(8\times\frac{22}{7}\times R + \frac{200,000 \div 60}{8\times\frac{22}{7}\times R\times 13}\times 0{\cdot}577\right) \times 0{\cdot}00046 = 3{\cdot}1$$

$$25R\left(25R + \frac{6}{R}\right) \times 0{\cdot}00046 = 3{\cdot}1$$

$$625R^2 + 150 = 6740$$

$$R^2 = \frac{6590}{625} = 10{\cdot}6$$

$$R = \sqrt{10{\cdot}6} = \underline{3{\cdot}25 \text{ revs. per sec.}} \text{ or } 195 \text{ revs. per min.} \quad (5)$$

$$\begin{aligned}
(V_T &= \text{Tangential velocity} \\
&= \text{Circ. of fan} \times \text{revs. per sec.} \\
V_R &= \text{Radial velocity} \\
&= \frac{\text{cu. ft. of air per sec.}}{\text{circ.} \times \text{width}} \\
R &= \text{Revs. per sec.})
\end{aligned}$$

Input of driving motor

$$= \frac{200{,}000 \times 2\tfrac{1}{2} \times 5{\cdot}2}{33{,}000} \times \frac{100}{70} = \underline{112 \text{ E.H.P.}} \quad . \quad (6)$$

Motor to be of the three-phase wound rotor type, 6 poles, 50 cycles, 960 revs. per min.; driving-wheel 2 feet in diameter, suitable for a rope-drive to the fan.

Gear ratio $\frac{960}{195}$ or $\frac{5}{1}$. Fan rope wheel $= 2 \times 5$, or 10 feet diameter.

Meaning of "Resistance" and "Perimeter" as used in Ventilation

3. What do you understand by the terms "resistance" and "perimeter" as used in ventilation? Compare the resistance of two roadways, each 1,000 yards long, one of them heavily timbered, 8 feet high, 6 feet wide near the roof and 9 feet wide at the floor; the other lined with standard steel arches, splayed type, 8 feet high, 8 feet wide at springings, and 9 feet wide at the floor, all measurements taken in the clear. If a quantity of 30,000 cu. ft. per min. is passing along each of these roads, calculate the air horse-power in each case, using your own estimate of the coefficient of friction. (30)

A. The term "resistance" in ventilation means the resistance to the passage of an air-current in a mine due to rubbing surfaces of the various airways, road linings, tubs, fittings, brattices, packs, bends in roads, road irregularities, and regulators.

The term "perimeter" in ventilation means the measurement round an airway, which includes the width at floor and roof, and the height of the sides in rectangular airways.

Let c = Coefficient of friction per 1,000 feet of air velocity per minute,
p = Perimeter of road, l = Length of road in feet,
v = Air velocity in thousands of feet per minute,
a = Cross-sectional area of roadway.

Then resistance of timbered roadway $= \dfrac{cplv^2}{a}$

$$= \frac{0{\cdot}01 \times 31 \times 3000 \times \left(\dfrac{30{,}000}{60{,}000}\right)^2}{60} = \underline{4 \text{ lb. per sq. foot}} \text{ of cross-sectional area} \quad . \ (1)$$

Resistance of steel-arched roadway

$$= \frac{0{\cdot}003 \times 30 \times 3000 \times \left(\dfrac{30{,}000}{59{,}000}\right)^2}{59} = \underline{1{\cdot}1 \text{ lb. per sq. foot}} \text{ of cross-sectional area} \quad . \ (2)$$

Air horse-power for timbered roadway

$$= \frac{30{,}000 \times 4}{33{,}000} = \underline{3{\cdot}6} \text{ almost} \quad . \quad . \quad . \ (3)$$

Air horse-power for steel-arched roadway

$$= \frac{30{,}000 \times 1{\cdot}1}{33{,}000} = \underline{1} \quad . \quad . \quad . \quad . \quad . \ (4)$$

Preventing Silica Dust in an Air-Current

4. In driving cross-measure drifts in strata containing a proportion of silica, state what arrangements you would adopt as to ventilation. Give an account of any special apparatus you would install to prevent the dust getting into the air. (30)

A. To facilitate the passage of dangerous dust from the face of a cross-measure drift, an auxiliary ventilating appliance might be installed. It comprises an exhausting device, consisting of two simple ejectors, operated from the compressed-air main. This appliance is fitted to a temporary brattice about 20 yards back from the face. In this way dust and fumes from shots are quickly removed from the vicinity of the face, and the workmen are comparatively free from the danger of silicosis without the use of dust traps.

To prevent the dust getting into the air, a dust trap could be applied to the drills when preparing the shot-holes. Fig. 89 shows the arrangement of the improved *Hay dust trap*. The suction hood is secured in position on the rock face by expanding two light steel tongues into the hole by means of a toggle motion. It is provided with two openings, one for the flexible tubing, and the

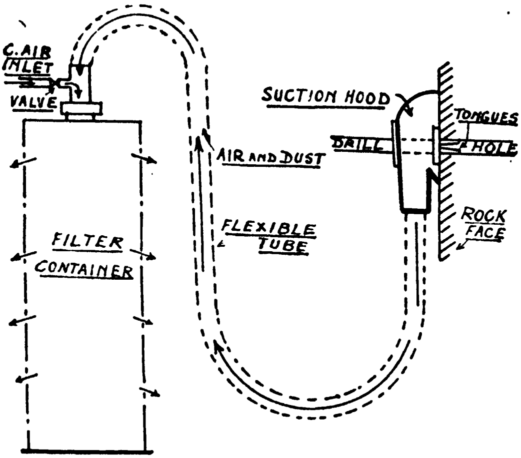

FIG. 89.—Improved Hay Dust Trap.

other for the drill. In starting to drill, the hood is pressed against the rock face by hand, the drill having been inserted through the back of the hood. After a depth of 2 inches has been bored, the mechanical holding device is put into action. The filtering appliance at the outer end of the flexible tubing comprises a fabric filtering medium enclosed in a perforated steel container, and an ejector-type suction producer operated from the compressed-air supply. The filter is designed to operate when drilling in either wet or dry ground.

JULY 1940 EXAMINATION

(*Six questions only to be answered.*)

Calculation of Quantity of Air passing in Airways of Given Size

1. An airway carrying 54,000 cu. ft. per min. of air divides at A into two airways that later come together at B and carry on as a single airway. Particulars of the two airways from A to B are :—

	No. 1.	No. 2.
Length in feet . . .	2000	2400
Area in sq. feet . .	42	64
Perimeter in feet . .	28	32
Coefficient of friction .	0·000000021	0·000000016 (velocity expressed in ft. per min.).

In what proportions would the air-current divide in the two airways ? (50)

A. Using the well-known equation relating to the passage of air in mine roads, which is stated below, problems of this kind can be solved quickly. Thus :—

$$ha = cplv^2, \quad \text{or} \quad ha^3 = cplq^2, \quad . \quad . \quad . \quad (1)$$

where h = Ventilating pressure in lbs. per sq. foot of sectional area of road.

a = Area of road in sq. feet.

c = Coefficient of friction for the road.

p = Perimeter in feet.

l = Length in feet.

v = Velocity of air-current in feet per minute.

q = Quantity of air circulating in cu. feet per minute.

The ventilating pressure at the junction of the splits is always the same for both splits, and assuming it to be the convenient figure of unity, an expression can be found for calculating relative quantities of air, and finally actual quantities.

The equation now becomes $a^3 = cplq^2$ . . . . (2)

and
$$q = \sqrt{\frac{a^3}{cpl}}, \text{ or } a\sqrt{\frac{a}{cpl}}$$

$\therefore$ Relative quantity for first split

$$= 42\sqrt{\frac{42}{21 \times 28 \times 2000}} \text{ or } 0{\cdot}251,$$

and Relative quantity for second split

$$= 64\sqrt{\frac{64}{16 \times 32 \times 2400}} \text{ or } 0{\cdot}462.$$

The total relative quantity is therefore $0{\cdot}251 + 0{\cdot}462$, or $0{\cdot}713$.

The actual quantity passing in No. 1 Split is therefore

$$54{,}000 \times \frac{251}{713}, \text{ or } \underline{19{,}000 \text{ cu. ft. per min.}}$$

and in No. 2 Split

$$54{,}000 \times \frac{462}{713}, \text{ or } \underline{35{,}000 \text{ cu. ft. per min.}}$$

Gases met with in Mines—Detection and Effects of Gas on Workmen

2. What gases are met with in the atmosphere of mines ? State how each is detected and what are the effects of each gas on the workmen. (30)

A. Answers to this question have been given in dealing with questions from previous papers. See pages 99–101 and 134 for mine gases, pages 104–6 and 110–11 for firedamp only, and pages 134–5 for carbon dioxide.

New Ventilating Plant—Details to give to Makers

3. Because of extending workings, the fan plant at a colliery is no longer adequate and you invite offers for a new ventilating plant, including the driving unit. What would you tell the makers so that they may offer suitable plant ? (30)

A. The first part of this question has been already dealt with in answers to similar questions ; see pages 148–151 dealing with the area of equivalent orifice of a mine and dimensions of fan suitable for such conditions. The possible limit of extension of the mine should be determined and sent on to the makers ; also the possible additional quantity of air desired for the ventilation of the pit. The makers should be fully informed about the motive power available at the pit. If steam only is available at the mine, the boiler pressure should be stated and details given of any auxiliary plant installed for dealing with exhaust steam. If electricity is also available at the mine, details as to D.C. or A.C., voltage, and cycles per second if A.C. should be given. It should also be stated which is to be the common driving unit and which the stand-by unit for driving the fan. Details of the drive required between the engine or motor and the fan should be given with those already referred to.

Mine Ventilating Fans

4. What method is employed to produce an air-current in a large mine ? What apparatus may be used for this purpose ? With the aid of a sketch, describe one type which is commonly employed. Give the important particulars, including arrangements for reversing the air. (30)

A. Answers to the above question have been already given in reference to similar questions, with details of Ventilating Fans ; see Figs. 58 for Steart, 66 for Sirocco, 73 for Capell, and 86 for Sirocco Aeroto.

Making a Ventilation Survey in a Mine

5. In connection with collieries, what is meant by a "Ventilation Survey"? How may a ventilation survey be carried out, and what kind of information is to be expected from it? (30)

A. A Ventilation Survey at a colliery means that certain important details relating to quality and quantity of the respective air circuits are recorded at definite places in the mine or district of the mine, so as to give the Colliery Manager a fairly accurate account of the existing standard of ventilation. Sometimes a pressure survey is carried out at the same time so as to give more detailed information. Surveys of this kind should be carried out at night or at week-ends when the pit is not drawing coal.

A *qualitative survey* is carried out daily by the underground officials of a colliery in examining the various districts, but a more reliable and detailed test might be desirable using a firedamp indicator, say of the McLuckie type, as described on pages 104–5. In this way the percentage of firedamp present in the air can be determined daily for one week at several selected spots of a district of workings.

A *quantitative survey* to obtain the quantity of air passing at selected spots can be made by using an Anemometer. The same procedure might be carried out by using a Velometer which would give instantaneous readings of air velocity at any point in the roadway. Quick readings obtained in this way are accurate for both high and low velocities. Points where quantity of air is measured should be marked on a prepared plan.

Pressure-drop surveys throughout a whole mine, or district of a mine, can be carried out by using a suitable liquid in an Inclined Manometer fixed to a suitable tripod. Lengths of rubber tubing 60 feet and 100 feet long, of ¼-inch diameter and with unions, can be used in connection with the manometer for surveying lengths of roadway. Static pressure can be measured by a special tube, and total pressure by a Pitot tube. Fig. 90 shows how a manometer and tubes might be applied. At the point *A* in the airway a static pressure tube is held at right angles to the direction of flow of the air-current. At *B*, with *A* closed, total pressures can be recorded by using a Pitot tube facing the direction of air flow; the pressure drop between the points *A* and *C* is therefore obtained. Pressure drops ascertained in this way might vary from 0·05 inch to 1·5 inches of water-gauge per 1,000-foot length of airway. Similar details to the above are noted at points *C* and *D*, to get the pressure drop

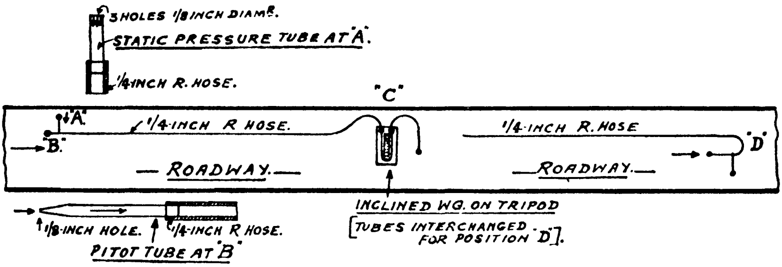

FIG. 90.—Manometer with Inclined Gauge and Tube for Measuring Pressures in Airway of a Mine.

between the points C and D. At all points where wood doors exist, the difference in pressure between intake and return should be obtained.

Where pressure-drop details are required over a wide area, the survey can be conducted at a quick rate by using a specially constructed Aneroid Barometer for the purpose. Such a survey becomes more complicated, owing to differences in level having to be taken into consideration and the introduction of specially-appointed persons to carry out the work. Fig. 91 gives diagram-

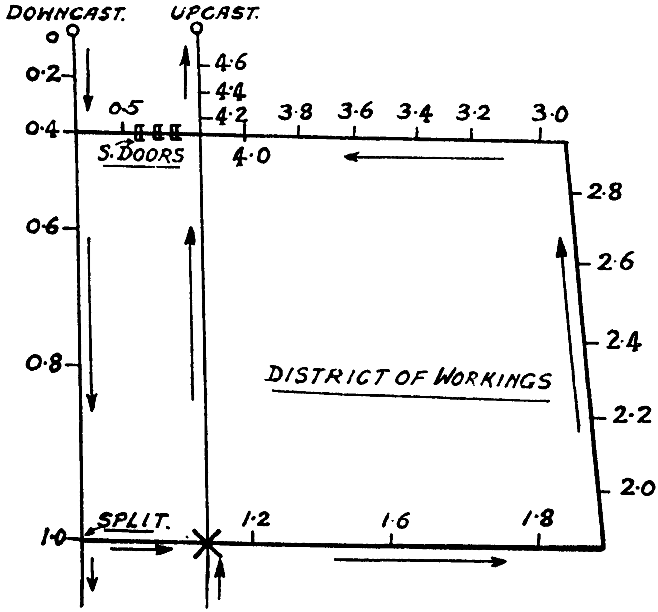

FIG. 91.—Sketch Plan showing Contour Method of indicating Pressure Drop. (Distances for pressure drop of 0·2 inch W.G.).

matically the contour method of showing pressure drop over a district of workings. Each fraction of an inch of water-gauge lost is marked off in the air-circuit. Close contours mean high resistance, while wide contours indicate low resistance.

Value of Ventilation Surveys.—The information obtained from a ventilation survey is very useful to the Colliery Manager in making readjustments in air circuits and, generally speaking, to maintain as much air as possible in the mine with the minimum expenditure of power. A survey of the qualitative kind, carried out as indicated above, would locate dangerous spots at faces and return airways which require attention. Some readjustment might be required in ventilating appliances to get more air to the seriously affected parts. Badly-fitted brattices and sheets might be the cause of the trouble.

By carrying out a quantitative survey, leakages of air at points between intake and return are indicated. Steps should be taken to carry out repairs to stoppings and wood doors, and also to attend to leakages through defective packing and breaks in strata.

When a pressure survey is carried out as suggested, valuable details are obtained regarding pressure losses, and these indicate what steps are required for increased efficiency, such as—

(*a*) Enlargement of airways may be recommended where a high resistance is recorded, say in return airways near the shafts.

(*b*) Bottle-necks in splits are indicated and can be removed.

(*c*) Air splitting delayed too long, thus giving insufficient main roads for adequate ventilation, can be remedied by driving new roads.

(*d*) Air splitting carried out beyond the economical limit at the faces can be rectified.

(*e*) Leakages of air from intakes to returns can be greatly reduced by repairs, as indicated above.

(*f*) Bad stowing of roads and badly-constructed brattices and sheets near the face can be attended to when the loss of pressure is located. Desirable standards of ventilation might include a loss of 0·3 inch water-gauge per 1,000 feet or 3 air horse-power per 1,000-foot length of road.

A combination of quantity and pressure surveying makes it possible to calculate power losses at various points ; thus abnormal power losses are easily detected. Corrections in the ventilating system, such as the details referred to above, result in the maximum quantity of air being circulated in the mine at a constant pressure, while the power required for circulating the air remains unchanged. A permanent cash saving is thereby made possible. Finally, the design of shafts and airways for a new colliery can be planned to give maximum efficiency in the ventilating system.

Considerations necessary for increasing Air-Current at the Face

6. It is considered desirable to increase the air-current passing along the face in a mine. State the factors that you would take into account in considering what should be done. (30)

A. A sluggish air-current at the face of a district in a mine might be caused by some defect in the ventilating system or by a fall of roof, and steps should be taken to investigate the cause with a view to improvement. A reliable official, having a good safety-lamp, a foot rule and a large-scale map of the district, should be detailed for this work. He would start investigation at the flat of the section, presuming that brick stoppings between intake and return are advanced to that part of the workings. All ventilating appliances, such as temporary stoppings, doors and screens, should be properly examined and their positions marked on the plan. The direction taken by the air-current should be closely followed and the size of airways noted, especially where restrictions are found. On reaching the working face, tests should be made for the presence of firedamp and the results noted. The state of the brattice (if any) and the width of air passage should also be noted. Packs should be examined for tightness, and bad points marked on the plan. The return airways from the face should be examined for percentage of firedamp and for the size of roads.

With a plan of this type and the booked details, the management should know where faults occur in the system, such as leakages of air-current and restrictions of roads. These faults should be attended to at once and the state of the air-current noted again by further examination. If the desired improvement does not materialise, a detailed quantity–pressure survey should be carried out in order to gain better information. Splitting of the air-current might be carried out with a view to reducing the length of single airway and increasing the quantity of air passing. As a last resort, it might be considered necessary to increase the ventilating pressure in the district so as to increase the quantity of air passing in the airways and faces. This could be accomplished by installing a small fan on the intake side, driven by electricity. If compressed air is in use, a number of jets of air in the circuit might be arranged to obtain the same end.

Safety-Lamp for a Fiery Mine

7. Describe the type of safety-lamp that you would adopt for a fiery mine, giving details. Would you adopt the same lamp for the officials? What proportion of flame safety-lamps would you adopt for a fiery mine where electric safety-lamps are used at the face? (30)

A. The first part of the question dealing with safety-lamps in fiery mines has been answered in dealing with previous questions of the same kind; see Figs. 60 for Wolf Alkaline Cap Lamp, 79 for Protector Combustion-tube Safety-lamp, and 85 for Hailwood Combustion-tube Safety-lamp.

If the Wolf Alkaline lamp be used, the officials might have the same type of lamp, but it would be necessary for them to be supplied with a Wolf small oil-flame safety-lamp fitted with re-lighting appliance, so that they could examine the mine for firedamp and blackdamp. Cap lamps often have a small firedamp-detecting apparatus incorporated in the lamp, and when this is supplied there is no need to issue safety-lamps of the flame type to the officials.

The proportions of flame safety-lamps required for fiery mines are as follows:—

1 safety-lamp to every 8 electric lamps in Longwall workings;
1 ,, ,, ,, ,, 4 ,, ,, ,, Pillar workings;
1 ,, ,, for each coal-cutting machine;
1 ,, ,, ,, ,, driller using an electric drill for shot-holes;
1 ,, ,, ,, ,, electric motor when automatic detectors are not in use.

JULY 1941 EXAMINATION

(*Six questions only to be answered.*)

Removing Firedamp from the Vicinity of a Conveyor Face

1. The given plan shows an advancing conveyor unit and two development roadways *AB* and *AC*. These roads have met a fault simultaneously and, as a result, have become filled with firedamp. The quantity of air circulating through the district is 25,000 cu. ft. per min. All roadways are supported by steel-arch girders 12 feet by $8\frac{1}{2}$ feet. How would you proceed to clear the roadways *AB* and

AC with the minimum interruption of work at the face of the conveyor unit ? (50)

A. Fig. 92 is the plan referred to in the above question. The full dip of the strata is 1 in 6 in the direction of the road *AC*, and firedamp will tend to accumulate along the road *AB* and towards the point *A* in the intake airway. If proper steps are not taken, this firedamp will be carried with the air-current round the working face of the district shown on the plan. To remove the firedamp in a

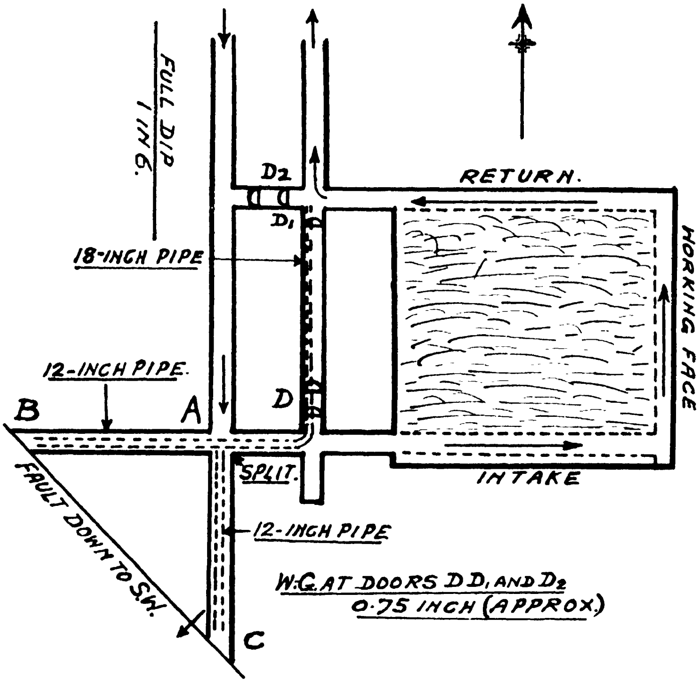

FIG. 92.—Clearing away an Accumulation of Firedamp.

safe way as it is given off from the two developing roadways *AB* and *AC*, and to prevent its being carried along the face of the conveyor unit, the two roadways should be ventilated by a separate split of air having a pipe return. Starting at the point *A*, 12-inch diameter thin metal pipes should be carried towards the faces *B* and *C* as shown by dotted lines on the plan. These two pipe-lines should merge into one 18-inch thin metal pipe extending from *A*, and passing the doors between the posts and the solid side of the coal at the positions D and D_1, to the return airway. In this way sufficient air would circulate by the separate splits to keep these clear from firedamp, while, at the same time, the quantity of air circulating along the conveyor face would not be decreased to any appreciable extent. With the material at hand, these alterations should be carried out during the week-end, thus giving time for the job to be completed without interrupting work on the conveyor face. In order to pass away the accumulated gas in a diluted state to the return airway, the main pipe through the doors should be installed first. The pipes into the roads AB and AC should be extended from the point A gradually towards the faces at the points B and C.

Duties to be carried out by a Ventilating Engineer

2. As manager of a large modern mine in which safety-lamps are required to be used, you are about to appoint a ventilating engineer. Outline in some detail the duties you would ask him to carry out. (30)

A. The following might be considered to be some of the most important duties given to a ventilating engineer at a large colliery where safety-lamps are used :—

(*a*) The underground officials and the safety engineer should keep a sharp eye on all persons using safety-lamps in carrying out their work on haulage roads and faces, to ensure that lamps are used carefully and efficiently. Frequent demonstrations might be necessary both at the surface and in the mine. Lamps should be examined at the appointed meeting-stations of the various districts of the pit, and such an examination should be complete externally, and internally as far as possible. Re-lighting of lamps at fixed points should be carried out by officials using a reliable apparatus.

(*b*) All workmen should be instructed in the correct method of detecting firedamp underground. The proper use of the lamp in the presence of firedamp should be fully explained to all workmen. The correct method of procedure, should firedamp be detected, must be explained to every workman at the face.

(*c*) Dealing with accumulations of firedamp in any district of the mine should be the work of the ventilating engineer and the firemen or deputies, after proper consultation.

(*d*) The engineer appointed should give his attention to the quality and quantity of air-current in the intake airways, faces, and return airways, such as :—

(i) Monthly records of ventilation.

(ii) Sections of the mine troubled with firedamp or other gases should be specially attended to. He should make suggestions to the management as to his method of increasing the volume of air to overcome the difficulty.

(iii) All ventilating appliances, air-crossings, doors and stoppings should be examined weekly and their condition reported.

(iv) The main ventilator, reversing arrangement and air ducts should be examined as far as possible every week.

(*e*) Advice should be given freely to workmen and officials alike by the ventilating engineer, who should see that such advice is carried into effect.

(*f*) If abnormal conditions exist in any district at any time owing to neglect on the part of officials or workmen, he should report such an occurrence to the management in his daily report and by personal interview.

Saving in Power by use of Evasée Chimney

3. The evasée chimney of a mine fan is 4 feet by 4 feet in cross-section at the bottom and 8 feet by 8 feet at the outlet. The volume of air being exhausted by the fan is 80,000 cu. feet per min. If the efficiency of the evasée is 40 per cent. and the density of the air (w) is 0·075 lb. per cu. foot, what saving in power is effected by the evasée ? (30)

A. The above question has been partly dealt with in previous answers ; see page 121. An alternative method of solution is as follows :—

Bernouilli's Theorem can be usefully applied in solving this question. This theorem states that the total energy in a flowing current of water or gas is a constant at any point in the direction of the flow, assuming friction loss be neglected. The kinetic energy plus pressure energy is a constant, and any loss sustained by the former is gained by the latter. At the base of an evasée chimney the kinetic energy is large, owing to the high velocity of the air discharged from the fan, but as the top of the evasée is reached this velocity is greatly reduced, and much of the kinetic energy has been converted into pressure energy, with an increase in pressure head.

Let m = Mass of air-current

$$= \frac{80{,}000 \text{ cu. ft.} \times 0{\cdot}075 \text{ lb. per cu. ft.}}{60} = 100 \text{ lbs. per sec.}$$

V = Air velocity at base of chimney

$$= \frac{80{,}000}{16 \times 60} = 83{\cdot}33 \text{ ft. per sec.}$$

V_1 = Air velocity at top of chimney

$$= \frac{80{,}000}{64 \times 60} = 20{\cdot}83 \text{ ft. per sec.}$$

g = Acceleration due to gravity $= 32{\cdot}2$ ft. per sec.^{2}

Then—

$$\text{Kinetic energy} = \frac{\frac{1}{2}mV^2}{g} \text{ ft.-lbs. per sec.} \quad . \quad . \quad (1)$$

$\therefore$ Gain in pressure energy

$$= \frac{(\frac{1}{2}mV^2 - \frac{1}{2}mV_1^2)}{g} \times \frac{40}{100} \text{ efficiency} \quad . \quad (2)$$

$$= \frac{m(V^2 - V_1^2)}{2g} \times \frac{40}{100}$$

$$= \frac{100(83{\cdot}33^2 - 20{\cdot}83^2)}{64{\cdot}4} \times \frac{40}{100}$$

$$= 4043{\cdot}4 \text{ ft.-lbs. per sec.} \quad . \quad . \quad . \quad (3)$$

$$\text{and Gain in Power} = \frac{4043{\cdot}4}{550} = \underline{7{\cdot}35 \text{ H.P.}} \quad . \quad . \quad . \quad . \quad (4)$$

Efficient Air-lock at the Top of an Upcast Shaft

4. The report of the Royal Commission on Safety in Coal Mines (1938) stresses the importance of a properly constructed air-lock at the top of an upcast shaft in order to minimise surface leakage of air. Sketch and describe what you consider to be an efficient air-lock at the top of an upcast shaft which is in regular use for winding purposes. (30)

A. The subject-matter of the above question has been dealt with in answering a previous question of the same kind ; see pages 92–4 and Figs. 62 and 63.

Ventilating Terms

5. Explain clearly any *four* of the following ventilation terms :—
(*a*) Coefficient of friction ; (*b*) Fan-drift water-gauge ; (*c*) Motive column ; (*d*) Mine characteristic curve ; (*e*) Fan efficiency ; (*f*) Blackdamp. (30)

A. *Coefficient of Friction.*—See pages 114–15 and Fig. 74.

Fan-drift Water-gauge.—This means the water-gauge reading in inches, obtained by passing a tube from one leg of the gauge through the brickwork into the fan drift, and leaving the other leg of the gauge open to the outside atmosphere.

Motive Column.—This is usually defined as a column of downcast air which is in excess of that part of the downcast-air column required to balance the upcast-air column. In the case of a natural motive column it can be calculated from the air weight or temperature in each of the shafts and the depth of the upcast shaft in feet, as follows :—

$$\text{Motive Column in feet} = \frac{w_1 - w_2}{w_1} \times d, \quad . \quad . \quad . \quad . \quad (1)$$

where w_1 = Weight of one cubic foot of downcast air,
w_2 = ,, ,, ,, ,, ,, ,, upcast air, and
d = Depth of upcast shaft in feet.

The following formula is often used when temperatures are given instead of pressures, being derived from the above :—

$$\text{Motive Column in feet} = \frac{T - t}{460^\circ + T} \times d, \quad . \quad . \quad . \quad . \quad (2)$$

where T = Average air temperature in upcast shaft,
t = ,, ,, ,, ,, downcast shaft, and
d = Depth of upcast shaft in feet.

For calculations on above, see pages 107–8 and 112–13.

In the case of a ventilating fan producing a motive column, the calculation of the latter is as follows :—

$$\text{Motive Column} = \frac{V_T(V_T \pm V_R \tan\theta)}{g}, \quad . \quad . \quad . \quad . \quad (3)$$

where V_T = Tangential velocity of fan rim per sec. = Circumference of fan wheel × Revs. per sec.
V_R = Radial velocity of air through fan = Cu. ft. of air per sec. ÷ Circ. × Width.
θ = Angle of curvature of fan blades (+ for forward-curved, − for backward-curved blades).

Mine Characteristic Curve.—This shows the capacity of the mine for the passage of an air-current, taking into consideration also the pressure which must be applied to circulate the air.

If Q be the quantity of air passing in cu. ft. per min.,
P the ventilating pressure in lbs. per sq. ft. of sectional area, and
R the resistance of the mine by Atkinson's method ;
then the curve has the form of $P = RQ^2$.

Fan Efficiency.—The relationship existing between the air horse-power and the fan-shaft horse-power determines the efficiency of a fan. Calculated as follows :—

Air horse-power or output

$$= \frac{Q \text{ in. cu. ft. per min.} \times \text{W.G. in inches} \times 5{\cdot}2}{33{,}000} \quad . \quad . \quad (1)$$

Fan-shaft horse-power or Brake horse-power of driving engine or motor . . . . (2)

Efficiency of Fan (per cent.)

$$= \frac{\text{Air horse-power or output}}{\text{Brake horse-power of driver}} \times 100 \quad . \quad . \quad (3)$$

Blackdamp.—See pages 99–100.

Control of Atmospheric Conditions in Hot and Deep Mines

6. The control of atmospheric conditions in hot and deep mines has been the subject of extensive investigations and several reports during the past 20 years. Summarise this research. (30)

A. The various sources of heat in deep mines might be tabulated as follows :—

(i) The air-current flowing in mines becomes heated by contact with the strata when passing through mines and taking the temperature of such ; also the oxidation of strata, chemical reactions of liquids in contact with strata, and the crushing and grinding of strata cause additional friction and heat : say approximately 46 per cent. of the whole source.

(ii) Heat caused by the compression of the air-current in shafts and inclines, and by the friction of the air-current in passing through the mine airways and shafts : say approximately 21 per cent. of the whole.

(iii) Men, horses and lamps in mines cause heat to be given to the air-current to the extent of about 12 per cent. of the whole.

(iv) Heat is caused in mines by the working of machinery, amounting to about 12 per cent. of the whole.

The most important source of heat to be considered in dealing with the given question comes under the first heading. The geothermic gradient varies at different points when penetrations are made into the earth's crust. In Great Britain an average figure is 1° F. for every 70 feet of depth. In some foreign countries the gradient varies from 1° F. for every 30 feet of depth to 1° F. for every 220 feet of depth. In parts of South Africa the gradient is 1° F. for every 182 feet of depth from the surface, and it has recently been suggested by experts that workings there might reach a depth of 12,000 feet in the future.

Face temperatures of about 100° F. dry-bulb and 87° F. wet-bulb, have been recorded in a British mine, while higher temperatures are recorded in other countries. Continuous hard work by men in such temperatures is possible only when an excellent ventilating current in continuously maintained. Heavy perspiration in high underground temperatures causes loss of salt from the blood and gradual weakening, and if a good air-current is not maintained this weakening might result in collapse. When the wet-bulb temperature at the face reaches 80° F. good ventilation is most essential to allow of healthy perspiration by the air-current absorbing moisture quickly from the bodies of the workmen.

The control of heat in deep mines might be summarised as follows :—

(i) Having an adequate air-current in such mines passing through shafts and smooth-lined airways of large size at a moderate velocity, thus reducing heat caused by friction, compression, spontaneous combustion and oxidation. To reduce the temperature of the inlet air-current, and at the same time the wet-bulb temperature at the face, refrigerating plant at the surface has been in use for some years, thus reducing the former temperature by 30° to 35° F. and the latter by at least 10° to 12° F.

(ii) Compressed air might be used to a greater extent for working underground machinery, thus preventing much evolution of heat and also cooling the air-current. The heat generated by machinery near the shafts might be transferred to the upcast shaft to increase the ventilating pressure.

(iii) The use of electricity in mines generally, where conditions are favourable and safe, will tend to reduce the air temperature. Electric lamps might be used instead of oil lamps, and horses might be dispensed with in favour of machinery.

(iv) Care should be taken on haulage roads to reduce friction by having good rope rollers, good track, easy-running tubs, and slow-moving haulages. The machinery introduced at working faces and trunk roads should be well set-out to work with the minimum amount of friction and prevent heat being generated on a large scale.

See also the answers given to similar questions on pages 125–6 and 145–6.

Determination of Candle-power of Miner's Electric Safety-Lamp

7. How would you measure (*a*) the mean horizontal candle-power, and (*b*) the mean spherical candle-power of a miner's electric safety-lamp ? What steps would you take to ensure that the minimum standards of light as laid down in the Lighting Regulations were being maintained ? (30)

A. The candle-power of lamps used in mines is measured by some form of modern photometer in which direct readings are obtained. To measure the mean horizontal candle-power, a photoelectric-cell swinging-arm photometer, with a galvanometer calibrated in standard candles, might be used. The lamp to be tested is placed on the vertically-adjustable turntable, which is graduated in 5-degree or 10-degree divisions all round the circle, and the vertical level is adjusted so that the lamp filament is central with the axis of the swinging arm. Set the swinging arm and read off the candle-power at every 5 degrees or 10 degrees round the circle. The results, when

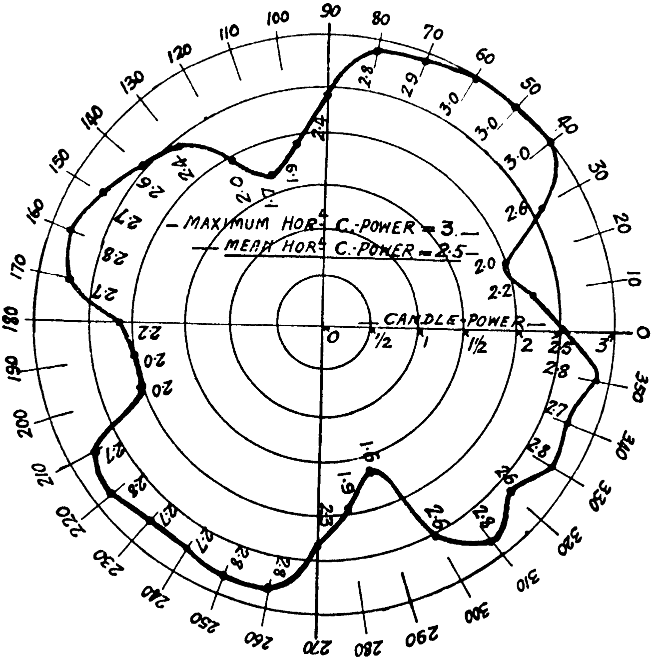

Fig. 93.—Polar Curve of Light Distribution in Horizontal Plane for Alkaline Hand-lamp with Cylindrical Well-glass.

plotted, give the horizontal polar curve, and the arithmetical mean of all the readings is the mean horizontal candle-power. Fig. 93 shows a polar curve of light distribution in the horizontal plane for an alkaline hand-lamp with cylindrical well-glass, obtained in the manner described above. By taking the mean of all the readings given, the M.H.C.P. is 2·5.

To measure the mean spherical candle-power of a lamp, the simplest method is to take a direct reading in a cube photometer. Failing this, it can be derived from the horizontal and vertical polar curves.

The minimum requirements of the Lighting Regulations are that the mean intensity over the horizontal angle of distribution must not be less than 1·5 candle-power for all lamps. The intensity of light for cap lamps shall not be less than one candle-power within a solid angle of 180°, and for hand-lamps one candle-power within a horizontal angle of 180° and a vertical angle of 50°. The mean spherical candle-power for hand-lamps shall not be less than 0·75, and for cap lamps not less than 0·4.

To ensure that these minimum standards of light are being obtained, lamps should be checked for maximum horizontal candle-power at the end of the shift by using a portable photometer specially designed to give accurate results quickly. If these tests show that the maximum H.C.P. is being maintained, it follows that the mean H.C.P. will be correct. Similar tests should be carried out for deterioration after 50 hours, 300 hours and 600 hours in use.

PART III.—EXPLOSIONS IN MINES, UNDERGROUND FIRES, AND INUNDATIONS

MAY 1931 EXAMINATION

Scientific Research on Explosions and Fires

1. In what directions has scientific research lessened the risks of explosions and fires in mines? (30)

A. *Explosions of Firedamp.*—Going back over a period of about 125 years, the invention of the safety-lamp by Sir Humphry Davy greatly reduced the risk of explosions in mines. Recent research work by Royal Commissions, and by the Safety in Mines Research Board, has resulted both in greater security and better illumination with safety-lamps.

Experiments have been carried out at the Buxton Research Station with various mixtures of firedamp and air, and the results have been tabulated for circulation in the coalfields. The addition of other compounds, such as carbon dioxide, nitrogen, and gases liberated by explosives, to the mixture of firedamp and air has also been properly investigated. Firedamp explosions by sparks from coal-cutter picks and hand-operated picks have been thoroughly investigated during recent years, using various rocks and various steels. Research work has likewise been carried out on the efficiency of flameproof casings for electrical equipment in mines.

Explosions of Coal-Dust.—Going back over a period of about 60 years, valuable research work in connection with the explosive properties of coal-dust has been carried out by Dr. William Galloway, Sir Henry Hall, and Sir William Garforth. During recent years this research has been continued by the Safety in Mines Research Board at Eskmeals Station, and latterly at the present Buxton Research Station, and the following results obtained:—

(*a*) The explosive property of fine coal-dust, in the entire absence

Note.—Under Examination conditions, candidates are required to answer *five* questions only from this Section. The figures in brackets indicate the maximum marks allotted to each question.

of firedamp, has been proved conclusively by practical experiments on a large scale.

(*b*) The quenching effect of fine stone-dust when mixed with the coal-dust, to prevent explosions of the latter and to arrest explosions in the initial stage, has been effectively demonstrated. Experiments have also been carried out to prove that more stone-dust is necessary for safety where the air-current contains firedamp.

(*c*) Dangerous dusts are those containing a high percentage of volatile matter, but it is combustion of the solid particles and not of the distilled volatile matter that causes the explosion. The following figures have been compiled as a result of the foregoing experiments at Buxton Research Station :—

Quenching Effect of Stone-Dust

Volatile matter in coal, percentage	50–42	42–36	36–32	32–28	28–25	25–21	21–16	16–12½
Stone-dust required for safety with no firedamp	75	70	65	60	55	50	40	20
Stone-dust required with 2½ per cent. firedamp	85	85	80	80	75	70	65	55

(*d*) The force developed by coal-dust explosions of limited range, and the effect of restrictions in the path of the explosion, have been investigated. The composition of the "afterdamp" has been determined.

(*e*) Dusts from all the British coalfields have been examined and tested in the experimental chambers.

Shot-firing.—During recent years, experiments have been carried out at the Research Station at Buxton to prove the danger of explosion by shot-firing in mines. The effects of the quantity of explosive, the amount and nature of the stemming, and the position of the detonator in the charge, have all been thoroughly investigated, and greater safety in connection with blasting in mines has resulted from this work. The effect of breaks in a shot-hole and the dangers attending such have also been fully demonstrated by experiments.

Underground Fires.—Experiments in a specially constructed chamber, dealing with the various kinds of materials for building packs, the leakage of air through packs, and the composition of the gases given off after heating, have been carried out within recent years at the Buxton Research Station. Explosions have occurred and all details have been collected. Graphs have been prepared from actual observations at collieries, showing the danger zone in building-off fires and in opening out after fires. Experiments have been carried through on oxidation of coal and strata, and much valuable information has been obtained thereby regarding pressure and composition of the gases given off.

Comparison of Systems of Rescue Organisation

2. Which of the two permissible systems of rescue organisation do you prefer ? Enumerate their relative advantages and disadvantages. (25)

A. The Coal Mines Act requires the following rescue arrangements :—

Colliery	with	under 500	employees,	one	trained	rescue	man.
,,	,,	500–1,000	,,	two	,,	,,	men.
,,	,,	over 1,000	,,	three	,,	,,	,,

Each rescue man must have 1 week of continuous training, or at least 4 complete days' training, during his first 3 months, and afterwards 6 days' training per annum (after the first 3 months), two of which must be at the mine. The above training also refers to rescue men employed at a Central Station.

The two systems of rescue organisation are as follows :—

I.—*Central Rescue Station System.*—This system is to operate in collieries within a radius of 15 to 20 miles from the station. There must be at least two trained brigades, full equipment, and rescue wagon.

II.—*Local System at Collieries.*—This is for mines not connected with a Central Rescue Station. There must be two sets of either smoke-helmet apparatus or self-contained breathing apparatus, in addition to the following equipment required at all collieries :—

(i) One oxygen reviving apparatus.
(ii) Two small birds or mice.
(iii) Two electric hand-lamps per brigade.
(iv) One safety-lamp for each brigade member.
(v) One ambulance box with antiseptic solution and fresh water.
(vi) Ventilation plans suitable for use by brigade.
(vii) Ambulance room.

Advantages of Central System.—The Central Rescue Station

System is preferable to the local system by reason of the following advantages :—

(1) The men are better trained in the use of apparatus, and are capable of using it successfully under various mine conditions.

(2) Supervision of the brigades is generally excellent, and thus very useful in case of disaster.

(3) All appliances and apparatus are kept continuously in good working order, and there are ample reserves.

(4) Distance is no object where a motor wagon is available for transport, as the brigade can be on the road to the colliery a few minutes after receiving a call, and at the colliery ready for work within half an hour.

(5) The superintendent of the Rescue Station is generally a man of wide experience in rescue work and practical mining, and his help in case of disaster is valuable.

The disadvantage of the Central System is the fact that local men are more intimately acquainted with the workings of their colliery than are the Central System men.

Local System.—The local system of rescue organisation has the following advantages and disadvantages :—

(1) The men are well acquainted with the colliery workings.

(2) Their pit training might be better than that of Central Station men.

(3) The apparatus might not be kept in proper working order and thus be useless in case of disaster.

(4) The accessories might be limited in number and thereby lead to accidents during rescue work.

(5) The men are not trained to use apparatus in the same way as Central Station men, and they might get into difficulties in case of accident to the apparatus.

Spontaneous Heating—Investigation and Dangers

3. As Manager you receive a report that spontaneous heating is suspected in a section of the mine. Describe in detail the steps you would take to make an investigation and to eliminate possible elements of danger. (25)

A. *Investigation.*—A thorough examination of the section should be carried out without delay, to determine the position of the fire and the condition of the circulating air. A small bird or mouse, a thermometer, and sampling-bottles should be used in doing this. If heating is discovered by this examination and the position of the same approximately located, the workmen on the return side should be withdrawn without delay until the defect has been remedied.

Control.—The steps taken to eliminate possible elements of danger should include the following :—

(*a*) A thorough stone-dusting of all roads in the section.

(*b*) Keep the section continually under expert observation to determine the progress of the heating and the condition of the circulating air-current. The ventilation should be kept up to the maximum quantity and great care exercised in dealing with fire-damp. Shot-firing should be discontinued until safe conditions are established.

(*c*) Preparations should start without delay for the building of preparatory stoppings and the transporting of the required material for closing these in case of necessity.

(*d*) If the conditions become worse, all the men should be withdrawn from the section and the preparatory stoppings closed.

(*e*) After a few days, during which air samples should be frequently taken from the area inside the stoppings and analysed, it may be necessary to open out the stoppings again and attack the fire or heating by driving into the affected area and removing the heated material.

Approaching Old Workings containing Water

4. A seam of coal is being worked adjoining an old colliery which is full of water. Give an account of all matters which should be investigated to ensure the safety of your workings. (25)

A. To ensure safety under the conditions given above, the following matters should receive attention :—

(i) The plans of the old colliery should be obtained, if possible, and a safety margin of about 50 yards arranged round the position of the faces and exploring drifts to allow for inaccuracy of such plans. Faults and dykes near the old faces should be traced out and specially marked on the new plans.

(ii) If any old shafts are found open in the old area, they might be pumped to the level of the workings by the use of a borehole rotary pump, the head of water being reduced in this way.

(iii) The working faces of the district approaching the old workings should be stopped at a distance of 100 yards from the safety margin designed on the old plans, and arrangements should then be made to drive an advance heading directly towards the old workings at a convenient part of the face. This heading should be only 8 feet wide, and after advancing a distance of, say, 10 to 15 yards, a borehole 100 yards long should be driven towards the old workings, using a Burnside safety boring appliance (p. 199). If a holing is not obtained in this way, the face of the heading should be advanced again to within 15 yards of the end of the hole, after which the boring should proceed again until the water is tapped and got under control.

(iv) Before the boring is carried out, a dam should be erected at the entrance to the heading, so that it can be closed quickly in case of accident during the boring operations.

(v) The subsequent development of the workings of the district would be determined after the water had been tapped and the exact position of the old workings had been found and corrected on the old plan.

Coal-Dust and Screening Plant

5. What arrangements should be made at the surface with the object of preventing coal-dust from the screens entering the mine? In what parts of the mine would you expect to find the most dangerous dust? Describe conditions that would lead you to consider that the dust is dangerous, and state how you would deal with such conditions. (25)

A. Unless exemption is obtained, the screening plant at a colliery must be placed at least 80 yards from the downcast shaft. In addition to this, it is both effective and economical to have a "Cyclone" dust-collecting plant at the screens, for the purpose of collecting all the fine coal-dust from the coal during its passage over the screens and picking tables. Such a dust might be used for firing the boilers.

The parts of a mine where the most dangerous dust is found are on the main intake airways and haulage roads, and also at the working faces. The former dust is generally very fine and dry and is also mixed with oxygen from the air; the latter kind is fresh and of a grade of fineness depending on the method of its production at the face.

Dangerous dust in a mine would be both fine and dry, while at the same time it would contain not less than 50 per cent. of combustible matter. The best safeguards against coal-dust explosions in these circumstances are: thorough stone-dusting of all roads and working faces with fine stone-dust so as to maintain at least a

50–50 mixture of coal-dust and stone-dust; great care in dealing with firedamp to prevent accumulations and explosions; and strict discipline in the use of explosives for blasting-down coal and other strata.

Recovery of Workmen from Area of Explosion

6. During recovery work after an explosion, men have been found alive, but unconscious, by a rescue team wearing self-contained breathing apparatus. Assuming that the team is 300 yards from the base, what steps should be taken? (25)

A. The rescue team should arrange to get the unconscious men to purer air as quickly as possible, but not necessarily to the base. Free use should be made of the oxygen reviver carried by the rescue team while these operations are being carried out. The by-pass valve of some of the breathing apparatus might be used to assist in the reviving work, but on no account must the rescue men put themselves in danger by doing this.

Upon reaching purer air the base party should be called on to help in the work of artificial respiration and the transport of the workmen to the base for medical treatment if necessary.

NOVEMBER 1931 EXAMINATION

Point of Origin and Cause of an Explosion

1. Write an account of an investigation after an explosion, with particular reference to indications as to the point of origin and cause. (30)

A. After an explosion in a mine, an investigation should be made into the same as soon as air is circulated through the mine and the afterdamp has cleared away. A rescue brigade should accompany the party, which is generally made up of colliery officials, miners' representatives, inspectors of mines, and a base party.

Starting from the downcast shaft and following out the path of the explosion, a large-scale plan of all roads should be sketched out from actual measurements. This plan should include width of roads, position and extent of all falls of roof, position of wrecked ventilating appliances, supports, trams and other obstacles. Care must be taken to mark on the plan the position of all victims of the explosion, and the direction of the force of the explosion as determined by the deposits of coke and dust on the supports and the marks of burning. All electrical equipment and appliances, together with signalling wires and cables, should be properly examined.

In tracing the point of origin of the explosion, the direction of the various forces would be determined from the foregoing details, and the point of origin might be established where several lines of force converge. This will probably lie some 50 or 60 yards from the spot where the maximum force and the greatest damage is indicated in the case of a dust explosion, and where the greatest heat has been produced in a gas explosion. The coked deposits found on one side of the supports, and the fine dust found on the opposite side of the same, are of great assistance in fixing the direction of the force of an explosion of coal-dust; the latter is due to the back-blast to the seat of the explosion, while the former is deposited by the explosive blast.

Careful search should be made for any of the following to establish the cause of the explosion at the point of origin :—

(*a*) A shot-hole from which a blown-out or overcharged shot has occurred, possibly at the face, or in a heading.

(*b*) A damaged or unsafe safety-lamp might be found where gas had been reported previous to the explosion, or at a lamp-lighting station.

(*c*) Defective electrical apparatus might be discovered, or a signalling wire station might be at hand.

(*d*) A damaged gob-fire stopping, or a heading liable to outbursts of firedamp, might be within reach.

(*e*) A haulage curve or a fall of sandstone roof capable of producing sparks and igniting firedamp might also be near at hand.

Reversal of Ventilation in Case of an Underground Fire

2. In consequence of an underground fire, the question of reversing the ventilation arises. What matters should be considered before deciding whether the reversal shall be carried out ? (25)

A. Underground fires on the intake side of a mine are more dangerous than those on the return side, as the lives of the workmen are endangered by the former, and it is in this connection that the question of reversing the ventilation arises.

The matters to be considered before deciding this question are as follows :—

(i) The position of the fire in the intake air path and the extent of the same.

(ii) The state of the air-current passing to the workings as regards dangerous gases and smoke, and the possibility of saving the lives of the workmen.

If the fire is situated at any point on the intake side inbye of the separation doors, and it is in the early stages, it might be attacked without delay, and the air-current short-circuited to the return on the inbye side, to prevent the fumes reaching the faces. The workmen could then be withdrawn to a point of safety.

In the event of the fire reaching an alarming or extensive stage quickly, so that short-circuiting of the air-current is not possible, then reversal of the air-current should be carried out without delay, to allow of the safe withdrawal of the workmen and subsequent work at the seat of the fire to stamp it out. All workmen should be warned, if possible, that the air-current is going to be reversed in direction.

The foregoing details are quite apart from any gob-fire which might occur in the workings of a mine, some distance from the usual path of the air-current, and which might be dealt with in other ways than by changing the direction of the circulating air or by short-circuiting the same.

Unwatering of Waterlogged Workings

3. Describe in detail the arrangement you would make for unwatering an area of waterlogged workings lying to the rise. The workings are sealed-off by dams in two cross-measure drifts, the water-pressure on the dams being 250 lbs. per sq. inch. (25)

A. Assuming that the cross-measure drifts connect up two seams, or two parts of the same seam separated by a fault, and that they are open to the lower seam workings, arrangements can be made for safely unwatering the waterlogged workings lying to the rise.

In one of the drifts, say 10 yards back from the face of the dam, a suitable spot should be selected and prepared for the building of a new dam of strong construction. After this dam is completed, the old dam might be tapped by means of a Burnside safety boring appliance, and the water run off as required by connecting suitable pipes to the borer. The new dam would then be ready for use in the event of the old dam giving off a leakage of water above the capacity of the pumping plant.

Fig. 94 shows the method of constructing a dam of modern design suitable for the above conditions. Such a dam is constructed with concrete in the form of a wedge, the concrete being reinforced with steel bars. A circular passage 4 feet in diameter, through the dam, is fitted with a cast-steel door and frame, both

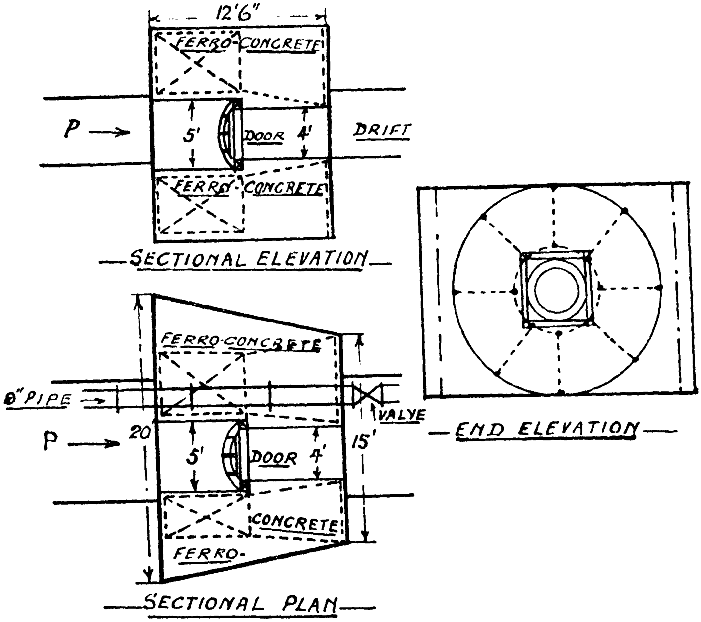

FIG. 94.—Ferro-concrete Dam.

having machined faces. The door is suspended on substantial hinges and can be closed on the frame, in case of emergency, with the cambered side towards the pressure.

Objections to Certain Kinds of Incombustible Dust

4. What are the objections to using certain kinds of incombustible dust on the roadways in mines for preventing the propagation of an explosion? Why is a highly hygroscopic dust undesirable? Describe suitable tests for this. (25)

A. During the process of exhaling, the lungs throw out certain kinds of dust of a coarse nature, while other kinds of a finer nature are retained. The latter, in the form of incombustible dust, are therefore objectionable for use in mines by reason of their dangerous effect upon the health of the workmen employed in the mine.

Fine dusts containing more than 30 per cent. of silica are highly injurious to the workmen when inhaled into the lungs, as they tend to produce silicosis, fibrosis and anthracosis of the lungs, which are usually followed by tuberculosis. The inertness of such dusts is also a dangerous feature, as they produce a slowing-down of physiological processes. It has recently been pointed out by experts on this subject that the most dangerous material of the silica type consists of minerals in which silica is in combination with other elements to form silicates, in particular the hydrated silicate of aluminium and potassium. A mineral of the mica family, termed Sericite, which is abundantly present in rocks and materials found in mines, gives rise to a fine fibrous dust which is considered very dangerous to inhale.

"Chance's mud," which is obtained from manufacturing processes, contains very fine calcium carbonate and sometimes sulphides. The carbonates in this state are injurious to the lungs, while the sulphides might produce dangerous quantities of H_2S.

The residue from water-softening plants is composed of very fine calcium carbonate ($CaCO_3$) and magnesium carbonate ($MgCO_3$) both of which are injurious when inhaled into the lungs.

Fuller's earth tends to produce the same effect as the finely-divided carbonates already referred to, that is, a clogging-up of the lungs.

A highly hygroscopic dust is not desirable for use in mines for mixing with coal-dust deposits. Unlike coal-dust, it absorbs moisture quickly from the air-current and becomes waterlogged and caked. In this condition it tends to separate from the coal-dust, which remains comparatively dry and dangerous. In the event of an explosion, the hygroscopic dust is not raised into cloud form with the coal-dust and does not prevent the propagation of flame as intended. For example, dusts containing 67½ per cent. of shale, or 57½ per cent. of limestone, are not liable to cake, but a dust containing 40 per cent. of gypsum is liable to cake.

Suitable tests of a highly hygroscopic dust might include the following :—

(*a*) Take a large flat tray and cover it with a layer of fine coal-dust and then a layer of the fine hygroscopic dust. Allow this tray to remain in a mine road for several weeks. By the use of compressed air at low pressure from a jet, the suitability of the inert dust on the tray can be determined. Some of the dust may be raised in cloud form by the jet, and the residue left on the tray can then be examined.

(*b*) Take a sample of dust from the tray, to include the whole vertical section, and without mixing or drying, test it in a small tube or shot cannon for the presence of flame.

(*c*) Take a mixed sample of dust, place it on a tray over a vessel containing water for a period of one week. After this period examine it as in (*a*).

Deductions from given Analysis of Return Air

5. An air sample taken from the return airway of a district in a seam of coal liable to spontaneous combustion shows the following analysis :—

	CO_2	0·40	per cent.
	CH_4	1·17	,,
Oxygen	(O_2)	19·92	,,
Nitrogen	(N_2)	78·49	,,
	CO	0·02	,,

What deductions would you make from this analysis and on what grounds would you base your deductions ? (25)

A. The given analysis of the air in the return airway of a mine subject to spontaneous combustion, indicates clearly that the oxygen is low and that the gases CO, CO_2 and CH_4 are being formed. It is therefore apparent that a heating is in progress in an advanced stage, and active fire might be expected at any moment at some point in the workings between the intake and return air-currents.

The following calculations from the figures given confirm the above statement :—

Normal air contains 0·03 per cent. CO_2, 20·93 per cent. oxygen and 79·04 per cent. nitrogen.

The percentage of oxygen corresponding to the percentage of nitrogen present in the sample is $\frac{78{\cdot}49 \times 20{\cdot}93}{79{\cdot}04}$ or 20·78 per cent. The amount of oxygen absorbed from the air is therefore 20·78 − 19·92 or 0·86 per cent.

The ratio $\frac{CO \text{ formed}}{O_2 \text{ absorbed}}$ is $\frac{0{\cdot}02}{0{\cdot}86}$ or 0·023 or 2·3 per cent., and this is high, which indicates that coal oxidation is in the advanced stage. Normal figures are 0·05 to 0·4 per cent.

The ratio $\frac{CO_2 \text{ formed}}{O_2 \text{ absorbed}}$ is $\frac{0{\cdot}4 - 0{\cdot}03}{0{\cdot}86}$ or 0·43 or 43 per cent., which is also above normal and confirms the above statement.

Finally, the figures given in the question are very closely related to those obtained during gob-fire experiments carried out recently at the Buxton Research Station.

Rescue Apparatus

6. In what respect does the following rescue apparatus differ? (*a*) Oxygen apparatus; (*b*) liquid-air apparatus; (*c*) smoke helmet. Make a diagrammatic sketch of a type of self-contained breathing apparatus. What are the limitations of the smoke helmet? (25)

A. (*a*) In the *oxygen type* of breathing apparatus, the exhaled air passes from the mouthpiece by way of a flexible tube and an exhaling valve to the breathing bag, which contains two compartments. In this bag the air is purified by the extraction of CO_2, use being made of a caustic absorbent, or "Protosorb," for this purpose, and on entering the second compartment of the bag it is replenished with oxygen from an oxygen cylinder fitted with hand and automatic reducing valves. The purified and replenished air then passes through a cooling device before being drawn into the mouthpiece of the apparatus by way of the inhaling valve and flexible tube. The action of the apparatus is thus continuous while it is in actual use. The nostrils are closed by a suitable clip.

(*b*) In the *liquid-air apparatus*, the liquid to the extent of about 5½ lb. is contained in a metal vessel, and the vaporised air passes from this to the breathing bag and to the inhaling valve, which is situated in the flexible tube attached to the mouthpiece. The exhaled air passes from the mouthpiece by way of a second flexible

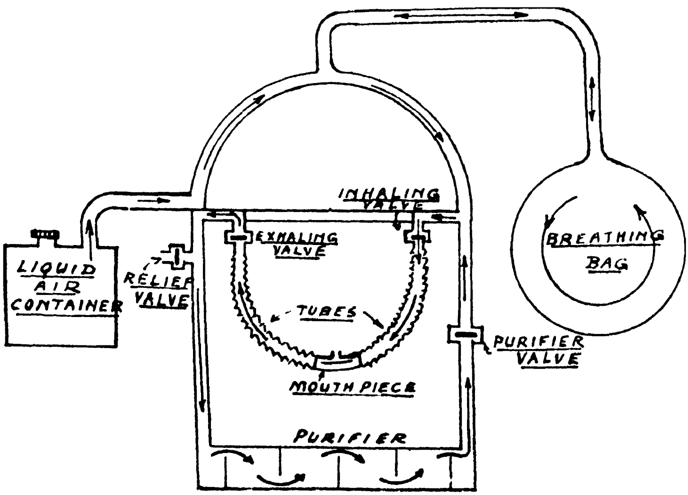

FIG. 95.—Flow Diagram of Aerophor Apparatus.

tube and thence through the exhaling valve to a purifier containing 2 lb. of caustic soda granules. The purified air then passes through a purifier valve before joining the supply of air coming from the breathing bag. This purifier valve is arranged to prevent air from flowing in a backward direction through the purifier. When the apparatus is first used, excess of air, together with the respired air, pass through a relief-valve to the outside atmosphere, this valve being situated between the exhaling valve and the purifier. Under these conditions the purifier is out of action. After a time, when the evaporation of air is less, a portion of the exhaled air passes through the purifier and is re-breathed, the apparatus thus becoming a regenerator. Fig. 95 is a flow diagram of the apparatus, from which the above description can be traced.

(c) The wearer of a *smoke helmet* obtains a supply of fresh air through two flexible tubes at the sides of the helmet, which are connected to a main air-tube coming from an air-pump placed in fresh air. The surplus and expired air pass away at the base of a leather fabric placed under the jacket of the wearer. Fig. 96 is an outline sketch of this apparatus.

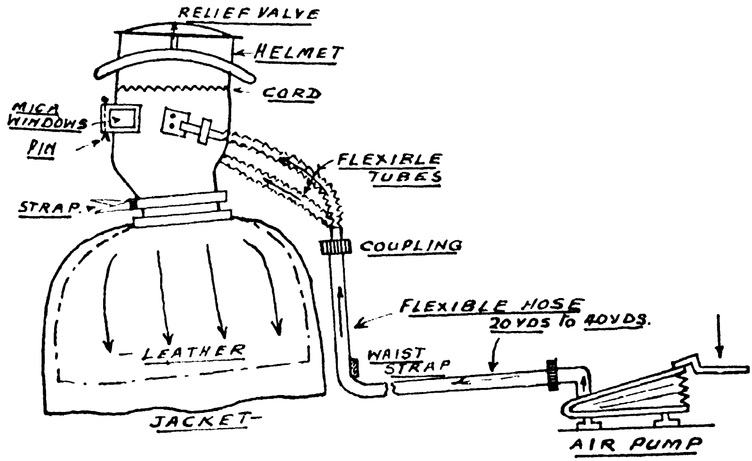

FIG. 96.—Smoke Helmet.

Fig. 97 shows diagrammatically the arrangement of the *Proto self-contained breathing apparatus* as required by the above question. Recent improvements in the apparatus are: zip fasteners for the breathing bag instead of clamps, thus giving easy access; improved mouthpiece, nose-clip, and fastenings; safety catch for

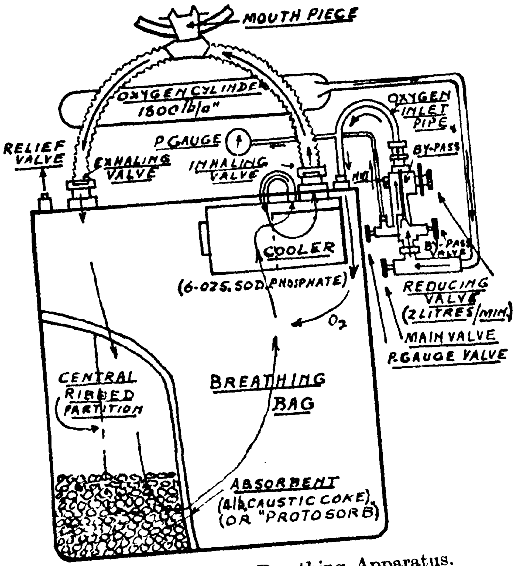

FIG. 97.—Proto Breathing Apparatus.

the main valve to prevent accidental closing ; lighter material of construction for the oxygen cylinder ; and improved valve con- struction to prevent accidents by the valves not operating properly.

The smoke helmet is limited in range to a distance of about 40 yards from fresh air, where the air-pump is operated by foot. It is heavy, and hearing is difficult during use. Owing to the large amount of fresh air circulating through the helmet the wearer is, however, enabled to work in high temperatures, say at the seat of a fire, for longer periods than with the oxygen apparatus. It might also be used for removing unconscious workmen from dangerous atmospheres situated within 40 yards from good air, but it is not advisable to use it continuously in air containing dangerous percentages of **CO** and **CO_2**. See also pages 188–9.

MAY 1932 EXAMINATION

Organisation of Rescue Work and Examination of Workings

1. A serious explosion occurs during a working shift, causing extensive damage. How would you organise and carry out the rescue of survivors, and the examination of the workings ? (30)

A. In the event of a serious colliery explosion the following action might be taken :—

(i) Instructions should be given immediately for calling-up the rescue brigade, mines inspectors, workmen's representatives, doctors, colliery officials, and assistance from neighbouring mines. At the same time the responsible officials should be instructed to carry out examinations of the ventilating and winding plants.

(ii) The condition of the air-current in the shafts should be tested by mice, small birds, and safety-lamps for the presence of **CO** and firedamp. If there is no evidence of a fire in the mine the ventilation should be restored without delay, and arrangements made for exploring underground.

(iii) The rescue brigades should make an examination of all airways near the shafts to ascertain the district or districts affected by the explosion, and a report should be made to the management as quickly as possible.

(iv) Rescue parties should then proceed with the work of exploration in each district, the rescue brigades being used in the affected area, and other parties in non-affected areas for the purpose of withdrawing all workmen from the mine.

(v) While the above operations are in progress a reliable person must be left in charge at the surface to organise base parties, ambulances, supply of material, and other details in connection with the exploring work. He should know all the details of the work and persons operating in the various parts of the mine.

(vi) The examination of the affected area should proceed as quickly as possible under the direction of the manager or his representative, the brigades forming the advance party. It is possible that better progress will be made in the return rather than in the intake airways, as the former are not damaged to the same extent as the latter. The intake airways should be properly examined after the rescue operations are completed. The rescue brigades should only accomplish such work as will enable them to proceed quickly into the various sections, other necessary work being performed by the following-up party in communication with a base party. In this way the workings would be explored as quickly as circumstances allow, and any survivors rescued without undue delay.

(vii) If any fires are discovered as exploring proceeds they should be attended to without delay, in a way likely to assist in the recovery of survivors of the explosion.

The Construction of a Concrete Dam

2. Sketch and describe the construction of a concrete dam suitable to withstand a water pressure of 300 lbs. per sq. inch in a level roadway 8 feet wide by 6 feet high. There is a feeder of 200 gallons of water per minute flowing along the roadway. How would you deal with this whilst constructing the dam? (25)

A. Fig. 98 shows sectional arrangement and front elevation of a concrete dam designed to resist a water pressure of 315 lbs. per sq. inch.

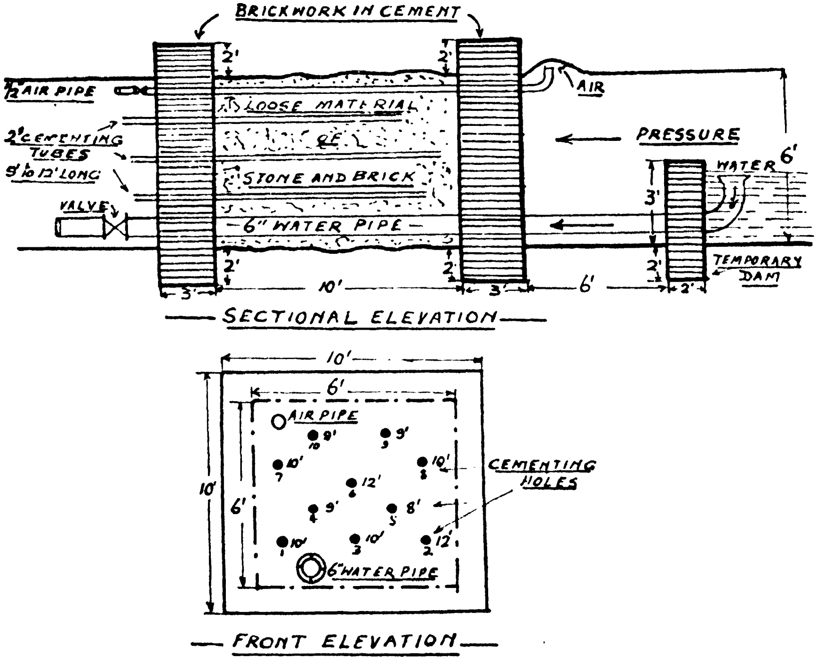

FIG. 98.—Construction of Concrete Dam.

The ends of the dam are supported by solid walls of brickwork 3 feet in thickness, built in cement. These walls penetrate the road surfaces to a depth of 2 feet. The centre plug between the retaining walls is made up of broken stone and broken brick for a length of 10 feet, and this material is cemented under a pressure of about 1,000 lb. per sq. inch, the cementing tubes being 2 inches in diameter and 8 to 12 feet long; about 10 tons of cement would be required.

The air-pipe near the top of the dam is ½ inch in diameter and is fitted with a valve.

The front elevation in Fig. 98 shows the position of the pipes in the dam, and the position and depth of the various cementing holes.

The back of the dam should finally be cemented by using the ½-inch air-pipe, and an additional pipe 17 feet long passing through the retaining walls. About 3 tons of cement would be used in this process.

To deal with the feeder of water during the construction of the dam, a temporary brick dam should be built as shown in the sketch, and the water passed away from the back of the dam by means of 6-inch pipes.

How Breaks in Roof Strata can contribute to an Explosion of Firedamp

3. How can breaks in roof strata contribute to an explosion of firedamp? Make a sketch plan and section to illustrate your answer. (25)

A. When a seam of coal is being worked by longwall, preliminary breaks are often produced in the roof strata by reason of sub-

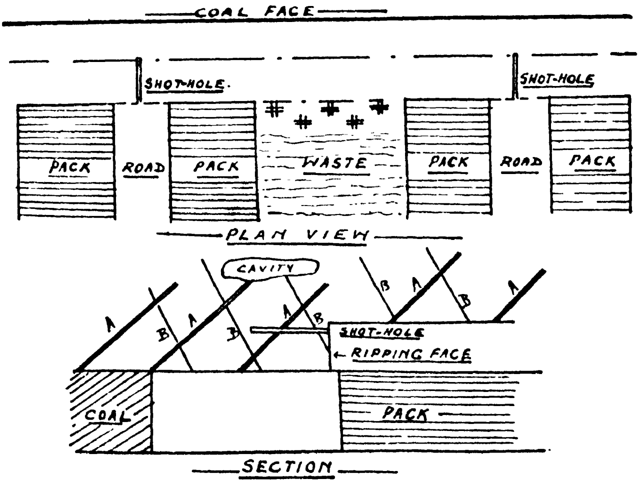

FIG. 99.—Longwall Workings and Roof Breaks.

sidence between the face and the waste workings. Such breaks occur a few feet in advance of the coal face and are inclined towards the waste as shown at *A*, Fig. 99. Main or secondary breaks are afterwards produced by weak face supports and by the strata subsiding on to the packs. These breaks are usually inclined towards the face, as shown at *B*, Fig. 99.

The breaks referred to above intersect some distance above the roof of the seam, and might extend to fissures and cavities containing firedamp, gas-laden strata in the vicinity of an upper seam, or the shear line of a fault in the strata. In any of these instances firedamp will be tapped and will pass into the breaks, and finally into the faces and waste workings of the seam being extracted. If a shot-hole penetrates a break containing firedamp, this can be exploded by the shot, even if permitted explosives are used, the gas being fired by compression.

The amount of firedamp in the workings can be considerably increased in this way and dangerous conditions produced, necessitating careful supervision in lighting, working and shot-firing, to prevent explosions. During the process of drilling a shot-hole in the coal, a break might be very close to the same, having a connection to a main break penetrating a fissure or cavity containing firedamp, and an explosion might result when the shot is fired. Explosives may be used for ripping-down roof for building packs in roadways, and, during the process of drilling, a break may be crossed which extends to the waste workings, thus giving rise to the danger of an explosion of firedamp when the shot is fired.

Effective roof control in longwall workings, by having good face supports and properly constructed roadside packs, tends to prevent roof breaks in the vicinity of the face, and thus reduces the danger of explosions of firedamp.

Fig. 99 shows the plan and section to illustrate the foregoing details.

Smoke Helmet—Uses and Limitations

4. For what purpose is the ordinary smoke helmet of use in coal mines, and what in your opinion are the limitations of its usefulness ? (25)

A. The smoke helmet has already been referred to in a previous question ; see pages 183–4 and Fig. 96. This apparatus might be used in mines for dealing with fires where a high temperature exists. It might also be used for short periods when penetrating into dangerous air zones in a mine to save lives, and for repairing or adjusting ventilating appliances in a dangerous mine atmosphere, provided that the air-pump is situated in good air.

The smoke helmet is limited in range to a distance not exceeding 40 yards from a fresh air base where the air-pump is established. The wide hose connection of $\frac{3}{4}$-inch bore between the helmet and the pump is heavy, springy, and difficult to work with; while at the same time it is liable to be flattened or cut by a fall of stone from the roof.

Messrs. Siebe Gorman & Co., of London, have recently made an improved type of hose apparatus which dispenses with the use of the helmet. The flexible connection between the wearer and the pump is made of thick-walled rubber tube of $\frac{1}{4}$-inch bore, and it is only a third of the weight of the original hose. The air is forced to the wearer of the apparatus through this tube from a hand-operated double-acting reciprocating pump, constructed in a special alloy known as Duralium. The flow of air is equalised by providing the pump with an annular spacing to serve as a receiver. The wearer carries a bag on his back to convert the regular flow of air into an intermittent flow as required by the lungs. The helmet is replaced by a mouthpiece, a nose-clip, and a suitable head-dress. This apparatus more than doubles the original practical range of hose-pipe apparatus in mines and greatly increases its safety and effectiveness.

Precautions where Electrically-driven Machinery is used

5. Electrically-driven coal-cutting machines and conveyors are being used on a coal face. What special precautions should be taken in connection with the equipment, and in the working operations, to prevent an ignition of firedamp or coal-dust? (25)

A. The special precautions to be taken in connection with the working of electrical coal-cutters and coal-conveyors might be as follows:—

(i) The electrical equipment should be of ample dimensions and robust construction for the heavy work to be undertaken, so as to reduce the danger of flashing and sparking. It should be properly encased to reduce the danger of explosions of firedamp, and all joints of such casing should be machine-faced. All bolts, nuts, etc., of the casing should be examined before and during the cutting shift to ensure that none is slack or missing. The casing should be examined frequently for internal wear by the moving parts inside the same.

(ii) The auxiliary apparatus, including starting switch and gate-end box, should likewise be of strong construction and effectively encased to prevent explosions. The gate-end box should be properly protected by overload and no-voltage releases, and the casing should be provided with a locking device to prevent opening while the current is on. The current should not be switched on until the flexible cables are in their sockets, and withdrawal of the socket should not be done with the current on. The main cables should be of ample size, flexible, insulated with vulcanised bitumen and armoured to resist wear and damage by falls of roof. The flexible trailing cable for the coal-cutter should have a strong cab-tyre sheathing, while the supply cables for the conveyor motor and the loader motor should be well insulated and protected by a strong and flexible wire armouring. All exposed parts of the equipment should be effectively earthed.

(iii) Coal-cutters should be provided with suitable gear to reduce the rate of cutting in very hard material, and the picks should be made of special steel to reduce the risk of extensive sparking while cutting in hard material. Coal-conveyors should be constructed so that they will have a free and easy movement without producing friction, heat and sparks.

(iv) During working hours, frequent examinations should be made with a safety-lamp for the presence of firedamp. The ventilating current should be maintained at the maximum quantity, and the face should have a treatment with fine stone-dust to reduce the risk of explosions of coal-dust.

(v) Shot-firing operations should not be carried out in coal or in ripping while coal-cutting and coal-conveying are in operation.

Tendency to Spontaneous Heating—Precautions in Working

6. How do you account for spontaneous heating being more prevalent in some coal seams than in others? What precautions should be taken in working seams liable to this danger? (25)

A. Spontaneous combustion depends on a number of conditions which might be prevalent in some seams and not in others, as follows:—

(i) According to available records, it is found that fires occur more frequently in deep seams than in shallow seams, the natural heat of the strata in the former being greater than in the latter.

(ii) The nature of the coal differs in different seams; in some cases it becomes friable after working, thus crumbling to powder form and therefore being subject to oxidation, while in others it remains firm and in block form.

(iii) The presence of iron pyrites in coal, in the finely disseminated form known as marcasite, together with moisture, results in the production of heat, and finally in fires in the waste workings of some seams.

(iv) The method of working seams of coal differs in different collieries and districts. In some cases, small coal may be left in the waste together with broken timber, and the oxidation of these leads to heating and fires. In other cases the method of working the coal and of building packs may be to blame; coal is left in the waste workings and air leaks through the packs; the result of this is crush, fractures of strata, grinding action, heat, and eventually fire.

(v) Faults might frequently be met with in some areas, thus producing bad strata and coal left unworked. The results of this would be similar to those given in the previous paragraph.

(vi) The composition of the coal differs in various seams; some coals contain much more humous bodies than others, and are more easily oxidised.

(vii) Finally, the atmospheric conditions might be different in the various coal-mining areas. The weather might be damp and foggy, and accompanied by frequent changes of atmospheric pressure, too small to be indicated by the barometer. The danger of heating is increased under such conditions.

Precautions.—The precautions to be taken in working seams of coal subject to spontaneous heating are :—

(i) Work the coal faces forward in a regular manner to reduce the danger of oxidation. Arrange for workings to have small districts of panel arrangement for longwall working. If many faults exist in the seam, and it is impossible to work all the coal, the retreating panel system of longwall working should be applied.

(ii) Install good and substantial packs to form roads, and stow the waste as completely as possible to prevent roof fractures and leakage of air. Exclude all coal, cuttings, and broken timber from the waste workings.

(iii) Good supervision is necessary to ensure that the foregoing details are carried out, the circulating air-current ample, the shot-firing conducted in a safe way, and the necessary precautions taken in detecting and dealing with firedamp.

(iv) The various districts of the mine should be treated regularly with fine stone-dust to reduce the risk of explosions.

(v) Samples of air and temperature readings should be taken at regular intervals in the return airways. The former should be analysed for dangerous gases and oxygen content. In this way the ratios $\frac{CO}{O_2}$ and $\frac{CO_2}{O_2}$ can be obtained and any increase noted.

(vi) A strict look-out should be maintained to detect heating in the early stages by moisture deposits on cool surfaces, musty smell, smell of paraffins, and gob-stink.

(vii) Prepared stoppings should be arranged in each district with the necessary material at hand for closing them. Water-pipes should extend to the lyes of the various districts with a connection to a suitable supply.

NOVEMBER 1932 EXAMINATION

Organisation for Dealing with Explosions and Fires

1. Describe fully the organisation you would establish, and give details of the equipment you would provide at a large colliery for dealing with: (*a*) an explosion, (*b*) fires underground, (*c*) fires on the surface. (25)

A. *Explosions.*—In case of an explosion occurring underground, the following organisation and equipment should be constantly maintained:—

(i) The colliery officials should be properly instructed as to their duties in case of emergency; the engineers should know what action to take in connection with examinations of plant; the store-keeper should have the necessary stores ready for use; the Chief Clerk should know what outside help is immediately required, such as rescue brigades, inspectors, doctors, etc.; the Under-manager and underground officials should be familiar with the details of rescue brigades and base parties.

(ii) Ambulance rooms on the surface and underground should be provided with proper equipment, and the trained men in charge of the same should be instructed as to their duties in case of emergency.

(iii) The persons to take charge at the surface in the absence of the Manager and other officials, who might be underground, should be instructed as to all the duties of such an office.

(iv) The person appointed to make suitable arrangements at the surface and underground for the convenience of the rescue brigades should be instructed as to the details of such work.

(v) The necessary rescue equipment, including birds and mice, should be constantly ready for use.

(vi) Special rescue plans should be prepared, and kept up-to-date, for the respective districts of the mine.

Fires Underground.—For dealing with fires underground the following details should receive attention, in addition to those already stated above :—

(i) Two sets of smoke helmets should be constantly available, and a number of men should be trained in their use.

(ii) The air-reversing arrangement of the fan equipment should be kept constantly in good working condition.

(iii) A good stock of the necessary materials required for building and backing-up stoppings should be kept at convenient positions on the surface and underground.

(iv) Apparatus should be constantly available for the collection and rapid analysis of samples of air, and a person trained in such work should be employed at the colliery.

(v) A system of water pipes, hoses and hydrants should be installed underground. See pages 233–4.

Fires on Surface.—For dealing with fires on the surface, water hydrants should be arranged, with suitable lengths of hose and jets, at convenient positions. Men trained in the use of the same should be constantly available, and frequent testing of such should be carried out. A suitable arrangement of fire alarms should be maintained. Arrangements might also exist for calling-up a central fire brigade in case of necessity. The air-reversing arrangement at the fan should be constantly effective and should be properly understood by the fan attendant. Finally, communication by telephone should be maintained between all districts of the mine and the surface.

Barrier for Retaining Water in old Workings

2. What are the principal factors which have to be taken into account in deciding the width of barriers to retain water safely in old workings ? (25)

A. The principal factors requiring attention in deciding the width of barriers to retain water in old workings are :—

(i) All old plans of the waterlogged workings should be properly examined to trace out such details as faces, faults, exploring headings, and sumps. A line giving a margin of safety should be designed on the plan, in advance of the above details, to allow for inaccuracy of the plans and a possible advance of the workings during a three months' period.

(ii) The position of any faults and dykes in the area being worked, the thickness and nature of the coal seam and the nature of the adjacent strata should be deciding factors in fixing the width of barrier from the designed safety-line already referred to.

(iii) The inclination of the seam of coal being worked, its depth from the surface, and the possibility of maintaining a safe standage for water, should likewise receive attention.

(iv) Consideration of the pumping plant available and its capabilities should not be neglected in coming to a decision on the width of barrier.

(v) Finally, surface items, including the position of old shafts and the depth of water contained in them, position of streams in relation to faults, and the nature of the surface beds, should not be neglected.

Recurrence of Spontaneous Combustion in Mines

3. Spontaneous heating occurs in colliery workings which have been free from such troubles for many years. What might be the cause of this heating, and what do you suggest might be done to prevent a recurrence ? (25)

A. The workings of a colliery might cause trouble owing to spontaneous heating in the following circumstances :—

(i) The nature of the coal might change from hard to soft, and thus give favourable conditions for oxidation.

(ii) An increased amount of " brasses," such as marcasite, or an increase in the fusain content in the coal, might make heating possible.

(iii) The area might become more troubled by the presence of faults and slips, and the nature of the roof may change so that more slips and breaks are produced in working, all of which tend to increase heat.

(iv) Some alterations may have been made in the method of working the seam of coal, whereby faces are not in regular line or in regular work ; buildings might be improperly constructed, thus allowing more roof breaks and air leakages through waste workings ; and care might not be taken to exclude coal and timber from the waste workings.

(v) A rearrangement of the method of working might also involve irregularities in the air passages near the face, thus producing higher and more fluctuating air velocities and pressures, instead of maintaining safe and uniform figures by having uniform areas.

To prevent a recurrence, the above details should be carefully taken into account, so that the following will receive adequate consideration :—

(i) The method of working the coal—to prevent as far as possible roof fractures, air leakages, and wastes containing combustible materials.

(ii) Arrangements should be made to have a good current of air circulating at the faces at a safe and uniform velocity. In this direction, systematic face arrangements for stripping coal and building of packs is important.

(iii) Changes in working arrangements might be necessary owing to changes in roof strength, faults, and the nature of the coal.

(iv) Temperature readings in the return airways and results of air analysis in the same roads should be properly recorded and examined to detect fires in the initial stages.

Properties of Mine Gases—Treatment of Persons Affected by Them

4. Write a short account of the physical, chemical and physiological properties of (*a*) firedamp, (*b*) blackdamp, (*c*) afterdamp. How do they generally occur in mines, and how would you deal with persons rendered unconscious by each ? (25)

A. The properties of the gases referred to above have already been given in the answer to a previous question under the heading of Ventilation ; see pages 99–101.

Firedamp is given off in a mine from the coal and its associated strata during the process of working the former ; also from slips in the strata and from faults. It might issue from cavities caused by falls of roof after the coal has been worked out, and from waste workings during periods of low and fluctuating atmospheric pressure. Persons found unconscious in this gas should be removed to a purer atmosphere, where the administration of oxygen and the application of artificial respiration can be carried on.

Blackdamp is produced in mines when carbon dioxide (CO_2) is formed at the expense of the oxygen contained in the air, thus leaving an excess of free nitrogen. CO_2 is produced by the burning of lights, breathing of men and animals, underground fires, oxidation of coal and other strata, and the decomposition of strata and timber. The treatment of persons found unconscious by reason of this gas should include removal to a better atmosphere for the administration of oxygen and the application of artificial respiration, as in the former case.

Afterdamp results from an explosion of firedamp or of coal-dust in a mine, the oxygen content of the air being mostly used up as a result of this action. The treatment of unconscious persons should include that already given for firedamp and blackdamp, with the following additions in the event of the presence of carbon monoxide (CO) :—Greater care is required under this condition, and, in addition to the methods used in the former cases, the affected persons must be kept warm and quiet until showing signs of recovery. After this, massage of the limbs should be started to induce circulation of the blood. A little walking exercise might then be resorted to. Removal of the patients from the mine atmosphere in this case should be accomplished only in easy stages until the surface is finally reached. In this way the danger of permanent injury to the lungs and sudden collapse by heart failure will be obviated. If a medical man is in attendance, he might give injections of about 50 c.c. of a 1 per cent. solution of Methylene blue while the patient is kept warm. Such injections have proved successful in America for bringing down the pulse rate and restoring normal breathing conditions.

MAY 1933 EXAMINATION

Limits of Inflammability of Firedamp—Precautions when Shutting off Fires

1. What are the limits of inflammability of firedamp under ordinary mine conditions, and how are they affected by the absorption of oxygen by the coal, and any other oxidisable material present ? What precautions should be taken when shutting-off gob-fires in a gassy seam to lessen the risk of an explosion ? (30)

A. The limits of inflammability of firedamp under ordinary mine conditions are 5·3 per cent. lower and 14·8 per cent. higher, the greatest pressure being produced with 9·8 per cent., this amounting to 102 lb. per sq. inch. When the percentage of firedamp is high and the percentage of oxygen is low, e.g. $12{\cdot}5\ CH_4 + 18{\cdot}13\ O_2 + 69{\cdot}37\ N_2$, incomplete combustion results in the following products: $0{\cdot}38\ CH_4 + 79\ N_2 + 6{\cdot}09\ CO_2 + 7{\cdot}74\ CO + 6{\cdot}7\ H_2$. Firedamp, carbon monoxide, and hydrogen are inflammable gases capable of combustion if a supply of oxygen is available.

When oxidation of coal and other material is taking place in a mine the percentage of oxygen will be low, CO_2 and **CO** will be produced, and this will result in the above limits being considerably altered. As the percentage of blackdamp increases from 10 to 37·5 per cent., the lower limit of inflammability of firedamp rises slowly and the higher limit falls rapidly until they coincide at about 6 per cent., when the air contains 16 per cent. of oxygen or more. Above 37·5 per cent. of blackdamp prevents explosion. If the air contains less than 12 per cent. of oxygen, no explosion of firedamp can take place. Nitrogen excess of 38·5 per cent., or 25 per cent. of CO_2, make firedamp non-inflammable. [S.M.R.B. experiments.]

When closing-off a gob-fire in a gassy seam of coal, care should be taken to build up explosion-proof stoppings as quickly as possible, with the return stopping a stage in advance of the intake stopping (see Fig. 107). This procedure will possibly prevent an explosion extending to the workings of the mine, and may also even prevent an explosion taking place. The percentage of oxygen in the closed-off area will fall rapidly, and the percentage of firedamp will rise quickly, thus reducing the risk of explosion. Air analysis during the process of building the stoppings, and afterwards at regular intervals, should be a guide as to the progress of the fire and the danger of explosion. The graph shown in Fig. 100 indicates the safe and unsafe ranges of composition for firedamp–oxygen mixtures. Other precautions necessary include the circulation of a good air-current near the stoppings, stone-dusting of the area near the stoppings to an excessive amount, and the provision of water-gauges at the stoppings to detect leakages of air into the affected area.

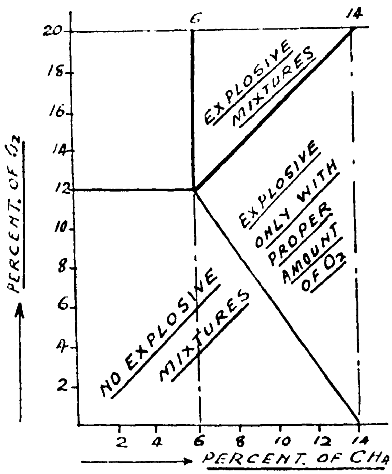

FIG. 100.—Graph of Firedamp–Oxygen Mixtures.

Approaching old Workings containing Water under Pressure

2. A longwall face is approaching old workings which end in a

pair of parallel narrow levels; the old workings contain water under pressure. Sketch and describe how you would approach, and drain the water from, the old workings. (25)

A. The precautions necessary as preliminaries in dealing with the above conditions have been fully stated in a previous answer; see pages 174–5 and 178–9.

The advancing longwall face should be stopped when within 75 yards of the safety margin drawn on the plan, and a pair of narrow headings should be advanced, say 10 to 15 yards, from which boring operations can be carried out to tap the old workings. Fig. 101 shows a systematic arrangement of boring suitable for the given conditions.

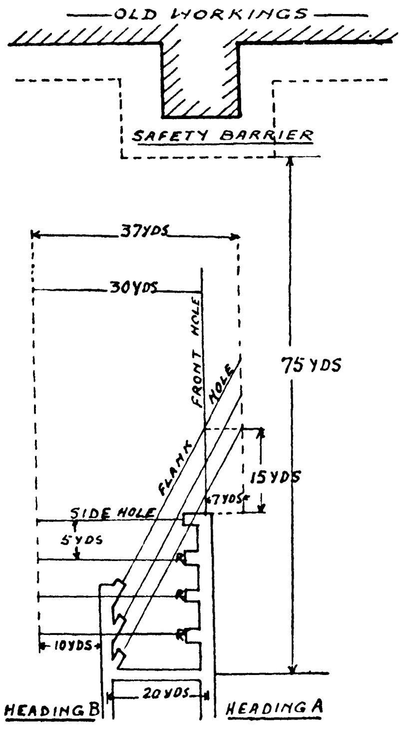

FIG. 101.—Advancing towards Old Workings—Position of Boreholes.

The narrow headings *A* and *B* are 20 yards apart, *A* being maintained in advance of *B*. At right-angles to *A*, roads marked *R*, 6 feet wide, are driven at intervals of 5 yards towards *B*, long enough to accommodate the boring apparatus used for drilling the holes. Right-angle bores from *A*, 30 yards long, prove the ground to the left and in advance of the heading *B*, so that this heading can be advanced with safety. From heading *B* similar short roads to those in *A* are driven at an angle of 35°, at which angle the flank holes are also drilled. These bores provide adequate protection for the advancement of level *A*. Advance boreholes are also drilled from the face of *A* to a distance of 60 yards, and when the heading advancement reduces this distance to 30 yards, new advance holes are commenced. The headings are advanced only 5 yards at a time, so that the protection afforded in this way is greatly in excess of that required by Section 68 of the Coal Mines

Act. A total area 37 yards wide by 60 yards long is effectively proved in the advance work.

The *Burnside Hydro Boring Apparatus*, driven by compressed air, might be used for boring the holes under safe conditions. Fig. 102 shows the construction of the apparatus, which consists of an air motor for rotating the rods, an air cylinder 6 feet long by 5 inches diameter fitted with piston, piston rod, crosshead, and push rod, the end of which latter is forked to fit a prop, so that when air is admitted to the cylinder the machine moves forward as the rods rotate. Speed is regulated by a hand-operated air release on the air-cylinder inlet. The holes are cleaned out by compressed air, or by water, as desired. The whole apparatus is fitted to a frame attached to tub wheels for easy transport. It is fitted with pressure-gauge, and stop-valves for shutting off gas or water. A tightening arrangement is included, consisting of a brass boring-tube about 16 feet long for the rods to pass through, and fitted alternately with 10 rubber rings and 12 brass rings to occupy about 4 feet of the tube. A loose iron sleeve, about 10 feet long, inside the brass tube, is forced by the tightening arrangement against the rubber and brass rings, thus forming a watertight joint. A hydro pump can be fitted to the rods for testing the boring-tube to a pressure of 300 lb. per sq. inch before boring starts.

After holing through, as described above, the water can be drained from the old workings by withdrawing the rods and fitting suitable pipes to the apparatus. The flow of water can be regulated by the main valve, and the water can be conducted by the pipes to a convenient spot for pumps to deal with it.

Dangerous Dusts in Mines

3. Name the two dangerous dusts which are produced in mining operations. How can the production of these dusts be minimised, and what should be done to safeguard the mine and workers ? Also to protect the health of the workers ? (25)

A. The two dangerous dusts produced in mining operations are coal-dust, produced by working, hauling, and screening of the coal ; and inert dust, produced by drilling shot-holes and by stone-dusting of roads.

The production of coal-dust can be minimised in mines as follows :—

(i) By having the screening plant placed at least 80 yards from the downcast shaft.

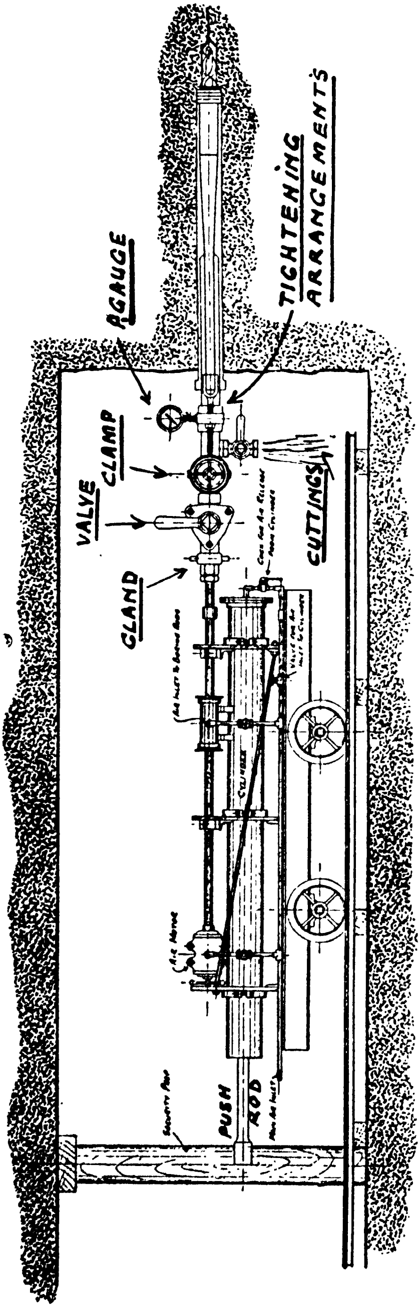

Fig. 102.—Burnside Hydro Boring Apparatus.

(ii) Installing slow-moving haulages and steel trams.

(iii) Working the coal without the use of explosives, and preventing excessive crush on the coal by good face supports and packs.

(iv) Installing de-dusting plant at inbye loading stations and in the vicinity of coal-cutting machines while working.

The production of stone-dust can be minimised by the use of some form of dust trap when drilling shot-holes and by having excessive dust deposits on roads cleaned up at regular intervals.

The mine and workers can be safeguarded as follows :—

(i) By treating the coal-dust deposits on roads and faces with stone-dust of a fine nature, to maintain a definite mixture according to the percentage of volatile matter in the coal and the percentage of firedamp in the air ; say 50 to 80 per cent. of stone-dust in the mixture. The stone-dust should pass through a 200-mesh sieve, and a highly hygroscopic dust should be avoided.

(ii) Taking every precaution in dealing with firedamp, and in shot-firing, to prevent ignitions of firedamp by blown-out shots, over-charged shots, badly placed shots, and breaks in the strata.

(iii) Having a good current of air continually circulating.

The health of the workers can be protected as follows :—

(*a*) By the use of some type of dust trap when drilling shot-holes ; see pages 152–3 and Fig. 89.

(*b*) Taking precautions to have stone-dust, as used for stone-dusting of roads, of such a nature as to prevent silicosis and fibrosis. Such dusts should not contain more than 30 per cent. of free silica. Sericite dust giving rise to fine fibrous particles should not be used. See pages 179–81.

Accidents when using Rescue Apparatus

4. Accidents have occurred to members of rescue teams when engaged in actual work underground, either through some fault in the apparatus, or some action or neglect on the part of the wearer. Set out the principal causes of such accidents, and the precautions that should be taken to guard against them. (25)

A. *Faults in Rescue Apparatus.*—Faults in the apparatus might be as follows :—

(i) The apparatus might not be properly airtight, and leakages of air into the apparatus from the surrounding atmosphere might take place, thus causing the wearer to inhale irrespirable gases.

(ii) The pressure in the oxygen cylinder might not be up to the required standard and thus endanger life by the supply giving out. The oxygen might not be of sufficient purity.

(iii) The main reducing valve might not operate successfully to give the necessary oxygen supply, and the by-pass valve might be ineffective, thus depriving the wearer of the necessary supply of oxygen.

(iv) The caustic absorbent might become caked by moisture, and thus not absorb CO_2 quickly enough, so producing danger of suffocation of the wearer of the apparatus.

(v) The relief-valve might not be effective, and thus cause a high pressure within the breathing bag and make breathing difficult.

(vi) The inhaling and exhaling valves might not be free and effective, and thus prevent easy breathing for the wearer.

(vii) Damage might be done to projecting parts of the apparatus by contact with road surfaces, e.g. main valve, tubes, breathing-bag, mouthpiece straps, and nose-clip.

Errors and neglect on the part of the wearer of apparatus might include the following :—

(i) Penetrating into places of small dimensions, less than 2 feet high by 3 feet wide, and becoming wedged.

(ii) Failure to couple-up the apparatus in good air and before entering the danger zone.

(iii) Separating from the team.

(iv) Ignoring the instructions of the Captain of the team.

(v) Wearing the apparatus for a period longer than 2 hours.

(vi) Failure to read the pressure-gauge and to give warning in case of low oxygen content.

(vii) Neglect to give warning in case of failure or accident to the apparatus.

(viii) Wilfully removing the mouthpiece or nose-clip in irrespirable atmospheres.

NOVEMBER 1933 EXAMINATION

Conditions under which Firedamp and Coal-Dust are Explosive—Scientific Research in relation thereto

1. Define the conditions under which firedamp and coal-dust are explosive. State what has been done by scientific research to make working operations safer with regard to (*a*) illumination and (*b*) electrical apparatus. (30)

A. *Firedamp.*—The conditions under which firedamp is explosive are as follows :—

(i) In ordinary air, 5·3 per cent. lower limit and 14·8 per cent. higher limit.

(ii) In air with 17 per cent. of oxygen, 5·8 per cent. lower limit and 10·6 per cent. higher limit.

(iii) In air with 12 per cent. of oxygen, 6·5 per cent. lower limit and 6·7 per cent. higher limit.

(iv) With low oxygen content and high percentages of firedamp, carbon monoxide and hydrogen are produced.

(v) Firedamp ranging from 2 to 80 per cent. can be exploded in a break by the compression due to the firing of a permitted explosive.

(vi) Firedamp and air within the limits given in (i) are explosive if blackdamp is present up to 37·5 per cent., and the oxygen content is not less than 16 per cent.

(vii) Nitrogen excess of 38·5 per cent., or 25 per cent. of CO_2 in air, makes firedamp non-inflammable.

Ignitions might be caused by open lights, sparks from strata, electrical machinery, shot-firing and gob-fires, when the temperature produced is not less than 1,200° F.

Coal-Dust.—The conditions under which coal-dust is explosive are as follows :—

(i) When the roads and faces contain deposits of fine and dry coal-dust, amounting to 3 oz. per cubic yard for fine dust, and 6 oz. per cubic yard for coarse dust, an explosion can be initiated by a small explosion of firedamp, by a bursting or ruptured electrical cable, or by blown-out, overcharged and undercharged shots. The air-current must be rich in oxygen.

(ii) If the air contains 1 per cent. of firedamp a dust cloud can be ignited by a flame of much lower intensity than mentioned above, such as already described under the heading of firedamp.

(iii) Stale dust contains more oxygen than fresh dust and it is highly inflammable.

(iv) The volatile content of the dust is important. Coals with high values are more dangerous than those with low values.

Safety Research.—Scientific research as regards *illumination* might be tabulated as follows :—

(i) Double 20-mesh gauzes of a height not less than 5 inches.

(ii) Enclosed burners with finer wicks and lighter burning oils.

(iii) Better lamp glasses and better methods of fitting.

(iv) Magnetic locks and safer methods of re-lighting lamps.

(v) Stronger construction of lamps to withstand blows.

(vi) Higher candle-power.

(vii) Electric hand- and cap-lamps of safe design and with better lights.

(viii) Fixed electric lights of more perfect and safe design for use at shafts, road junctions, loading stations and faces.

Scientific research as regards *electrical apparatus* includes the following :—

(i) Stronger insulation of all current-carrying parts and better earthing of all exposed parts.

(ii) Flame- and explosion-proof casings for electric motors and switchgear.

(iii) Automatic appliances instead of fuses, excepting for small plants and lighting.

(iv) Better construction of armouring for cables to reduce the risk of fracture and shorts. Trailing cables for machines of better design to resist wear and rough usage.

(v) Plugs and sockets designed to prevent disengaging the plug with the current on.

(vi) A general all-round improvement in construction and in the provision of safeguards against open flashing and explosions of firedamp.

Water-blast

2. Describe conditions under which an occurrence known as " Water-blast " might be expected. (25)

A. " Water-blast " might occur at a mine when water is being pumped from a vertical shaft connected with flooded rise workings, as shown in Fig. 103. Air and gases under the pressure head (h)

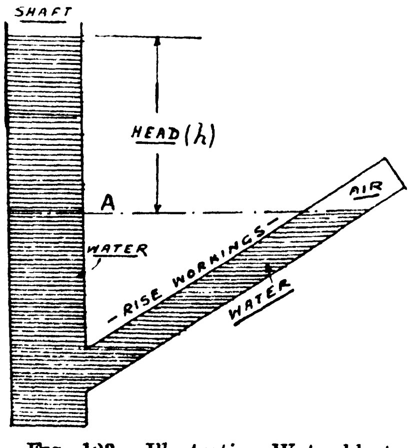

FIG. 103.—Illustrating Water-blast.

are trapped at the face of the workings. When the water-level in the shaft is approaching the same level as that in the mine, say at A in the sketch, the trapped air and gases might escape through the water to the shaft. In this way large volumes of air and gas will be liberated and pass into the shaft, and at the same time the water-level in the shaft will quickly fall to a lower point. This is known as "Water-blast." When the water-level in the shaft is approaching the line A, the water surface does not fall in proportion to the water pumped, and caution is then required by slowing down the pumping rate and thereby allowing the trapped gases to expand slowly without causing "water-blast."

Stone-dust Grinding Machinery, and Estimation of Quantity of Stone-dust required under Stated Conditions

3. Apparatus is to be installed at a colliery to prepare inert dust for the underground roadways. Describe briefly the principal features of suitable machinery. State the kind of material you would select for the dust, and if the output of coal is 2,000 tons per day from a dry and dusty seam 6 feet thick, state the approximate quantity of inert dust required to maintain the roadways in a safe condition. (25)

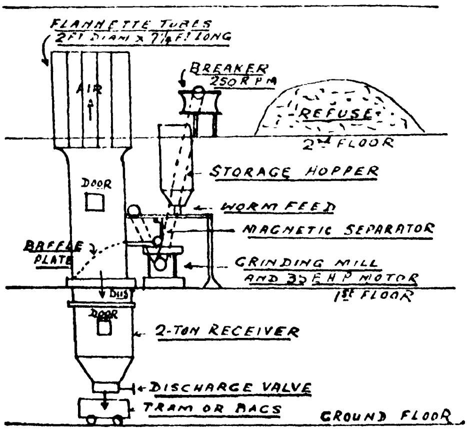

FIG. 104.—Kek Crushing and Grinding Mill for Inert Dust.

A. Fig. 104 shows a sketch of the *Kek crushing and grinding plant* for preparing inert dust at the rate of 1 ton per hour. Coal-measure strata, free from silica and sericite particles, are much used for preparing this dust. The material is stored on the second floor of the building and is broken up to small size by the breaker before passing to the storage hopper. The first floor is provided with an automatic feeding device to pass the broken stone to a magnetic separator, and to a grinding mill to reduce it to a fine powder (200-mesh sieve), before passing to the receiver. The dust is loaded into tubs on the ground floor ready for despatch to the mine. Air escapes from the receiver by way of a number of flannette tubes, as shown in the sketch.

In a dry and dusty seam 6 feet thick, as much as 5 tons of inert dust might be required every 24 hours to keep the roads in a safe condition. Much will depend on the method of working, haulage, tubs, and the de-dusting carried out at loading stations. The correct amount of inert dust required depends on the result of analyses of the dust deposits on the various roads.

Spontaneous Heating—Suitable Method of Working Seam —Dealing with a Case of Heating

4. Make a sectional sketch, with dimensions, of a seam of coal in which spontaneous heating might occur. Sketch and describe a method of working the seam to reduce the danger of spontaneous heating; and in the event of heating taking place, explain how you would deal with it. (25)

A. Fig. 105 shows a section of a seam liable to spontaneous heating owing to the top stone being an oily bat, and the bottom composed of about 3 inches of soft strata containing sulphur.

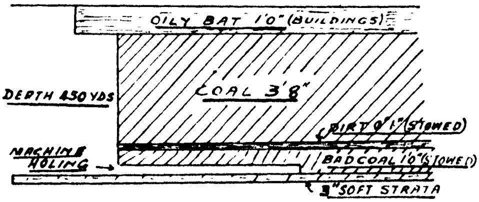

FIG. 105.—Section of Coal Seam liable to Spontaneous Heating.

Fig. 106 shows a panel system of working the seam to reduce the risk of heating. In connection with this method, bricks of a soft nature, and sand, are included in the packs to reduce air leakages. Sand dams are built on the return side of all gate roads. The packs consist of: bind + 4 inches of dry brick wall to within

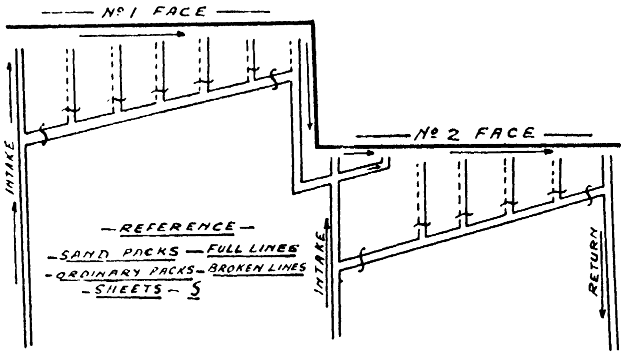

FIG. 106.—Working by Sand Packs to Prevent Spontaneous Heating.

1 foot of the roof + 9 to 12 inches of sand or flue dust + retaining walls of slack packing.

When gob-stink is detected, it is usually traced to some unusual condition, and a further local sanding usually stamps out the heating. In working the coal, all timber is withdrawn from the waste workings ; no coal pillars are left unworked ; doors are set up as far as possible from the face ; and a good current of air is circulated with a low water-gauge by duplication of airways.

Carbon Monoxide : Formation, Properties, Precautions

5. How is carbon monoxide usually formed in coal mines ? What are the characteristics of this gas ? How is it usually detected, and what precautions should be taken when its presence is suspected ? (25)

A. Carbon monoxide is formed in coal mines by shot-firing on an extensive scale, blown-out shots, oxidation of coal and gob-fires, heating of engine parts and burning of lubricating oils, cracking of lubricating oils during air-compression, and explosions of firedamp and coal-dust.

The characteristics of carbon monoxide include the following :—

(i) No colour, taste, or smell.

(ii) Non-supporter of life and combustion.

(iii) Combustible in air and oxygen with a characteristic blue flame.

(iv) Small percentages, such as 0·1 per cent., are highly poisonous and cause death in 2 hours.

(v) Combines with the blood and reduces the oxygen carried by it; the process might be compared with slow drowning. Has a great affinity for the red corpuscles of the blood, 300 times greater than their affinity for oxygen.

(vi) Very difficult to detect in mines.

The detection of carbon monoxide in mine air being difficult, the use of small birds and mice is generally necessary for this purpose, as they show symptoms of poisoning in **CO** sooner than do human beings. Small birds leave the perch in the cage and spread out their wings when affected by **CO**. Mice become sluggish in their movements and finally sprawl owing to loss of leg power. Human beings experience a feeling of pressure and throbbing in the head, a ringing or roaring in the ears, a feeling of weakness in the knees, and a dulling of the mind.

The precautions to be taken when carbon monoxide is suspected in the mine air should include the stopping of work at once, with as little excitement and exertion as possible, and withdrawal to a purer atmosphere without delay until the mine atmosphere has been properly tested. Persons suffering from dullness of mind should be carried out to a safer atmosphere. Unconscious persons should be removed to a safe atmosphere, and revived by an oxygen reviver of the "Carbogen" type, combined with the application of artificial respiration. All persons affected by carbon monoxide should be attended to by a medical man as early as possible.

MAY 1934 EXAMINATION

Causes of Colliery Explosions—Precautions necessary to Guard against Them

1. Specify the possible causes of colliery explosions. Describe fully the precautions that should be enforced to guard against the risk of explosions. (30)

A. Colliery explosions can be caused by any of the items included in the following list, which also includes the precautions necessary to prevent explosions.

(i) *Shot-firing.*—Shots not properly placed, causing over-charged, under-charged and blown-out shots; shots with detonators wrongly placed in cartridges; successive shots fired without examination for firedamp; shots fired where breaks exist in the hole, where adequate treatment of coal-dust has not been carried out, and where insufficient stemming has been used.

The precautions necessary should include: the appointment of a competent person to fire shots; to enforce the requirements of the Coal Mines Act and Regulations strictly; and to see that the above irregularities do not take place.

(ii) *Open Lights and Defective Lamps.*—Open lights and defective lamps igniting firedamp; lamps tampered with and exposing the flame; damaged lamps not being extinguished.

Precautions should be taken to limit the use of open lights to safe parts of the mine; to install good safety-lamps in all parts where firedamp is given off; and to have good supervision governing the use of safety-lamps in a mine.

(iii) *Electrical Appliances.*—Bursting of power cables owing to weak insulation and rough usage; rupturing of cables by falls of roof; open sparking at motors, switchgear, controls, plugs, etc.

Constructional details should receive adequate attention to have stronger insulation of all current-carrying parts, better construction of armoured cables to reduce the risk of rupture and shorts, and good design of trailing cables to resist wear and rough usage. Explosion-proof casings should be fitted to all motors and switchgear. Special protective devices should be used instead of fuses. Plugs and sockets should be designed to prevent disengaging with the current on. Better earthing of all exposed parts should be included in the arrangement of plant.

(iv) *Friction.*—An intensive frictional spark at a high temperature for a comparatively long period is capable of causing an explosion; such sparks might be produced by falling sandstone, defective machinery, haulage friction, and coal-cutter picks in quartzitic sandstone.

The precautions to be taken should include good roof supports, well lubricated machinery, properly fitted haulage appliances which receive adequate attention, and the use of special alloy steels for coal-cutter picks to reduce the risk of sparking.

(v) *Fires.*—Accidental fires, gob-fires, and fires caused by neglect on the part of the workmen.

Proper design of workings to reduce the risk of fires, and good supervision by the officials of the mine, are factors necessary to prevent explosions by fires.

(vi) *Underground Signalling.*—Failure of wires owing to bad insulating material; fusing of wires of inadequate dimensions; and sparks at signalling stations from unprotected devices, might cause ignitions of firedamp. Good constructional details with good insulation, and protected or flameproof casings, should be included in the precautionary measures adopted.

(vii) Finally, great care should be exercised in dealing with firedamp in the mine, and in rendering coal-dust deposits safe by the application of inert dust at frequent intervals.

Spontaneous Heating in Coal Pillars and Goaves of Longwall Workings

2. Set out the conditions which may give rise to spontaneous heating (*a*) in coal pillars, (*b*) in the goaves of longwall workings. What can be done to reduce the liability to heating in each case ? (25)

A. Spontaneous heating might occur in coal pillars under the following conditions :—

(i) Where the coal is of a soft and friable nature and the sides fall and crumble.

(ii) When coal is left at the loose ends of lifts in extracting pillars.

(iii) When stoops of coal are left unworked near faults, and the coal is subject to crushing and grinding action.

(iv) When timber is not withdrawn after the coal is worked, such as broken props and softwood chocks.

(v) By the bulk of the air-current passing through the goaf.

The liability to heating might be reduced by having good roof and side supports in all roads, and by instituting a regular cleaning-up of coal in the roads. Sand packing might be applied in all roads used for haulage and ventilation. The coal should be worked off completely in lifts and near faults by adopting good systematic timbering and good packing. All timber should be recovered from the goaf, useless material being sent to the surface. An improved system of ventilation should be applied to prevent the air-current circulating through goaves.

Spontaneous heating might take place in the goaves of longwall workings as follows :—

(i) By working the coal in an irregular manner, thereby causing bad roof, crushed coal, and unworked coal.

(ii) Leaving cuttings, bad coal, and timber in the goaf.

(iii) By building loose and insecure packs to protect the roads, thus causing crushing and grinding of the strata, friction, heat, and the leakage of air through the goaf.

The liability of the seam to spontaneous heating might be reduced by having a proper systematic method of working the coal to give good roof conditions and complete coal extraction ; by excluding all inflammable substances from the waste or goaf ; by building good packs to prevent crushing, heating, and air leakage ; and by having a good ventilating current passing along a definite path.

Present state of Knowledge with regard to the Dangers of Coal-Dust

3. Give fully, but concisely, an account of the present state of knowledge with regard to the dangers of coal-dust and the measures which should be taken to ensure safety. (25)

A. *The dangers of coal-dust* in mines might be summarised as follows :—

(i) The explosive property of fine and dry coal-dust, in the entire absence of firedamp, has been proved by experiments on a large scale at Buxton.

(ii) Dangerous mine dusts are those containing a high percentage of volatile matter, but it is the combustion of the solid particles, and not the distilled volatiles, that forms the explosion.

(iii) Stale coal-dust is more dangerous than fresh coal-dust, it having absorbed a certain amount of oxygen from the air.

(iv) Both fine and coarse dusts are dangerous in mines and either can be ignited in the entire absence of firedamp.

(v) If a small percentage of firedamp, say 1 per cent., exists in the mine atmosphere, coal-dust is more easily ignited by open lights.

(vi) The quenching effect of fine inert dust, when mixed with the coal-dust, to prevent dust explosions and to arrest explosions that have already started, has been effectively demonstrated at Buxton. Inert dust, in excess of the 50/50 mixture, is necessary for safety when the coal-dust is rich in volatiles, and when the air-current contains firedamp. See also pages 170–1.

Safety Measures.—The measures necessary to ensure safety are as follows :—

(i) Reduce the dust production as much as possible by installing dust-collecting plant at all points where dust is made, such as at screening plants, where coal-cutting operations are going on, in the vicinity of coal-conveying plant, and at inbye loading stations.

(ii) Select conveyors for face work which will produce a minimum of dust.

(iii) Treat all coal-dust deposits regularly with fine inert dust to render them safe, after removing as much of the coal-dust as possible.

(iv) Have a regular and systematic method of cleaning-up dust deposits.

(v) Enforce strict rules regarding the methods of dealing with firedamp to avoid ignitions ; take all necessary precautions when firing shots ; have good electrical plant installed with explosion-proof casings and good protective devices; avoid gob-fires if possible, and have arrangements made for quickly detecting fires and quickly sealing them off; install good face machinery and see that it gets the necessary attention to avoid friction and sparking. Finally, insist on cleanliness and care being exercised at all points where accidental fires might occur in a mine, and provide adequate appliances for extinguishing fires.

Exploring Mine Workings after an Explosion

4. What are the principal dangers to be guarded against by an underground exploring party immediately after an explosion and before the arrival of the rescue brigade? How should they proceed, and what should they take with them? What evidence should ultimately be looked for to locate the point of origin? (25)

A. *The principal dangers* to be guarded against by an underground exploring party are as follows:—

(i) A deadly mine atmosphere containing **CO**, and possibly hydrogen and firedamp Also accumulations of thick black smoke in the mine roads, which might be circulated and cause suffocation of the party.

(ii) Active fires might exist in the mine, and there is the risk of a second explosion of the gases already named when the ventilation is restored.

(iii) Dangerous fumes might be produced by fires in active progress.

(iv) Owing to roof supports being displaced, there is a great danger of falls of roof taking place, and thus cutting off the safe retreat of the party.

The procedure to be followed by the exploring party should include the following:—

(i) Examination of shaft and winding appliances, and testing of the air in the shafts for the presence of **CO**.

(ii) A small area, including the shaft landings, junction of splits, and junction of the returns from the various mine sections, should be thoroughly examined. All the party should not leave the cage and shaft bottom until a preliminary examination has been completed and a base established.

(iii) If there is no evidence of an active fire, the ventilation should be restored as quickly as possible.

(iv) The possible extent of the explosion, and the districts mostly affected by it, might be determined before the rescue brigade starts operations.

(v) Rescue operations might be started into districts not affected by the explosion, for the purpose of rescuing the workmen.

(vi) The work undertaken by the party should be more of a preparatory nature to facilitate the work of the rescue brigade when it is ready for operations.

The following *appliances* should be carried by the party :—

(i) A good safety-lamp for each member and two or more electric hand lamps.

(ii) A cage containing small birds for detecting the presence of CO.

(iii) Fire extinguishers.

(iv) An oxygen reviving apparatus.

(v) An ambulance box properly fitted.

A stretcher party might also be included.

The part of the question relating to the point of origin of the explosion has been answered in dealing with a previous question; see pages 176–7.

Dealing with a Gob-Fire in a Fiery Mine

5. A gob-fire occurs unexpectedly in a district of a mine in which large quantities of firedamp are given off. No preparations have been made, and there is risk of explosion. What would you do ? (25)

A. All work should be stopped immediately in the affected district, and all the workmen should be withdrawn from it without delay. If any other sections are at work on the return side of the affected one, they should be treated in a similar way. Fighting the existing fire is an impossible proposition under the given conditions, and arrangements must be made for quickly and effectively building-off the district by erecting explosion-proof stoppings. The materials for the stoppings should be obtained from refuse found in the mine.

Fig. 107 shows by a plan view how an explosion-proof stopping might be constructed with steel props, stone packs, and refuse.

Steps should be taken to minimise the effect of an explosion as follows :—

(i) The mine should be thoroughly stone-dusted in all parts.

(ii) Good ventilation should be continued in every district.

(iii) Building material should be collected at convenient places for use in the event of damage to the stoppings.

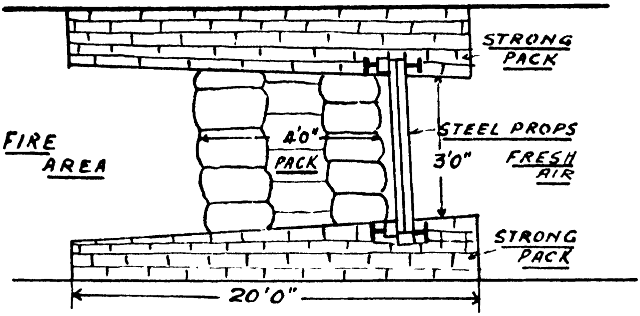

FIG. 107.—Plan of Explosion-proof Stopping.

(iv) A systematic method of sampling and analysis of the atmosphere at the back of the stoppings should be undertaken to ascertain the progress of the fire and to detect the possibility of an explosion taking place.

NOVEMBER 1934 EXAMINATION

Underground Fire and Reversal of the Ventilation

1. Describe a hypothetical case of an underground fire where a reversal of the ventilation would be advisable. When, and by whom, should the order for the reversal of the ventilation be given? (25)

A. A fire has started in an engine-room situated about 100 yards from the downcast shaft, in the intake airway at the top of a stone mine going to the dip, and it is beyond the first stage of control. No direct connection to the return airway is available immediately inbye of the fire, and short-circuiting of the air-current to the return airway is not possible. Men are at work in the district at the inbye end of the drift and they are in danger of suffocation by the fumes from the fire. Under these conditions the fire has already been partly isolated by shutting off the air-current to the engine-room, and reversal of the ventilation is necessary to rescue the men by the return airway and upcast shaft. After this has been accomplished, the ventilation should be restored to its original direction, to allow of the fire being extinguished by water or other suitable means.

The order for reversing the air-current should be given by the person knowing all the details in connection with the fire and the mine, that is, the Manager, Under-manager, or the official responsible for the safe working of the mine in their absence.

Working a Seam below Extensive Old Workings containing Water

2. At a depth of 200 yards from the surface there are extensive old workings containing water lying 100 yards above a seam of coal which is about to be worked. What precautions should be taken in working the seam below the old workings, the strata being intersected by a fault? (25)

A. The pressure due to the head of water, in the seam to be worked, will be 150 fathoms × 2·6 or 390 lbs. per sq. inch, and this is a very dangerous pressure.

The precautions to be taken should include the following:—

(i) The plan of the old workings should be recovered and the fault accurately traced out on the plan of the workings of the lower seam.

(ii) If any old shafts are situated in the waterlogged area, they should be pumped dry by means of borehole-type turbine pumps.

(iii) A surface survey of the waterlogged area should be undertaken to get the position of the fault and its details, and to get the position of any streams of water. A diversion of such streams might reduce the quantity of water reaching the old workings.

(iv) All working faces in the lower seam should be stopped when water is detected from the roof, and boring in advance should be applied to such faces.

(v) The workings within 50 yards of the fault on both sides should be stowed solid to prevent movement of the strata.

(vi) A barrier at least 20 yards in width should be left unworked at each side of the fault.

(vii) All roads driven through the barrier, for ventilation and haulage, should not exceed 8 feet in width and should be preceded by boreholes at least 5 yards in advance, to detect the presence of water, and to give safe working conditions.

Constructing a Dam in a Vertical Shaft

3. It is proposed to construct a dam in a vertical shaft, which is 14 feet in diameter within a lining of 9-inch brickwork, to prevent

water passing down the shaft to lower seams. Sketch and describe the construction of a dam capable of withstanding the pressure due to 120 yards' head of water. (25)

A. Fig. 108 shows the details of construction of a two-part cement concrete dam 72 feet thick, suitable for the conditions given in the above question. Such a dam would be constructed in stages between three successive walling cribs. Starting at the bottom crib, a suitable platform would be erected with H-beams, 8 × 6 inches section, covered with 9 × 3-inch white pine deals. The two wedge-shaped portions of the dam would be made up of 1 cement, 2 sand and 3 aggregate, the sides of the shaft being dressed back by hand if possible. To allow of access to the shaft sides, holes would be cut in the brick lining of the shaft 6 feet wide tapering to 6 feet high, leaving at least 3 feet of brickwork lining between the holes to support the lining. These holes would allow of the strata being removed from the shaft sides, and the passage of the concrete to build the dam.

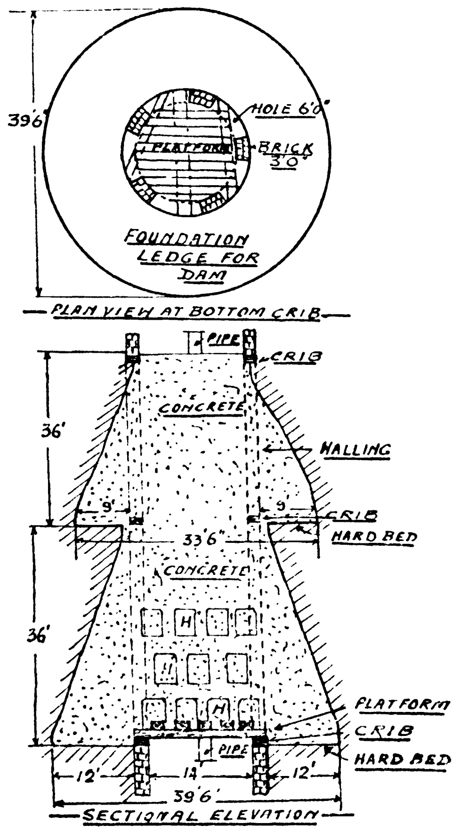

FIG. 108.—Construction of Cement-concrete Dam in a Circular Shaft.

After the first 6 feet have been filled in, the next series of holes 6 by 6 feet would be cut in the lining and the work continued as before. The work will thus proceed in this way, in 6-foot lengths, until the dam is finished. When completed, the shaft lining will not have to support the dam in any way. It would be advisable to have a pipe running through the centre of the dam to allow water to drain off during its construction. It would be properly plugged after completion of the dam. All dimensions are included in the sketch.

Air Samples and Analysis from Heated Area

4. A sample of air has been taken from the pipe leading through the stopping of a district which has been sealed-off in consequence of recent spontaneous heating. Give approximately the figures of an analysis of such a sample, and state in what way further samples and analyses would indicate an increase, or diminution, of the heating. (25)

A. Analysis of sample:

CO_2	0·42
CH_4	4·70
O_2	18·98
N_2	75·87
CO	0·03
CO_2/O_2 ratio	2·21
CO/O_2 ratio	0·16

Analysis showing *diminution* of heating :—

CO_2	0·20	Oxidation lessening. CO_2 and CO reduced. O_2 greatly reduced. CH_4 greatly increased.
CH_4	69·79	
O_2	6·00	
N_2	24·00	
CO	0·01	
CO_2/O_2 ratio	3·33	
CO/O_2 ratio	0·17	

Analysis showing *increase* of heating :—

CO_2	1·54	Oxidation increasing. CO_2 and CO increased. O_2 slightly reduced (air leakage). CH_4 slightly increased. Danger of explosion.
CH_4	9·00	
O_2	17·25	
N_2	72·16	
CO	0·05	
CO_2/O_2 ratio	9·00	
CO/O_2 ratio	0·30	

Rescue of Workmen Trapped by Accumulation of Firedamp

5. A large fall of roof in the intake haulage road near the coal face has caused an accumulation of firedamp on the return side of a working place in which several men are employed. The place can only be approached by passing through the gas by way of the

return with rescue apparatus. How would you proceed to rescue the men ? (25)

A. The apparatus generally available at a colliery for rescue-work includes an oxygen reviver, a self-contained breathing apparatus or a smoke helmet having a range of 40 yards. This equipment is not sufficient to carry out the necessary rescue work under the given conditions. The rescue brigade from a Central Station should be called upon to undertake the work. One brigade equipped with stretcher, oxygen reviver, and half-hour " Salvus " apparatus, would penetrate the danger zone and rescue the men by fitting the reviver or " Salvus " apparatus to them as circumstances required. A second brigade should be standing by at the fresh-air base to receive the workmen brought out by the first brigade.

JULY 1940 EXAMINATION

(*Five questions only to be answered.*)

Sealing off a District of Workings after a Firedamp Explosion

1. After an explosion of firedamp, a fire breaks out which cannot be reached owing to a large fall in the intake airway. The fire extends, and it becomes necessary to seal the district. What places would you select for the stoppings ? How would you build the stoppings, and what precautions should be taken for the safety of those engaged on the work ? (30)

A. The places selected for the building of the stoppings should be as far from the seat of the fire as is reasonably possible, so that good firm strata may be available, free from faults and roof breaks. Under similar conditions, stoppings have been erected as far as 900 yards from the seat of the fire. Such a position could be adequately ventilated by fresh air and would be easily accessible for the quick transport of the necessary building materials required to complete the work quickly. Air samples could be collected from the closed-off area at frequent intervals without danger, and water-gauge readings could be obtained at the same time. In the event of another explosion taking place near the seat of the fire, the workmen engaged in building the stoppings would be in comparative safety.

In building the stoppings at the point suggested above, materials might be quickly obtained for the construction of explosion-proof stoppings as described on pages 213–14 and in Fig. 107. Care should be taken, however, to have an air-sampling pipe built into the stopping to an extent not less than 9 feet into the fire area. Such a pipe could also be used for taking water-gauge readings, which should be positive. If it is not possible to get the necessary materials quickly for the above construction, asbestos bags 20 by 14 ins., half filled with sand or flue dust, might be used to build airtight stoppings. Flue dust is considered more suitable than sand owing to its heat-resisting properties. For work of this kind it is desirable to call in at least two teams of Rescue Brigade men from a Central Station, because they are properly trained in the construction of such stoppings. If this could not be arranged, the Rescue Brigade should be in attendance at all times with the necessary rescue appliances.

Safety Precautions.—The precautions to be taken for the safety of the workmen engaged in building the stoppings should include the following :—

(i) A good current of fresh air should be available at the spot.

(ii) Liberal stone-dusting should be done as far as possible in the fire area and near the position of the stoppings on the outbye side.

(iii) Safety-lamps and electric lamps should be used.

(iv) The return-side stopping should be closed a little in advance of the intake-side stopping in order to keep up a positive pressure on the intake side and to reduce the risk of explosion.

After the stoppings are completed, air samples can be taken from behind the stoppings, and subsequent analysis would show the progress of the fire and the possibility of dangerous conditions developing. See page 217.

Reconditioning of Cast-Iron Tubbing in Old Shafts

2. A pair of old shafts, each 250 yards deep, one 10 feet diameter, the other 8 feet diameter, and 5 yards apart, are lined with cast-iron tubbing to a depth of 80 yards from the surface. The mine was ventilated by a furnace for many years, and the tubbing has become weakened by corrosion and cracked in several places, allowing much water to flow into the shafts. How would you deal with the situation to render the shafts safe and serviceable ? (25)

A. The shafts referred to in the above question might be permanently reconditioned in a quick and efficient manner by the application of the Cementation process, as already described; see pages 37–9 and Figs. 25 and 26. This method of sinking and lining of new shafts has been applied during recent years to strengthen cast-iron tubbing in old shafts, and at the same time to prevent water leaking into such shafts.

The old segments of tubbing, especially in the upcast shaft, should be properly examined for cracks, and all corroded metal carefully removed until a reasonably sound metal is found in which to make connections for cementation in the plug holes which pass through the brackets of the tubbing. Starting near the base of the tubbing, the selected number of plug holes should be tapped and connections made with the cement-pumping plant for injecting the liquid cement to the back of the tubbing. A similar number of open holes should be provided as a sufficient safety-release of water from the tubbing during the injection process, to safeguard the tubbing against high pressure. In this way the pressure on the tubbing should not exceed the existing pressure plus the cementation pressure. It might be necessary to use a fairly thick liquid to start with, gradually passing on to a thinner liquid and chemicals to effect a final stoppage of the water entering the shaft.

To prevent the old tubbing being displaced by the pressure of the cementation process applied as suggested, especially where cracks are detected in the segments, a temporary strengthening of such segments might be carried out by using mild-steel rings similar to those applied for the temporary lining of sinking pits. See Fig. 4.

Precautions in Working Coal under Quicksand and Moss —Undersea Workings

3. What precautions should be taken when rise workings (not undersea) are approaching the outcrop of the seam under irregular deposits of quicksand and moss? In undersea workings, what percentage of coal may be worked with safety in seams averaging 6 feet in thickness, at depths below the sea bottom of 200 yards, 300 yards and 400 yards respectively? (25)

A. The precautions necessary in working coal under quicksand and moss should include the following:—

(i) The nature and thickness of the soft beds, and the thickness of strata between these soft beds and the workings, should be ascertained by boring at a sufficient number of points.

(ii) If the thickness of the intervening strata is less than 60 feet or 10 times the thickness of the seam, whichever is the greater, work must be stopped, except for the preservation of the mine. Work must not be continued except by consent of the Secretary of State, who will refer the question to a committee composed of the District Inspector of Mines, representatives of management and representatives of workmen. If disagreement arises, the management can refer the matter to another tribunal provided for in the Coal Mines Act (C.M.R.A.) for settling disputes.

[The thickness of cover referred to above should be obtained in systematic fashion by boring from the surface and by levelling processes on the surface and underground. This thickness of cover and depth of the soft beds should be determined accurately in advance of every area of workings. The area under which workings are being extended should be marked out into squares of, say, 50 yards side, and soundings should be taken by boring at the centre-point of each square. Surface and underground levels with the dumpy level should be determined at the same points. In this way the thickness of the solid strata above the coal seam, at the various points, could be determined and marked on the working plan.]

(iii) The position of any faults in the area of workings should be carefully noted, and the roof supports should be strengthened at such points to prevent roof breaks.

(iv) Working districts should be arranged as small as possible.

(v) It would be a decided advantage to work by longwall with solid stowing of the waste workings, thus reducing the danger of producing extensive roof fractures and falls of roof.

Undersea Coal.—In working undersea coal, a number of important factors must be taken into consideration, such as the nature of the overlying strata, the hardness of the coal, the presence of faults, and the method of working the coal. The following are the recommendations made by Mr. W. Foster Brown, Chief Mining Adviser to the Commissioners of Crown Lands, in giving evidence before the Water Dangers Committee in 1927.

(*a*) *Conditions applying in the Firth of Forth area of Eastern Scotland.*

(i) All undersea working of coal must be stopped when the thickness of strata covering a seam being worked is reduced to 270 feet. In all undersea workings, the issue of salt and brackish water must be reported immediately to the Lessor or to his engineer.

(ii) *Conditions of Working*: see Table below.

Thickness of Coal Seam.	Depth of Cover.	Method of Working.	Details.
Up to 2 ft. 6 ins.	270 ft. & under 360 ft.	Longwall	Solid Stowing of waste. If the ground is not already proved in other seams, an exploring heading must be driven 50 yds. in advance to get the position of faults. A barrier of 10 yds. must be left if the throw of the fault exceeds 30 ft., or where the sides of the fault are more than 2 ft. apart. Narrow roads through the barrier must be driven 12 ft. wide and in pairs 22 yds. apart, the respective pairs being 100 yds. apart.
Over 2 ft. 6 ins. & up to 4 ft.	Over 270 ft. & under 360 ft.	Pillar & Stall	No pillars to be removed. Percentage of coal worked to be determined by Lessor.
	Over 360 ft. & under 810 ft.	,, ,,	Solid Stowing if pillars removed or if worked by Longwall.
4 ft. to 4 ft. 6 ins.	Over 270 ft. & under 480 ft.	Pillar & Stall	No pillars to be removed. Percentage of coal worked to be determined by Lessor.
	Over 480 ft. & under 810 ft.	,, ,,	Pillars removed, or Longwall if Solid Stowing.
	810 ft. and over	Pillar & Stall or Longwall	Pillars removed if Lessor agrees. Longwall without restriction.
4 ft. 6 ins. to 7 ft. 6 ins.	Over 270 ft. & under 810 ft.	Pillar & Stall.	No pillars to be removed. Size of pillars to be determined by Lessor.
	Over 810 ft. & under 1080 ft.	,, ,,	Pillars removed if approved by Lessor. Longwall with Solid Stowing.
	1080 ft. & over	Pillar & Stall, or Longwall.	Without restriction.
7 ft. 6 ins. and over.	Under 1080 ft.	Pillar & Stall	Conditions to have written approval of Lessor.
	1080 ft. & over	Pillar & Stall, or Longwall	Pillars removed by permission of Lessor. No restriction of Longwall if Lessor approves.

(*b*) *Conditions applying in Durham, Northumberland and Cumberland.*

(i) No workings are allowed, without consent of Lessor, where thickness of cover is reduced to 270 feet.

(ii) No longwall workings where seams exceed 4 feet thick. Exploring headings must be driven in advance of the working face.

(iii) There must be left unwrought barriers of coal 44 yards wide against north and south boundaries and 100 yards wide against east boundary.

(iv) Dams must be constructed to separate undersea and other workings.

(v) In working under old workings, a barrier of coal 100 yards wide must be left all round the area, with three penetrating headings only through it 22 yards apart and 7 feet high by 8 feet wide. Two dams must be erected in the headings, one for quick closing.

(vi) Winning-out places driven alongside the barrier must have boreholes in advance and sufficient flank holes into the barrier.

The above details are given as a guide in answering the latter part of the question. The percentage of coal worked with safety in seams averaging 6 feet thick might be as follows :—

Depth of Solid Cover.	Method.	
600 feet.	Pillar-and-Stall.	Work 35 per cent. of coal and leave the pillars.
900 feet.	Pillar-and-Stall.	50 per cent. of coal worked. Pillars to be extracted only if approved by Lessor or his engineer.
,, ,,	Longwall.	Work all the coal and have solid stowing of the waste or goaf.
1200 feet.	Pillar-and-Stall.	Work 65 per cent. of coal. Pillars to be extracted only if approved by Lessor.
,, ,,	Longwall.	No restriction if 75 per cent. of the waste is stowed.

Emergency Door for Stopping Sudden Inrush of Water

4. A heading, with the necessary boreholes, is being driven towards old workings which contain a dangerous accumulation of water. Make a sketch, with dimensions, of a bulkhead or emergency door, and show how it should be erected in the heading, so that it could be closed quickly in the event of a sudden inrush of water. (35).

A. The above subject has been dealt with previously in the answer to a similar question; see Fig. 94, and pages 178–9.

Recommendations—Water Dangers Committee Report 1927

5. State what you know of the recommendations included in the report of the Water Dangers Committee, which was published in 1927. (25)

A. The recommendations referred to above include details of " Plans," " Barriers," " Dams " and " Approaching Old Workings." The following is a tabulated list of such recommendations from which a suitable answer might be given.

(A) **Plans.**—(i) Suggestions to Secretary of Mines on the plans of abandoned mines: a new catalogue of such plans is now available.

(ii) The reliability of old plans is very limited owing to omissions in extent, surface correlation, variations of magnetic meridian, shrinkage allowance and deterioration. Plans made before 1887 should not be taken as accurate.

(iii) Lessor should reveal to his Lessee all plans and other information in his possession.

(iv) All roads in mines which have ceased to work should be surveyed within one month, or earlier if the road is likely to become inaccessible.

(v) The following improvements in plans are recommended:— Separate plans should be made of every seam separately worked. True North and Magnetic North should be shown on all plans, the latter should be checked at least yearly. Plans should be marked in squares not more than 20 chains side. Ordnance Datum, without addition or subtraction, should be used to indicate levels on plans. The owner, agent or manager should notify the Divisional Inspector of Mines when workings are within 100 yards of the boundary. Duplicate plans should be provided for use in case of fire, accident, etc.

(B) **Barriers.**—(i) Barriers in mines are designed to secure safety of life and property, and 20 yards should be the minimum thickness. Sometimes Central Pumping Stations may be advantageous and barriers between Collieries might not be required.

(ii) The design and width of barriers depend on numerous factors, and each case must be considered on the details available.

(iii) Barriers should be supported by adequate pillars of coal in any seam beneath them.

(iv) Faults forming boundaries of mines should have barriers of unworked coal left. These should not be less than the size of barrier determined when no fault existed.

(v) In highly inclined seams the tendency of the barrier to slip is marked. In the same seams exceeding 3 feet in thickness, barriers on the line of strike are not reliable.

(C) **Dams.**—The following are the chief recommendations. The strata at the site of the dam should be cemented after being carefully cleaned. Dams should be properly protected against subsidence, and strongly built according to existing conditions. All details of position, pressure, and construction should be marked on abandoned plans. Their position and dimensions should be marked on all working plans in ink.

(D) **Approaching Old Workings.**—(i) The Coal Mines Act of

1911 is defective as regards boreholes. Single centre hole is not sufficient. " Sufficient flank holes " is indefinite. A limit of 40 yards is inadequate for security in all cases. Continuous boring is unnecessary when workings are in continuous contact with old workings. Apparatus for closing boreholes is not included. The Act does not apply to water over or under an approaching working. The examination of old workings is not always possible.

(ii) *Suggestions by Committee.*

(*a*) The manager to notify the Divisional Inspector of Mines and representatives of the workmen when any working is within 100 yards of any place likely to contain accumulated water. He should state the precautions taken and the reliability of his information as to position and extent of the old workings.

(*b*) A Committee of Reference should be formed of owner, workman, and representative of Mines Department, each holding a first-class or second-class certificate of competency as manager or under-manager. In case of disagreement the case to be referred to an independent mining engineer. This Committee of Reference applies only to disused workings which cannot be examined. Width of heading might be increased to 10 or 12 feet if committee agree.

(*c*) The boreholes should be arranged to make inadvertent contact with any place in the same seam impossible, when approach is at right angles to the general outline of the face of the old workings.

(*d*) Boreholes can be advanced up or down at any angle and successfully connected with old workings.

(*e*) In cross-measure drifts and sinking pits, two boreholes at least should penetrate into old workings before boring is discontinued.

(*f*) Boreholes crossing measures of soft strata and friable coal should be lined with tubes, and a concrete dam, or other cement structure, should be arranged at the face of the hole.

(*g*) Long boreholes should be followed up by the necessary flank holes when headings are being driven forward. Boring should be done through a fixed valve for safety.

(*h*) Old bores from the surface, as well as old shafts, should be properly filled in.

Working a Coal Seam liable to Spontaneous Heating

6. Make a sketch of a panel of workings suitable for a seam of coal 4 feet thick and inclined at 1 in 4, which is liable to spontaneous heating. The coal is to be cut by machine and brought out by conveyors along the face and the gate road. Show the connections with the main haulage road and return airway respectively. What provisions should be made for dealing with a fire on the coal face and for dealing with spontaneous heating in the waste or goaf? (25)

A. Fig. 109 shows a sketch of a panel of workings, on the longwall method, suitable for the conditions stated in the above question.

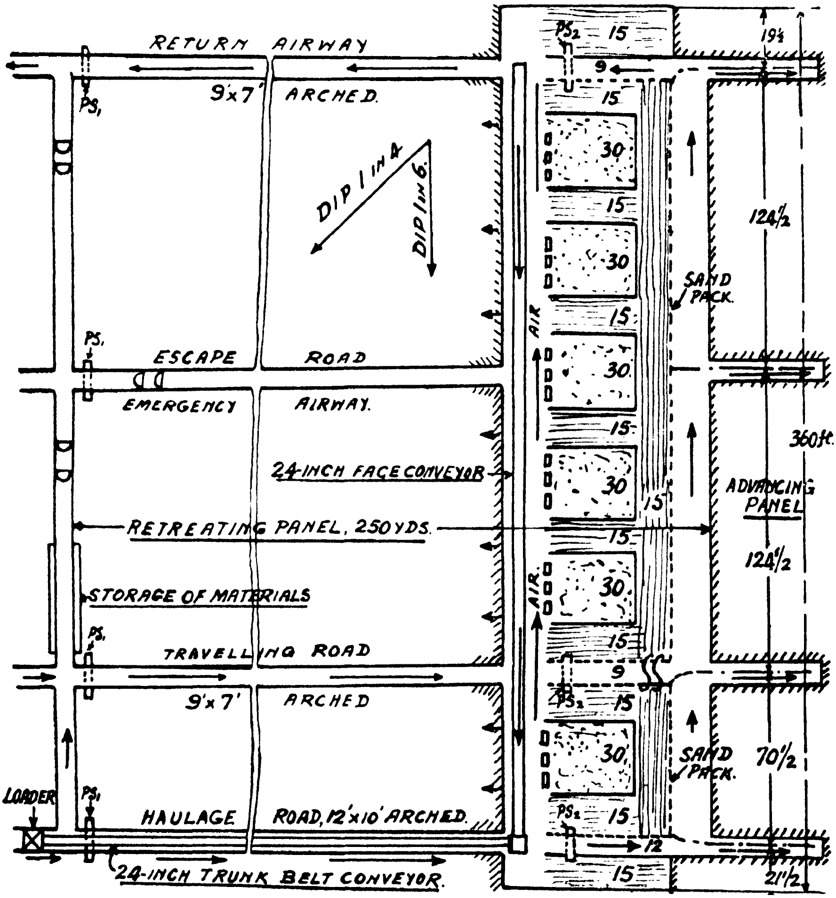

FIG. 109.—Working a Seam Liable to Spontaneous Heating.

The coal seam is worked on the retreating system with panels 250 yards wide on a face length of 120 yards. Useful dimensions and other important details are included in the sketch. There are four roads in the advancing panel of workings, which are to make provision for the next retreating panel. The retreating panel face is protected by 15-feet-wide packs having 30-feet-wide waste spacings. There are two intake airways and one return airway shown in the sketch, also an additional airway centrally placed to be used in case of emergency. The coals from the advancing panel can be conveyed to the main haulage by means of tubs.

The provisions necessary for dealing with heating on the coal face of the retreating panel should include the following :—

(i) A length of water-pipes from the surface should be installed along the main haulage road to the position of the loader.

(ii) Hose pipes with suitable jointing arrangements and jets should be at hand for reaching to the face.

(iii) A good supply of sand, bags, light girders and strong stone should be stocked at the point indicated on the plan for the storage of materials.

(iv) The prepared stoppings, indicated by PS_1 on the plan, are arranged in the main intake and emergency airways, also in the return airway, for short-circuiting or for cutting off the air-current, according to the position of the fire at the face of the retreating panel. The stoppings PS_2 are for dealing with heating at the faces of the advancing panel.

If the heating takes place in the waste or goaf of the retreating panel, the sand packs shown by the broken lines in the sketch should be given a liberal re-sanding near the position of the heating, thus stopping any circulation of air through the waste. The stoppings PS_2 might have to be closed if the re-sanding is not successful. See pages 206–7 for the details of the construction of sand packs in longwall workings.

JULY 1941 EXAMINATION

(*Five questions only to be answered.*)

Position of Main Ventilating Fan—Auxiliary Ventilation if Fan is Damaged

1. What provisions are usually made to prevent injury to the fan in the event of an explosion ? In the event of the fan being damaged by an explosion, what means would you adopt, as a tem-

porary expedient, to restore the ventilation so that exploration and repair work might go on whilst the fan was being repaired ? (30)

A. A ventilating fan, whether forcing or exhausting air, must not be placed directly over the mouth of a shaft, so that in the event of an explosion at the mine it would be less liable to injury. The fan is placed a short and safe distance away from the shaft and a connection, termed the *fan drift,* is constructed to pass the air from the shaft to the fan, or from the fan to the shaft, as circumstances require. The air-current leaves or enters the shaft at a convenient point below the banking level, and to prevent leakage the shaft mouth is covered by doors, or by an airtight building. Reinforced-glass windows are often fitted in the fan drift.

In the event of the fan being damaged by an explosion, the ventilation might be partly restored in a temporary way by having a steam jet erected in the upcast shaft as near the bottom of the shaft as possible. An air-current would be produced by the velocity of the escaping steam giving a velocity to the air in the shaft. A certain amount of heat from the steam would be given to the air in the shaft and its temperature would be increased. If compressed air is in use at the colliery, with pipes in the downcast and intake airways, jets of compressed air might be installed at pre-arranged points to give the air in these airways a velocity. Small compressed-air-driven ventilators might be installed in the intake airways to produce an air-current, such as the Rateau made aero-ventilator. Jets of water from water mains in shafts and mines have been made use of on a small scale for producing a current of air, the velocity of the water at escape being given to the air.

The return airways and upcast shaft might contain poisonous gases from the explosion, and the fixing of steam jets or an exhaust fan might be impossible. In these circumstances, auxiliary ventilation from the downcast side should be carried out.

Fires in Mines and How to Deal with Them

2. Enumerate the possible causes of fires in mines under two categories : (*a*) superficial, and (*b*) deep-seated. State how you would deal with the fires due to the causes you have named. (25)

A. **Superficial Fires.**—The causes of superficial fires in mines might be conveniently classed as follows :—

(*a*) *Fires at the Coal Face.*

(i) Coal-cutting machines igniting firedamp given off from the cut.

(ii) Fusing or failure of trailing cable under load.

(iii) Friction and fire caused by coal-conveyors rubbing against supports, owing to rollers not being sufficiently lubricated.

(iv) Deposits of oil and grease being allowed to accumulate and causing fire by naked lights coming into contact with them.

(v) Shot-firing at the face igniting firedamp and causing a fire.

(vi) Spontaneous heating of loose and exposed coal finally resulting in fire.

In dealing with fires under this heading, all motors should be stopped immediately, and the gate-end switch should be put in the "off" position. If the fire is in the vicinity of the coal-cutter, use should be made of the fire-extinguisher, and sand or stone-dust carried on the machine. More sand or stone-dust should be available at the roadheads. In the other cases of fire, the sand stored at roadheads should be used liberally and, if necessary and possible, the water-supply near the face should also come into use. All workers on the inbye side of the fire should be warned to make their escape immediately.

(*b*) *Fires at Points where Motors and Machinery are Working.*

(i) Motors breaking down and taking fire.

(ii) Careless handling of current-carrying cables, causing damage and shorts.

(iii) Dirty engine- and motor-rooms, oil and waste becoming ignited.

Fires at or near electrical equipment should be attacked with foam extinguishers, dry "Mynex" powder, and sand or stone-dust. In other cases of fire, extinguishers and sand or stone-dust should be used.

(*c*) *Fires on Haulage Roads.*

(i) Fires caused by the friction produced at rollers, wheels, drums, brake wheels and by ropes rubbing.

(ii) By electric cables failing under load and causing fire.

In the second item the current should be cut off immediately. Sand, and water, if necessary, should then be applied for dealing with the fires.

(*d*) *Fires in Naked-light Mines.*

(i) Careless throwing away of old wicks in a smouldering condition.

(ii) Leaving open lights unattended near inflammable material.

(iii) Workmen carrying matches and smoking in cabins, engine-rooms or stables.

(iv) Ignition of inflammable materials, oil, waste and grease at temporary storage-points.

When attacking fires caused by the above items, use should be made of fire extinguishers, sand or stone-dust, and water, according to position and material available.

(e) *Fires in Mines where Safety-lamps are used.*

(i) Defective lamps causing ignitions of firedamp and fires.

(ii) Wilful damage to lamps causing ignitions of firedamp and fires.

(iii) Accidents damaging lamps causing ignitions of firedamp and fires.

(iv) Hanging clothing too near burning lamps and causing fire.

The same method of attack should be applied as in the previous case.

The above methods of dealing with fires underground are only possible if the necessary fire-fighting appliances and materials are constantly available at points in the mine referred to on pages 233–4.

Deep-seated Fires.—The causes of deep-seated fires in mines are as follows :—

(*a*) By superficial fires, originating as stated above, getting beyond control.

(*b*) In the event of spontaneous combustion of coal or fires in waste workings reaching an advanced stage.

(*c*) After colliery explosions of firedamp and coal-dust, extensive fires are often caused by the great heat produced by such explosions.

When dealing with fires of this description it is necessary to withdraw any workmen on the inbye side of the fire, and to call upon the services of the Central Rescue Brigade as speedily as possible with all the equipment for fire-fighting that they have at hand. It might be practicable for the brigade to attack the fires outside the waste workings by using water under pressure from existing or specially-laid pipe-lines, or compressed-air mains already installed. The direction of the ventilating current might have to be reversed in dealing with such fires. Should the above methods prove impossible, it will be necessary to seal off the area by suitable stoppings in the intake and return airways. The stoppings should be built by the members of the Rescue Brigade who are specially trained in this work under all conditions likely to be met with. Pending the arrival of the Central Brigade, the colliery fire-fighting parties should be called out by the management, and their work should include the clearing of tracks for the conveyance of fire-fighting material, making a fresh-air base as near the seat of the fire as possible (say 50 feet), and obtaining as much information as possible regarding the position and extent of the fire. The management should arrange for the necessary supply of asbestos bags, 30 ins. × 14 ins., and the supply of, say, 500 tons of sand daily until the operations are completed.

Approaching Flooded Workings—Water appearing in Faulty Ground

3. A heading with boreholes, required by the Coal Mines Act, is being driven towards old workings which contain a dangerous accumulation of water. At a point about 15 yards from the old workings, faulty ground is met with and water appears, although there has not been any sign of water from the boreholes. How would you account for the appearance of the water at the face and what would you do? (25)

A. The water entering the heading in the above case is evidently coming from the faulty ground and possibly from the "leader" or "Vees" of the fault itself, and it is quite reasonable to assume that the old workings have actually reached the fault at a higher level than the heading and boreholes referred to. A number of exploring headings, down the actual leader or vees of the fault, to prove its size, might have been driven before the workings were abandoned. Details of such headings are rarely shown on old plans, and extreme caution is always necessary under such conditions. Taking the above details into consideration, it would be courting disaster to proceed any further with the heading and the bore-holes. The details given are almost identical with those existing at the time of the Redding Pit disaster in Stirlingshire many years ago, where flooding of the mine occurred and lives were lost.

Procedure.—The following procedure appears to be necessary in dealing with the situation:—

(i) The existing heading should be stopped and strongly supported, and a reliable type of dam or bulkhead should be built in it at a distance of 15 to 20 yards from its face, to be quickly closed in case of emergency, so as to safeguard the mine against possible flooding.

(ii) The exact position, or line of the fault, should be determined well in advance of the general line of the working face by driving exploring headings at about 50 yards apart, with a centre borehole in advance at least 50 yards long at all times.

(iii) A barrier of coal, adjacent to the fault, should be left unworked in the seam under consideration, and its thickness should not be less than 25 yards. Care should be taken by the Colliery Company to prevent this barrier of coal being worked off at any future time until the old workings are pumped dry.

Details of a Seam liable to Spontaneous Combustion—Method of Working the Seam

4. Give a detailed section of a seam of coal which you consider may be liable to spontaneous combustion. What precautionary measures should be taken in the opening out and working of the seam ? Make a sketch to illustrate your answer. (25)

A. The above question has been fully answered previously ; see pages 206-7 and Figs. 105 and 106, also Fig. 109.

Rules to be Observed in the event of an Underground Fire

5. What rules would you make, as Manager of a Colliery, to be put into operation in the event of a fire occurring underground ? (25)

A. The following details might be included in the rules referred to in the above question :—

(i) The fire should be attacked on the intake side by using the fire-fighting equipment near at hand, and at the same time all electric current in the district should be cut off. If the fire is not put out quickly, all men should be withdrawn from the return side of the fire. The deputy or fireman of the district must take charge of the operations as quickly as possible.

(ii) Communication with the Manager or Under-manager should be established as quickly as possible by telephone or messenger, with a view to allowing them to co-operate in the work of fire-fighting by arranging for the services of the Central Rescue Brigade, necessary materials, and aid from other districts.

(iii) The fire must be attacked continuously until help arrives, and the Manager, or Under-manager, should be kept informed of the progress of the operations.

(iv) The official in charge of the operations should continue to combat the fire as effectively as possible until withdrawal becomes necessary. When this stage is reached, a number of reliable workmen should be retained to make ready for the Rescue Brigade to operate quickly. As much information as possible regarding the extent of the fire and the purity of the air-current should be collected and noted for the Rescue Brigade, and a fresh-air base should be established, say 50 feet from the fire.

(v) The above rules are supplementary to those carried out at the surface by the Manager or Under-manager, respecting the Central Rescue Brigade, other help, necessary materials, and the clearing of roads for transport of materials to the fresh-air base.

Fire Training.—In order to carry out the above rules efficiently, the following method of training fire-fighters should be carried out :—

(*a*) Have all deputies, or firemen, and men in charge of haulage roads, inclines, shafts, and other important underground work, specially trained in fire-fighting at a Central Rescue Station. At least two workmen in each district of the mine, during each shift, should have received similar training.

(*b*) All men trained as above should have fire drill from time to time which should include a study of the details contained in the plan of the particular district in which they are for the time being employed.

Fire-fighting Equipment.—The question of fire-fighting materials in a mine is important, as fire-fighting is not successful if this part of the organisation has not received the necessary attention. The following materials should be kept in constant readiness for use in the event of a fire :—

(i) At the shaft bottom and shaft sidings, etc., there should be a good supply of water with hydrants, hoses, and nozzles, sand or stone-dust in bulk, and fire extinguishers of the foam and Mynex types.

(ii) A line of water-pipes, say 2 inches to 2½ inches diameter, should be laid down the shafts, and at least 1-inch diameter pipes to all districts of the mine. Standard connections might be installed every 400 yards to suit rescue station equipment. These water-pipes should be carried as far as the flats or lyes of districts, loading stations, or other convenient positions not more than 100 yards from the working face of the district.

(iii) The equipment at flats or loading stations should include eight pieces of canvas hose in 50-feet lengths with standard connections and nozzles, a portable hand-pump for face work, three to six portable tanks on wheels of about 100 gallons capacity, six old oil drums with handles and filled with water for immediate use, fire-fighting tools, and sand in bulk with the necessary buckets for conveying it to the face.

(iv) At all roadheads near the working face there should be stored sand or stone-dust, the amount for each face amounting to 5 tons. Two fire extinguishers with the necessary refills, say 2 gallons each, should be stored near the roadhead where electrical equipment is installed.

(v) On all haulage roads and inclines, in addition to water-pipes already mentioned, there should be at least one cwt. of sand or stone-dust stored at points, say 100 yards apart, with the necessary buckets for carrying it in case of fire.

(vi) Motor-rooms should be provided with foam fire extinguishers, dry Mynex powder, and at least 5 cwts. of stone-dust.

(vii) Cabins, stables, oil storage points, and greasers should be provided with fire extinguishers and a liberal supply of sand or stone-dust.

Organisation of Rescue and Recovery Work after an Explosion

6. An explosion occurs during a working shift in a section of a mine, causing considerable damage. What should be done immediately, and how would you organise and carry out the rescue of survivors and the examination of the workings ? (25)

A. This question has been fully answered previously; see pages 176 and 185.

PART IV.—MACHINERY

MAY 1931 EXAMINATION

ELECTRICAL SECTION

Details of Electrical Transformer and Tests

1. What particulars and data do you expect to find on the name-plate of an electrical transformer ? What tests would you apply to a transformer to ascertain that it was in good and correct working order ? (28)

A. The following particulars and data might be expected on the name-plate of an electrical transformer :—

Maker's name and type of transformer.
Voltage ratio of transformation.
kVA. capacity and periodicity in cycles per second.
Primary and secondary volts and secondary amperes.
Internal connection of transformer coils, such as delta-star or star-delta, etc.
Serial number.

To ascertain if the transformer is in good working order it is necessary to carry out insulation and continuity tests. The insulation test should be made with a megger testing set and the following readings taken :—

(i) Insulation resistance between primary winding and frame.
(ii) Insulation resistance between secondary winding and frame.
(iii) Insulation resistance between primary and secondary windings.

The continuity test of the windings should be made by using a galvanometer. The instrument is connected to the primary and

Note.—Under Examination conditions, candidates are required to answer *five* questions only from this Section : at least *one*, and not more than *two*, to be chosen from the Electrical Section, the remainder from the Mechanical Section. The figures in brackets indicate the maximum marks allotted to each question.

secondary terminals of the transformer successively, and if the resultant tests are satisfactory, the current can be switched on to the primary terminals with the secondary terminals on open circuit.

Construction of Induction Motor

2. Describe carefully the construction of an induction motor with a wound rotor for three-phase current (say for 100 H.P. at 960 r.p.m., 50 cycles). Deal especially with the electrical circuits of the motor, giving simple diagrams to show the connections of the coils amongst themselves and with exterior cables. (28)

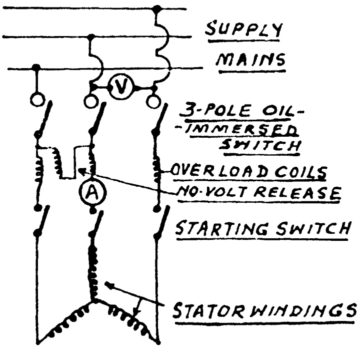

FIG. 110.—Mains and Stator Windings for Induction Motor.

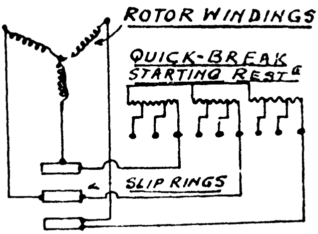

FIG. 111.—Secondary or Rotor Windings for Induction Motor.

A. An induction motor with a wound rotor for three-phase current has two separate copper circuits which are known as the *primary* or Stator and the *secondary* or Rotor.

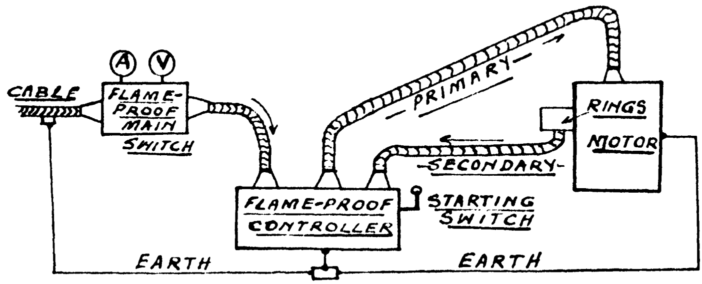

FIG. 112.—General Arrangement of Induction Motor and Auxiliary Plant.

The conductors of the primary circuit are disposed in slots in the inner periphery of a hollow laminated iron cylinder forming the stationary part of the motor, termed the Stator. Fig. 110 shows the connection between the supply mains or exterior cables and the Stator windings.

The conductors of the secondary circuit are disposed in slots on the outer periphery of a laminated iron cylinder fixed on a shaft so as to be capable of rotation. Three slip-rings, which are insulated from each other, and also from the shaft, are connected to the secondary conductors at equidistant points. This part of the motor is termed the Rotor. Fig. 111 shows the connection between the rotor windings and the slip-rings on the rotor shaft.

Fig. 112 shows the general arrangement of the motor and plant. See also pages 301–2.

Electrical Apparatus in a Haulage-gear House

3. Describe the apparatus that you would expect to find in an electrical haulage-gear house for working, controlling and protecting the motor. What are the essentials of flameproof switchgear ? (28)

A. Assuming an endless-rope haulage gear which is driven by a three-phase squirrel-cage motor, the electrical apparatus used in connection with this motor should comprise the following :—

A main air-break flameproof three-pole automatic circuit-breaker fitted with no-voltage release coil and three overload trip coils with time-limit device. In this way the motor would be protected against heavy overloads, and the supply current would be entirely cut off from the motor and starter in the event of overload. A motor starting switch of either the star-delta or the auto-transformer type would be required to limit the starting current of the motor.

Flameproof Switchgear.—The essentials of a flameproof switchgear are as follows :—

(i) All apparatus must be of robust construction to withstand rough usage, while at the same time it should withstand, without injury, the force of an explosion which might occur within it.

(ii) It should be provided with a pressure-relief device, so that flame and hot gases, produced by an internal explosion, may be cooled by contact with metal surfaces and by expansion, the apertures or gaps being so dimensioned that a certain amount of release of pressure is obtained, while at the same time the flame from the internal explosion is prevented from passing to the external atmosphere. The maximum size of aperture, or width of gap, allowed under the present requirements of the Coal Mines Act is 0·02 inch.

(iii) If bolts, studs or screws are provided for the attachment of any component parts of a flameproof casing, the holes for the same should not pass through the walls of the flameproof casing.

(iv) The direct entry of cables into flameproof casings is not permissible by the Coal Mines Act. The acceptable construction is that which comprises the use of insulated terminal stems projecting through the flameproof construction, so that the external cables terminate outside the enclosure.

MECHANICAL SECTION *

Pumping Plant for a Sinking Shaft

4. Describe a pumping plant for a sinking shaft to include a centrifugal (or turbine) pump, coupled direct to an electric motor, and the accessories of the pump, which is to be arranged to follow the sinking as it proceeds. (28)

A. A pumping plant of the above description might be fixed in a frame made up of girders, and containing suitable platforms for access to the working parts, with a ladder connecting the various platforms. The suction branch is attached to the pump through the medium of a ball-and-socket joint, so that the suction pipe can be placed at any desired spot in the bottom of the sinking pit. The motor is usually water-cooled, and the entire plant is suspended in the shaft by a rope attached to a crab engine, so that the lowering of the plant is easily accomplished in stages of 9 to 18 inches as the sinking progresses. The delivery pipes are glanded to the crab-engine rope by suitable glands 48 feet apart, the rope having a free movement through the eyes at the ends of the glands. The connection between the crown delivery pipe and the water-receiving tank on the surface is accomplished by means of a length of flexible hose and a wooden tower. Pipes of 16-feet length are easily fixed to the top of the column as required.

The power cable is clamped to the delivery pipes by means of wood blocks and straps. The suction branch is provided with a circular strainer and foot valve at the base, and also a regulating valve near the pump inlet. The impeller casings are provided with priming cocks for priming the pump from the delivery column. The delivery column is provided with a reflux valve and with a master valve to regulate the discharge of water from the pump. The motor is provided with a main switch fitted with no-voltage release and overload coils, and a suitable starting switch.

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* See note on page 235.

Bad Boiler Feed-waters

5. Explain carefully what causes certain waters to be bad for feeding boilers. When a bad water has been treated to make it suitable for use in boilers, does it behave within the boiler in the same way as a good untreated water ? Explain your answer. (28)

A. Bad water for feeding boilers might contain in solution carbon dioxide, oil, and soluble carbonates, chlorides and sulphates, and it is an advantage to remove these before the water enters the boiler. Carbon dioxide in feed-water causes pitting of the boiler shell at the water-line, and it can be removed by heating the water before it enters the boiler. Oil in feed-water breaks up into acids and glycerine, and the acids attack the boiler shell. It can be largely removed by a coke filter, or by treating the water with caustic potash (KOH) before it enters the boiler. Soluble carbonates, chlorides and sulphates of calcium and magnesium in feed-water are objectionable, as they form incrustations inside the boiler and thus prevent absorption of heat by the water. These compounds might be removed in the form of a precipitate by adding suitable chemicals to the water, such as lime to precipitate the bicarbonates, which would otherwise deposit at the first point of boiling of the water. Soda is added to the feed-water to deal with the permanent hardness caused by sulphates and chlorides ; it acts by causing the precipitation of some salts and the changing of others to a form in which they remain in solution in the boiler under normal conditions. Phosphate of soda might be added to boiler feed-water to correct an excess of soda and to prevent the formation of caustic soda inside the boiler. Boiler feed-water treated in the above manner, so as to soften it, might contain soluble compounds, and if care is not taken during the process of steam generation the water in the boiler might become concentrated with these compounds. The boiling temperature of the water (solution) will then be raised, while the temperature of the steam remains constant, and priming will result. Blowing-down will be necessary daily, or at more frequent intervals, so as to reduce the water-level by an inch or more. Part of the water containing concentrated solution will be removed in this way, and fresh water can be added to take its place. By these measures, solution concentration can be kept at a reasonable figure, and priming during the process of steam generation in the boiler is obviated.

Calculation of Dimensions of Haulage Engine

6. An output of 500 tons of coal is to be delivered up an incline 700 yards long, dipping 1 in 8, laid with two tracks, in tubs holding 20 cwt., in 7 hours, by direct two-rope balanced haulage, the power being steam at about 80 lb. per sq. inch. Describe the hauling engine that you would install, giving the chief drum and cylinder dimensions. (28)

A. Let the speed of the haulage rope be 5 miles per hour or 440 ft. per min.

$$\text{Time taken to haul up the load} = \frac{700 \times 3}{440} = 5 \text{ minutes.}$$

$$\text{Time taken to haul and change} = 10 \text{ minutes.}$$

$$\text{Number of trips made per hour} = \frac{60}{10} = 6$$

$$\text{Tons to be hauled per trip} = \frac{\text{tons per hour}}{\text{trips per hour}} = \frac{500}{6 \times 7} = 12 \text{ tons.}$$

Trams required per trip = 12, each holding 20 cwt. of coal and having 7 cwt. tare.

Pull on the full side rope :—

$$\text{Gravity of the load} = \frac{12 \times 27}{8} = 40{\cdot}5 \text{ cwt.}$$

$$\text{Friction of the load} = \frac{12 \times 27}{60} = 5{\cdot}4 \text{ ,,}$$

$$\text{Gravity of rope (6 lb. per fm.)} = \frac{350 \times 6}{112 \times 8} = 2{\cdot}4 \text{ ,,}$$

$$\text{Friction of rope} = \frac{350 \times 6}{112 \times 20} = 1{\cdot}0 \text{ ,,}$$

$$\text{Total} = \underline{49{\cdot}3} \text{ cwt. Say 50 cwt.}$$

Size of hauling rope :—

$$\text{Safe working load on rope} = 2\tfrac{1}{2} \text{ tons.}$$

$$\text{i.e.} \quad \frac{C^2 \times 3}{8} = 2\tfrac{1}{2} \text{ ,,}$$

$\therefore C^2 = 6{\cdot}7$ and $C = 2{\cdot}6''$ circumference or $\frac{7}{8}''$ diameter.

Pull on the engine :—

$$\text{Gravity of the coal} = \frac{12 \times 20}{8} = 30{\cdot}0 \text{ cwt.}$$

$$\text{Friction of both loads} = \frac{12 \times (27 + 7)}{60} = 6{\cdot}8 \text{ cwt.}$$

$$\text{Gravity of full rope} = \frac{350 \times 6}{112 \times 8} = 2{\cdot}4 \text{ ,,}$$

$$\text{Friction of full rope} = \frac{350 \times 6}{112 \times 20} = 1{\cdot}0 \text{ ,,}$$

Total $= \underline{\underline{40{\cdot}2}}$ cwt. Say 40 cwt.

Diameter of drums $= \frac{7}{8}''$ rope $\times$ 80 = 70 inches. Say 6 feet.

$$\text{Revolutions of drum} = \frac{\text{Speed}}{\text{Circ. of drum}} = \frac{440}{6 \times \frac{22}{7}} = 23 \text{ r.p.m.}$$

Steam-Engine Details : The engine to have two cylinders with a gear ratio of 3 to 1, and a stroke equal to twice the diameter of the cylinder.

Effective steam pressure in the cylinders $= 80 \times 0{\cdot}6$, or, say, 50 lb. per sq. in.

Let P = Effective steam pressure = 50

L = Length of stroke in feet $= \dfrac{2d}{12}$

A = Sectional area of the two cylinders $= 2(d^2 \times \frac{11}{14})$

N = Number of strokes = 23 (revs. of drum) $\times$ 3 (gear) $\times$ 2 strokes per rev.

Then work done by the engine $\times$ modulus = work done on the incline.

$P.L.A.N. \times$ modulus = load in lb. $\times$ speed per min.

$$50 \times \frac{2d}{12} \times 2\left(d^2 \times \frac{11}{14}\right) \times 23 \times 3 \times 2 \times \frac{6}{10} = 40 \times 112 \times 440 \text{ ft.-lb. per min.}$$

$\therefore d^3 = 1818$

$\therefore d = 12$ inches, and stroke = 24 inches.

The engine is to be geared to the drums by spur and pinion wheels, the pinion wheel on the crankshaft to be 2 feet in diameter and the spur wheel on the drum shaft 6 feet in diameter. The two drums required for working the balanced haulage to be fixed on the drum shaft. Clutches will not be required. An emergency strap brake should be provided on one of the drums.

Water-Tube Boiler of the Stirling Type

7. Describe a boiler of the Stirling type, i.e. with bent, highly-inclined tubes, illustrating your answer with a simple sketch showing

a side elevation of the boiler. What is the usual diameter for the water-tubes ? (28)

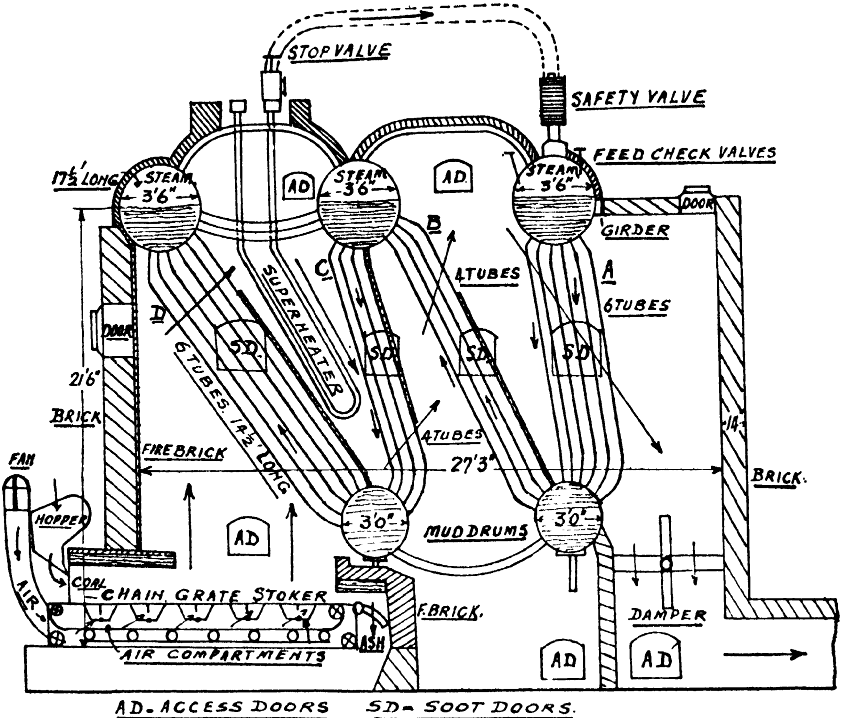


FIG. 113.—Sectional Side Elevation of Stirling Water-Tube Boiler.

A. Fig. 113 shows a side elevation of a Stirling boiler. Such a boiler is of the water-tube pattern with five drums, three of which are steam drums and the two remaining ones mud drums. These drums are interconnected by four banks of water-tubes. Both the steam and the water spaces of the upper drums are interconnected by a number of short curved tubes. The steam drums are 3 feet 6 inches in diameter and the mud drums 3 feet. They are supported by a steel framework strengthened by brickwork.

The feed-water is admitted to the rear steam drum and it descends the rear bank of tubes *A*, passing up the next bank *B*, down the bank *C*, and finally up the front bank *D*. The gases are directed between the banks of tubes by means of firebrick baffles. The flow of the gases is in counter-direction to the flow of water, as shown by the arrows, the coldest water meeting with the coolest gases.

The steam is drawn from the front and middle drums and passes through a superheater before entering the main steam column.

The tubes are made of solid-drawn steel tested to 1,000 lb. per sq. inch, the standard size being $3\frac{1}{4}$ inches external diameter. All the tubes are bent so as to enter the drums radially to give increased strength.

NOVEMBER 1931 EXAMINATION

Electrical Section

Switchgear and Protective Apparatus for Alternator

1. A three-phase alternator delivers power to an overhead bare copper transmission line at a pressure of 3,300 volts without transformation. Give a list of the apparatus and instruments that might be installed between the alternator and the line for controlling and protecting the alternator. (28)

A. *Apparatus required for generator panel* :—

Three-pole bus-bar isolating switch interlocked with main oil switch.

Three-pole oil-break main circuit-breaker with free handle, fitted with three trip coils for balanced current protection, and one A.C. shunt trip coil with auxiliary contacts.

Three overload and three time-limit fuses.

Six current transformers for balanced current protection.

Three moving-iron ammeters with current transformers.

One indicating wattmeter.

One moving-iron voltmeter with potential transformer.

One oil-break neutral-earthing switch.

Apparatus for exciter panel :—

One moving-coil main field ammeter.

One moving-coil voltmeter.

One automatic field-breaking switch.

One field exciter rheostat.

One Sinell regulator for automatic voltage regulation.

Calculation of Loss in an Electric Transmission Line

2. A three-phase transmission line has bare copper conductors 0·04 square inch sectional area and is 1 mile long. The resistance of copper wire 0·04 square inch is 0·6 ohm per 1,000 yards. Electric power is fed to the line at the rate of 1,000 kW. at 3,300 volts,

and the power factor is 0·8. What kilowatts are delivered by the transmission line ? (28)

A.

Resistance (R) of each transmission line

$$= 0{\cdot}6 \times \frac{1760}{1000} = 1{\cdot}056 \text{ ohms} \quad . \quad . \quad . \quad (1)$$

$$\text{Current } (I) = \frac{\text{kW.} \times 1000}{\text{Volts} \times \text{power factor} \times \sqrt{3}} = \frac{1000 \times 1000}{3300 \times 0{\cdot}8 \times 1{\cdot}732}$$

$$= 218{\cdot}7 \text{ amps.} \quad . \quad . \quad (2)$$

$$\text{Watts lost in Resistance} = I^2 \times R \times 3 \text{ lines}$$

$$= (218{\cdot}7)^2 \times 1{\cdot}056 \times 3$$

$$= 151{,}924 \text{ watts or } 152 \text{ kW.} \quad . \quad . \quad (3)$$

$$\therefore \text{Kilowatts delivered by line} = 1000 - 152$$

$$= \underline{848 \text{ kW.}} \quad . \quad . \quad . \quad . \quad (4)$$

Direct-Current Motors

3. Describe briefly, using simple diagrams, the three chief kinds of direct-current motors, and state a purpose for which each type is peculiarly suitable. How is the direction of turning of each type reversed ? (28)

A. Direct-current motors are usually divided into three types, known as " Series," " Shunt " and " Compound," respectively.

A " Series " motor is one in which the whole of the load current is passed through the field coils and armature, and the strength of the field depends upon the current flowing in the armature. Such a motor takes a large starting current, which gives to it a large starting torque. The speed of a " Series " motor acts automatically with the load, an increase in the load causing a drop in the speed of the motor, while a decrease in load causes an increase in speed. The " Series " motor might be used where a heavy starting torque is required and where a variation in the speed with the load is desirable, such as main rope or dook haulage.

A " Shunt " motor is one in which only part of the main current passes through the field coils. It has a field consisting of many turns of fine wire connected in shunt across the armature terminals. Owing to the high resistance of the shunt windings, only a small portion of the main current flows in the shunt. At no-load the speed is constant, but for a large range of load currents the speed is only approximately constant. The latter is fully explained in the following way: The flux is fairly constant, and the speed is proportional to the back E.M.F.; if the load is increased the difference between the applied E.M.F. and the back E.M.F. increases, and the speed curve falls slightly with increasing load. Variation in speed is readily obtained by varying the field current.

"Shunt" motors are usually applied where only a moderate starting torque is required, such as for endless haulage, turbine pumps and auxiliary ventilating fans.

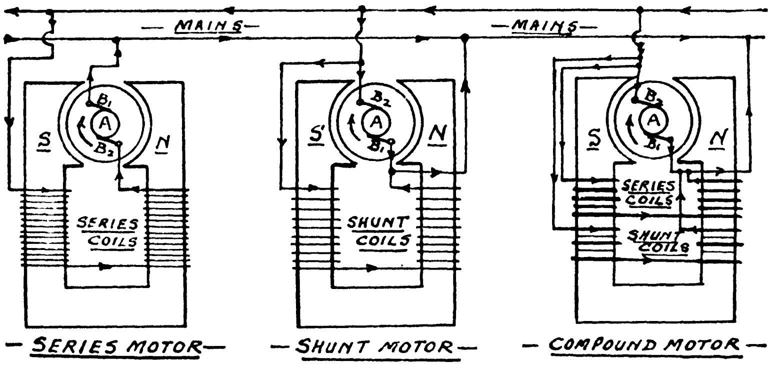


Fig. 114.—Direct-Current Motors.

A "Compound" motor has two windings, one being in shunt across the brushes and taking only a small current, the other in series with the armature and taking the full current of the motor. These two windings are so arranged that they assist each other, and as the load is increased a stronger magnetisation and an increased turning effort is obtained. This motor is generally applied where good starting torque is essential, such as for coal-cutting machines, three-throw pumps, and main-and-tail haulage.

The direction of turning is reversed in a series motor by changing over the field leads, in a shunt motor by changing over the shunt leads or the armature leads, and in a compound motor by changing over both the shunt and series leads.

Fig. 114 shows simple outline sketches of the connections for the three types of motor described above.

Mechanical Section

Identification of Special Materials

4. State what is meant by the following descriptions of materials,

and name one article for which each material is specially suitable: (*a*) Manganese steel; (*b*) Chilled cast iron; (*c*) Nickel steel; (*d*) Case-hardened steel. (28)

A. *Manganese* is put into steel to prevent formation of blow-holes during casting operations, and also to act as a corrective for small percentages of phosphorus and sulphur. Its tendency is to give increased tensile strength, but reduced ductility. Manganese up to 2 per cent. does not alter the steel, while 5 to 14 per cent. increases its toughness and tenacity to a maximum, above or below these figures producing a weak, brittle, and intensely hard alloy. Castings of manganese steel are easily forged hot and are used for car wheels, bevel wheels, axles, tyres, crossings, and headers for water-tube boilers, where violent shock has to be resisted.

Chilled Cast Iron is made by cooling-off the castings quickly by the use of an air-blast. Sometimes they are re-heated and again air-cooled. Such castings are very hard on the outer surface and resist wear in the same way as case-hardened steel. They are used in cases where a surface is required to resist wear, such as rolls in steel-works and the treads of pulleys used for endless-rope haulage. In the construction of centrifugal pumps, chilled cast iron is used for the blocks which contain the bearing metal for the pump bearings.

Nickel Steel is a steel containing a percentage of nickel, a common percentage being 12 to 14. It is difficult to weld, however, above 3 per cent. A good working percentage is 2 to 4. Steel containing 36 per cent. of nickel is termed "Invar," as it does not expand with heat; it is used in making measuring-tapes. Nickel steels are rustless, strong, and have a low coefficient of expansion. Their power to resist shock, and their toughness, makes them useful for armour plates, propeller shafts, and machine parts.

Case-hardened Steel.—When steel is heated up in a special stove so as to increase the percentage of carbon in its outer surface, giving a hard wearing surface, and at the same time producing a ductile and malleable steel, it is said to be "case-hardened." In the Harvey process, charcoal is used for this purpose; in the Krupp process, gases rich in hydrocarbons are used, while in the Beardmore process high-carbon steel combined with soft steel is used, to get a range of transition from soft to hard steel. Such a steel resists wear and is used for machine spindles which are liable to wear quickly when rotating within fixed bushes.

Description of a Lancashire Boiler

5. Describe a Lancashire boiler. Name approximately the largest Lancashire boilers made (length and diameter). State the highest steam-pressure for which Lancashire boilers are suitable. (28)

A. Fig. 115 shows a sectional elevation of a Lancashire boiler, suitable dimensions being marked on the drawing. The boiler

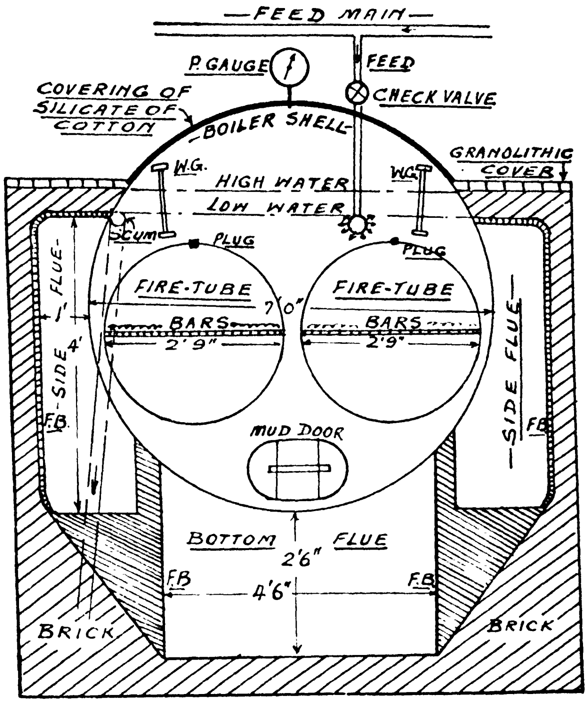


FIG. 115.—Front Sectional Elevation of Lancashire Boiler.

shell is 7 feet in diameter and 30 feet long, and it contains two fire-tubes which are $2\frac{3}{4}$ feet in diameter. The heated gases from the fires pass through the fire-tubes to the back of the boiler, traverse the bottom flue, and pass to the chimney by way of the side flues. They give up their heat to the water inside the boiler, and thus generate steam. The flues are lined with firebrick, as shown in the sketch, to withstand the high temperature of the furnace gases. Such a boiler has a large steam and water capacity and is suitable for colliery engines which do not run constantly. They are considered best for use where the feed-water is hard and dirty, owing to their simplicity in construction, strength, easy examination, and easy cleaning. Heat is absorbed slowly in this boiler, while there is a lack of proper circulation of the water and steam generation is slow.

The largest Lancashire boiler is 9½ feet in diameter and 32 feet in length. The highest steam pressures for which Lancashire boilers are suitable range between 200 and 250 lb. per sq. inch.

Drawing a Butt Joint for a Boiler Plate

6. The longitudinal joints of the plates in a Lancashire boiler are of the butt type with the following specification: Plate thickness, 1 inch; rivet holes, 1 inch diameter; cover plates, 9¼ inches wide; rows of rivets, 4; spacing of rivets longitudinally, 3 inches; spacing of rows of rivets transversely from one side of cover plate to the other, 1¾ inches, 1½ inches, joint, 1½ inches, 1¾ inches.

Make a plan of this joint to a scale of one-quarter full size, showing a length of about 18 inches of the joint. The rivet holes can be indicated by crossed centre-lines at the correct points except for one hole in each row, which should be shown correctly to scale. Show all necessary dimensions. The curvature of the shell plate can be ignored. (38)

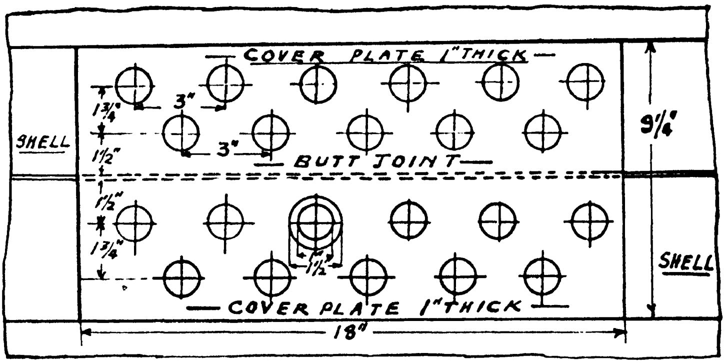


FIG. 116.—Plan of Butt Joint and Cover Plate for Lancashire Boiler.

A. Fig. 116 shows the drawing asked for in the above question. All the rivet holes for the joint are shown to the required scale.

Steel Wire Winding-Ropes

7. Write a short essay on steel wire winding-ropes, dealing with their construction, installation, and treatment in use. Note especially points connected with safety. (28)

A. Wire ropes used in connection with shaft-winding plants should have good flexibility for safe bending over drums and pulleys, high tensile strength for safety against excessive live loads, sufficient hardness to resist abrasion, and great durability to resist heat and corrosion. The materials of construction for winding-ropes might be as tabulated below :—

	Tons per sq. inch ult. strength.
Improved patent steel	80 to 90.
Mild plough steel	95 to 100.
Best plough steel	100 to 110.
Special improved plough steel ...	110 to 120.
Special high-tension steel	120 to 140.

Wire ropes for use in winding-shafts are chiefly manufactured round in shape and might be classified as ordinary lay, Lang's lay, flattened-strand, galvanised, and locked-coil. In the ordinary lay rope the strands of the rope receive a right-hand twist, while the wires of the strand get a left-hand twist, thus exposing a small surface of each wire to wear. Lang's lay ropes have a uniform twist, as the wires of the strands are twisted in the same direction as the strands in the rope, thus obtaining a larger wearing surface for each of the wires. Flattened-strand wire ropes have strands with one or more flattened surfaces instead of the circular form, thus giving great flexibility and a very smooth wearing surface. Galvanised wire ropes are chiefly used where shafts are wet and the water has corrosive properties. The hempen core is usually replaced by a galvanised iron core, and all the wires are galvanised to resist corrosion. Locked-coil wire ropes are made with specially shaped interlocking wires, and they closely resemble a round bar with a regular and uniform surface. They are non-rotating and are much used in sinking pits.

When wire ropes are installed in a shaft, care should be taken to remove twists before use. They should be effectively capped and arrangements made to allow for the stretch of the rope when loaded. Hard rubbing surfaces should be avoided while they are running, and they should receive a regular treatment with special oil to preserve them and to prevent internal corrosion. Re-capping of the rope should be carried out every 3 to 6 months, and the cut end of the rope should be thoroughly examined for corrosion. Bad parts should be removed before re-capping is carried out. The ropes should not be worked round drums and pulleys of too small diameter. Shock at the start of a wind should be reduced to the lowest possible figure, and care should be exercised to obtain gradual retardation at the end of the wind, without excessive braking. The factor of safety for winding-ropes should not be below 10, and their life should be limited to $3\frac{1}{2}$ years.

Calculation of Duty of Centrifugal or Turbine Pump

8. A centrifugal, or turbine, pump has 5 impellers in series, 13 inches in diameter, running at 1,440 r.p.m. The delivery branch is 7 inches bore and the suction branch is 8 inches bore. For what rate of delivery of water and head would this pump be suitable ? (28)

A. Assuming a discharge velocity of $5\frac{1}{2}$ feet per second, the volume of water can be calculated from the size of the delivery branch as under :—

$$\text{Area of pump outlet} = \frac{\text{Gallons per min.} \times 144}{5{\cdot}5 \times 60 \times 6\frac{1}{4}}$$

$$7'' \times 7'' \times \tfrac{11}{14} = \frac{\text{Gallons per min.} \times 144}{5{\cdot}5 \times 60 \times 6\frac{1}{4}}$$

$$\therefore \text{Gallons per min.} = \frac{7 \times 7 \times \frac{11}{14} \times 5{\cdot}5 \times 60 \times 6\frac{1}{4}}{144}$$

$$= \underline{550 \text{ gallons}} \quad . \quad . \quad . \quad . \quad (1)$$

The head to which the pump will discharge water can be calculated from the size of the impellers, the revolutions per minute, and the number of stages, as under :—

$$\text{Revs. per min.} = \frac{\text{Velocity per sec.} \times 60}{\text{Circumference of impeller}} \;\ldots \text{ for single impeller.}$$

$$\text{Revs. per min.} \times \sqrt{n} = \frac{\text{Velocity per sec.} \times 60}{\text{Circumference of impeller}} \;\ldots \text{ for } n \text{ impellers.}$$

$$1440 \times \sqrt{n} = \frac{\sqrt{2gh} \times 60}{\text{Diam. of impeller} \times \frac{22}{7}}$$

$$1440 \times \sqrt{5} = \frac{8 \times \sqrt{h} \times 60}{\frac{13}{12} \times \frac{22}{7}} \quad \text{(where } h \text{ is the head of water).}$$

$$\therefore \sqrt{h} = \sqrt{5} \times 10{\cdot}2 = 22{\cdot}8$$

$$\text{and } h = \underline{520 \text{ feet}} \quad . \quad . \quad . \quad . \quad . \quad (2)$$

MAY 1932 EXAMINATION

Electrical Section

Description of Three-Phase Squirrel-Cage Motor

1. Describe a three-phase motor with squirrel-cage or short-circuited rotor (say for 50 H.P. at 440 volts, 50 cycles). State the class of plant that may be driven by such a motor, and the methods of starting-up. What are the devices that can be adopted—both internal to and external to the motor—to increase the torque at starting whilst keeping down the starting current ? (28)

A. A squirrel-cage induction motor has a ring-shaped stationary iron shell known as the *stator*. The stator carries the magnetising windings, which are supplied with current from the source of power, and thus a rotating magnetic field is produced within it. A *rotor* is mounted on bearings and is free to rotate inside the stator. It carries a number of short-circuited copper bars in which currents are induced by the rotating field of the stator. The action between the induced currents and the rotating magnetic field gives rise to rotation of the rotor.

Squirrel-cage motors might be used for driving endless-rope haulages, auxiliary fans, coal-cutting machines, coal-conveyors, and turbine pumps.

Three methods might be employed for starting-up squirrel-cage motors, known as the Direct, Star-delta, and Auto-transformer methods respectively. The starting torque can be increased and the starting current reduced by using a high-torque rotor (internal). This type of rotor is provided with a double set of short-circuited copper bars, one set being of higher resistance than the other set. The starting torque is increased (externally) by the use of star-delta and auto-transformer starters when ordinary squirrel-cage motors are used. The following table shows the difference in torque and starting current with the two types of rotor :—

Method of Starting.	Motor.	Line Volts.	Line Current.	Full-load Torque.
Direct . .	Plain squirrel-cage	100 per cent.	600 per cent.	100 per cent.
Direct . .	High-torque	100 ,,	300 ,,	200 ,,

Power Factor

2. In connection with electric power, what is meant by the term "power factor"? What may be the causes of a low power factor? If a three-core cable were carrying its maximum current of 300 amperes at 0·6 power factor and 3,300 volts between phases, what extra power would be delivered if the power factor were raised to 0·8? (28)

A. In an alternating-current circuit, when the current is in phase with the impressed voltage the power of such circuit is represented by the product of current and voltage; but if the current is leading or lagging behind the impressed voltage, the product of current and voltage is called the "apparent load," and the true power of the circuit is then the product of current, voltage, and a factor. This factor, which is always less than unity and depends for its value on the angle of lead or lag, is termed the "power factor." It may be expressed as follows:—

$$\text{Power factor} = \frac{\text{True power}}{\text{Apparent power}} = \frac{\text{kW.}}{\text{kVA.}} = \frac{\text{Kilowatts}}{\text{Kilovolt-amperes}}$$

The power factor of a system is chiefly determined by the nature of the load; with normal lighting circuits it closely approaches unity, because the circuit possesses very little inductance. If an alternating-current circuit is highly inductive, the power factor is usually very low. The induction motor is usually the cause of low power factor, especially when worked on loads which are only a fraction of the normal output.

$$\text{Power in kW.} = \frac{\text{Volts} \times \text{amps.} \times \text{P.F.} \times \sqrt{3}}{1000} = \frac{3300 \times 300 \times 0{\cdot}6 \times 1{\cdot}732}{1000}$$

$$= 1029 \text{ kW.} \qquad (1)$$

If the power factor is raised to 0·8 then

$$\text{kW.} = \frac{1029 \times 0{\cdot}8}{0{\cdot}6} = 1372 \text{ kW.} \qquad (2)$$

$$\therefore \text{Extra power} = 1372 - 1029 = \underline{343 \text{ kW.}} \qquad (3)$$

Transformer and Switchgear Plant

3. A three-core cable, paper-insulated, lead-sheathed and double wire-armoured, carries three-phase current at 3,300 volts, 50 cycles, to the bottom of a shaft. Describe switchgear and apparatus

suitable for dividing the power into 5 circuits, 3 each of 100, and 2 each of 50 kW. capacity, the power being delivered to branch cables at 440 volts between phases. (28)

A. As the shaft cable is paper-insulated, it is essential that its cores be divided in a suitable trifurcating box and properly sealed. Fig. 117 shows in diagrammatic fashion the arrangement of the

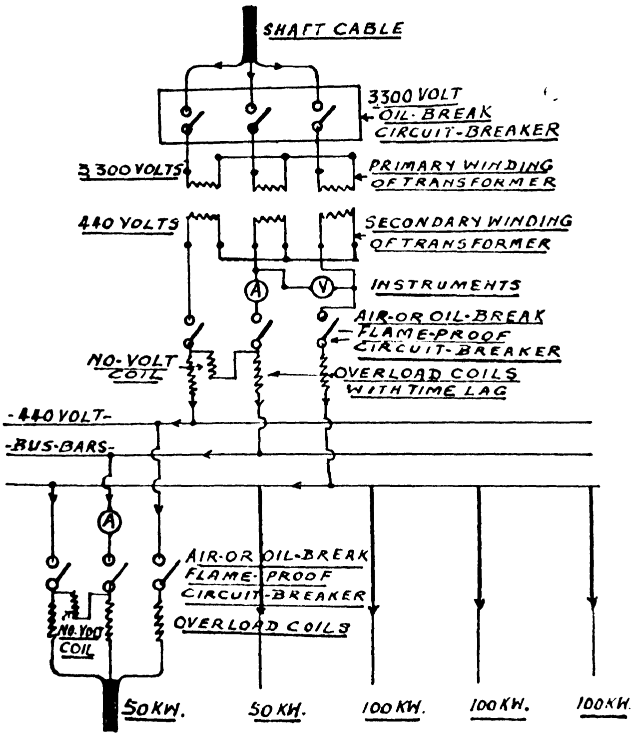


FIG. 117.—Switchgear and Apparatus for Transforming Voltage.

switchgear and apparatus suitable for dividing the power into the required circuits. The main supply, or shaft cable, is connected to a three-pole oil-break circuit-breaker of the flameproof type, from which it passes to the primary terminals of an oil-filled three-phase static transformer at a pressure of 3,300 volts. By mutual induction the supply pressure is reduced to 440 volts at the secondary terminals. The 440-volt pressure at the secondary is then connected to either an oil-break or an air-break three-pole flameproof circuit-breaker, capable of controlling and carrying the full current of the five feeder circuits. A main ammeter and a voltmeter should be connected with the main low-tension circuit.

Each feeder circuit should be controlled by a suitable three-phase flameproof oil or air-break circuit-breaker, with an ammeter fixed on each feeder. As these circuit-breakers are usually only suitable for overload protection, it will be necessary to have incorporated in them a leakage protection device ; for this purpose core-balance transformers might be used.

Mechanical Section

Description of Water-Tube Boiler

4. Describe one type of water-tube boiler, say for a duty of 10,000 lb. of steam per hour at 175 lb. per sq. inch. State in detail how the tubes are secured in the headers or drums so as to obtain a steam-tight joint. (28)

A. Fig. 118 shows in diagrammatic fashion the arrangement of a Babcock & Wilcox water-tube boiler for a duty of 10,000 to 12,000 lb. of steam per hour at a pressure of 175 lb. per sq. inch. Suitable dimensions are marked on the diagram.

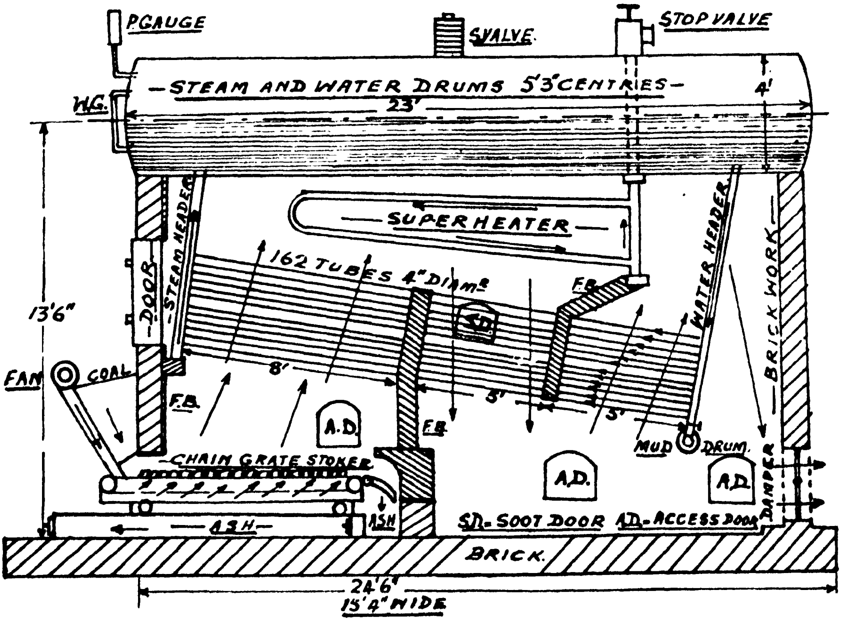


Fig. 118.—Sectional Side Elevation of Babcock & Wilcox Water-Tube Boiler.

This boiler consists of two steam and water drums supported on brickwork and connected by headers with 162 tubes and a superheater. It is fired at the front end by hand or by mechanical means, and is adapted for burning coal, oil or powdered fuel. The heated products of combustion pass over the outside of the tubes three times to heat the water which is circulated inside them, the greatest heat being applied to complete the final stage of the steam generation and the superheating. The water from the overhead drums passes to the lower end of the inclined tubes and has been converted into steam on leaving the tubes at the higher end. In this way there is a continuous circulation of water through the tubes, and dry steam is produced. The heating surface is thin and effective, and a thoroughly good absorption of heat is obtained from the fires. The draught area is large in this type of boiler and more complete combustion is obtained than with fire-tube boilers. Such a boiler has a heating surface of 3,654 square feet, a grate surface of 65 square feet, and a water capacity of 330 cubic feet. The balanced-draught chain-grate stoker is so arranged that the amount of coal and air can be increased or diminished at will, according to the demands for steam.

The tubes are of the lap-welded wrought-iron type and are 4 inches in external diameter. The headers are made of car-wheel steel castings (manganese steel containing about 13 per cent. of manganese). The holes in the headers are accurately sized and made tapering, and the tubes are fixed into them by using an expanding tool. The connections to the steam and water drums are made by using short tubes which are expanded into bored holes.

Characteristics of Centrifugal or Turbine Pumps

5. In connection with a centrifugal, or turbine, pump whose economical duty is 1,000 g.p.m., 700 feet head at 750 r.p.m., coupled direct to an electric motor :—(*a*) If the speed is increased to 1,000 r.p.m., what is then the economical head and quantity per minute ? (*b*) If the motor were overloaded at the economical duty of 1,000 g.p.m., 700 feet head, how could you reduce the load whilst maintaining the speed ? (28)

A. If the speed of the pump is increased from 750 r.p.m. to 1,000 r.p.m. the following results might be obtained :—

(*a*) The quantity of water varies directly with the speed.

$$\therefore \text{Quantity} = 1000 \text{ gallons} \times \frac{1000}{750} = \underline{1333 \text{ gallons per min.}} \qquad (1)$$

(*b*) The head of water varies directly with the speed squared.

$$\therefore \text{Head} = 700 \text{ feet} \times \left(\frac{1000}{750}\right)^2 = \underline{1245 \text{ feet}} \qquad . \quad . \quad (2)$$

(*c*) If the motor were overloaded at the economical duty of 1,000 gallons per minute at 700 feet head, the discharge opening could be reduced by partly closing the master valve on the delivery column. This would reduce the quantity of water and increase the head. The adjustment would therefore result in a reduction of the load on the motor, owing to the reduction in lbs. of water being much greater than the increase in head.

Shaft Pumping Plant

6. The feeder of water to be pumped up a shaft 1,000 feet deep is 500 g.p.m. in winter and 250 g.p.m. in summer. Electric power is to be used. Reliability is important. Discuss the types of pump that might be used and say which you would adopt, and in what numbers and capacities. Give reasons for your choice. (28)

A. The types of pumping plant for the above conditions might include the three-throw ram pump and the turbine pump. Both types of pump are very reliable, but the ram pump would take up more space than the turbine pump, while at the same time its cost of installation and running would be greater.

The turbine plant would be the best plant for the work, owing to the fact of its being both economical and reliable. The plant should be a duplicate one, say a turbine pump for constant use,. and a three-throw ram pump as stand-by plant.

Dimensions of a turbine pump capable of dealing with 750 gallons per minute in winter time and working 16 hours per day, and the same quantity in summer time when working 8 hours per day :—

$$\text{Area of outlet} = \frac{\text{Cu. ft. per sec.}}{5{\cdot}5} \times 144 = 53 \text{ sq. in. (velocity } 5\tfrac{1}{2} \text{ ft. per sec.)}$$

$$\text{Diameter of outlet} = \sqrt{53 \times \tfrac{14}{11}} = 8 \text{ inches.}$$

$$\text{Diameter of delivery pipes (3 ft. per sec.)} = 11\tfrac{1}{4} \text{ inches.}$$

$$\text{Diameter of impellers} = 8 \times 2 = 16 \text{ inches.}$$

$$\text{Width of impellers} = \tfrac{16}{20} = 0{\cdot}8 \text{ inch.}$$

$$\text{Total head} = 1000 + 20 = 1020 \text{ feet.}$$

$$\text{Speed of pump} = 1450 \text{ r.p.m., 4 poles, 50 cycles.}$$

Stages of pump (n) can be calculated as follows :—

$$\text{r.p.m.} = \frac{\sqrt{\dfrac{v^2}{n}} \times 60}{\text{Circ. of impeller}}$$

$\therefore$ r.p.m. $\times \sqrt{n} \times \frac{16}{12} \times \frac{22}{7} = \sqrt{2g \times 1020} \times 60 \quad (v^2 = 2gh)$

$$\text{and } n = \left(\frac{256 \times 60 \times 12 \times 7}{1450 \times 16 \times 22}\right)^2 = (2\cdot5)^2$$
$$= 6 \text{ stages}$$

$$\text{Electrical H.P. of motor} = \frac{750 \times 10 \times 1000}{33{,}000} \times \frac{100}{60} = 380 \text{ E.H.P.}$$

Dimensions of three-throw ram pump capable of dealing with 500 gallons per minute as a stand-by pump :—

Gallons per minute allowing 10 per cent. slip $= 500 \times \frac{100}{90} = 556$.

Effective pumping speed, 120 ft. per min., 40 ft. for each ram.

$$\text{Diameter of rams} = \sqrt{\frac{556}{0\cdot034 \times 120}} = 11\tfrac{1}{2} \text{ inches.}$$

$$\left(\begin{aligned} \text{g.p.m.} &= \frac{d^2 \times \frac{11}{14}}{144} \times 6\tfrac{1}{4} \times \text{speed} \\ &= d^2 \times 0\cdot034 \times \text{speed} \end{aligned}\right)$$

Stroke 12 inches, number of strokes per minute 40, r.p.m. 40.
Motor 6 poles, 50 cycles, 960 r.p.m.

$$\text{Electrical H.P. of motor} = \frac{556 \times 10 \times 1000}{33{,}000} \times \frac{100}{60} = 280 \text{ E.H.P.}$$

Gearing ratio $= \frac{960}{40}$ or $\frac{24}{1}$. Say 2 stages, of 4 to 1 and 6 to 1.

Factor of Safety in Winding-Rope

7. A steel winding-rope is $1\frac{1}{4}$ inches in diameter, and has a breaking strength of 65 tons. It weighs 2·5 lb. per foot. The loaded cage and attachments weigh 5 tons. The length of the rope from the pithead sheave to the cage at the bottom of the shaft is 900 feet. Calculate the static factor of safety in the rope. If the acceleration is 3 feet per sec.^{2}, what is the actual factor of safety in the rope as the wind commences ? (28)

A. Weight of winding-rope $= \frac{900 \times 2\cdot5}{2240} = 1$ ton

Loaded cage and attachments $= 5$ tons

Total static load $= 6$ tons

$\therefore$ Static factor of safety in the rope $= \frac{65}{6} = \underline{10\cdot83}$ . . (1)

Load due to acceleration at start $=$ mass $\times$ acceleration
$= \frac{6}{32} \times 3$
$= 0\cdot56$ ton.

$\therefore$ Actual factor of safety in the rope $= \frac{65}{6\cdot56} = \underline{9\cdot9}$ (almost 10) (2)

NOVEMBER 1932 EXAMINATION

Electrical Section

Star-Delta Starters

1. Describe a star-delta starter for a three-phase squirrel-cage motor of about 10 H.P., including automatic overload trips and a low-voltage release. Show diagrams of connections for the "starting" and the "running" positions. (28)

A. When the star-delta starter is used for starting squirrel-cage motors, the normal operating connection of the stator is *mesh*, but the switch is so arranged that when the handle is placed in the "starting" position the stator windings are *star*-connected, the voltage impressed therefore on these windings being the normal operating voltage divided by $\sqrt{3}$, or about 58 per cent. of the line volts. In this way the current taken by the motor at starting is

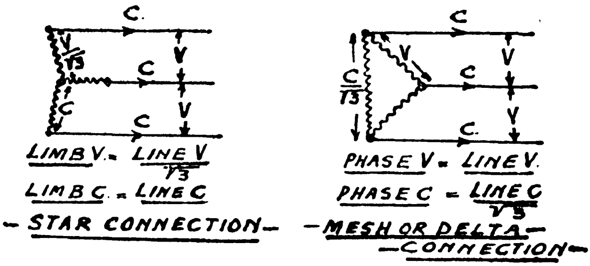


FIG. 119.—Star and Mesh or Delta Starter Connections.

considerably reduced. When the motor is running at normal speed, the switch handle is put over to the "running" position and the motor operates with the stator windings *mesh*- or *delta*-connected, the full line voltage being impressed on the winding. The automatic overload trips are connected in series with each phase of the supply to the motor and may be set to trip the circuit-breaker when the current exceeds about 50 per cent. of the full-load current of the motor. The low-voltage release is connected in parallel across any two phases, so that in the event of failure of the supply the switch handle falls to the "off" position.

Fig. 119 shows diagrams of star and mesh or delta connections, respectively. Fig. 120 shows the required connections for a star-delta starter.

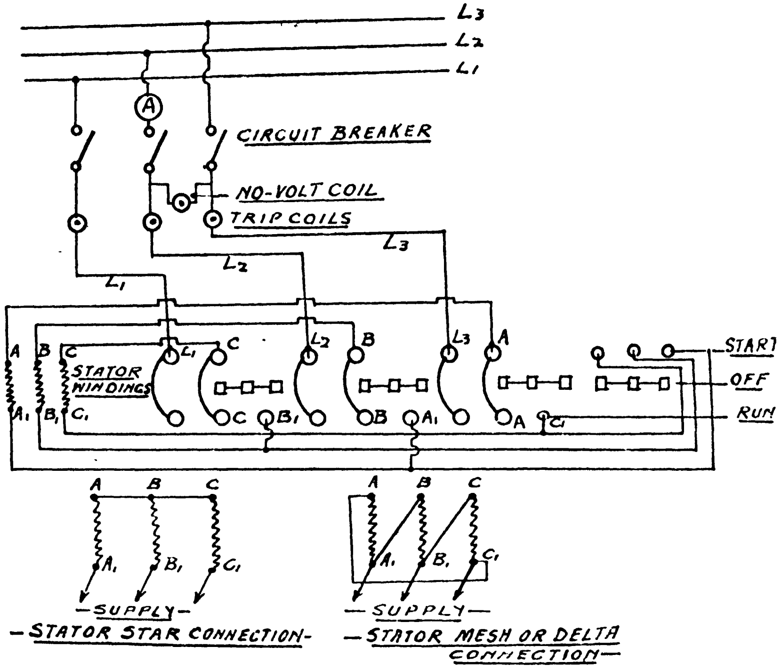


FIG. 120.—Diagram of Connections for Star-Delta Starter.

Synchronous Induction Motor and Power Factor

2. Describe a synchronous induction motor, say for use with three-phase, 2,000-volt, 50-cycle current. If the motor is taking 100 kW. whilst the rest of the load on the system is 120 kilovolt-amperes at 0·8 lagging power factor, what must be the power factor of the motor to make the general power factor unity ? (28)

A. A synchronous induction motor is one which is designed to run on an alternating-current supply at a constant speed, depending entirely on frequency, without slip.

It is similar in appearance to a slip-ring motor, but it has a wider air gap. A direct-current low-voltage generator, or exciter, is connected to the rotor shaft of the motor. This exciter supplies current to the rotor circuit.

The synchronous motor may be used for power application and for power-factor improvement, and works most efficiently on a constant load, such as with turbine pumps and ventilating fans. The motor is started by means of resistance in the rotor circuit, the resistance being gradually cut out until normal speed is attained, the exciter then generating its normal voltage. When the change-over switch is operated to introduce the direct current into the rotor circuit, the rotor immediately falls into synchronism. When the load is applied, the exciter rheostat is adjusted to give the desired power factor, or if unity power factor is required, the rheostat is adjusted to give the lowest reading on the main ammeter. The starting torque and overload capacity are similar to those of a slip-ring motor, as the motor will start against a torque up to $2\frac{1}{2}$ times full load, and will synchronise against loads up to $1\frac{1}{4}$ times the full-load torque.

$$\text{kW. component of inductive load} = \text{kVA.} \times \text{power factor}$$
$$= 120 \times 0{\cdot}8 = 96\,\text{kW.} \qquad . \quad (1)$$

$$\text{Wattless component} = \sqrt{\text{kVA.}^2 - \text{kW.}^2}$$
$$= \sqrt{120^2 - 96^2} = 72\,\text{kVA.} \quad . \quad (2)$$

The 72 kVA. wattless lagging component must be neutralised by 72 kVA. loading, and the load on the motor is 100 kW.

$$\therefore \text{Total kVA. of motor} = \sqrt{\text{kW.}^2 + \text{wattless kVA.}^2}$$
$$= \sqrt{100^2 + 72^2} = 123\,\text{kVA.} \qquad . \quad . \quad (3)$$

$$\text{Power factor} = \frac{\text{kW.}}{\text{kVA.}} = \frac{100}{123} = \underline{0{\cdot}81\ (\text{leading})} \quad . \quad . \quad (4)$$

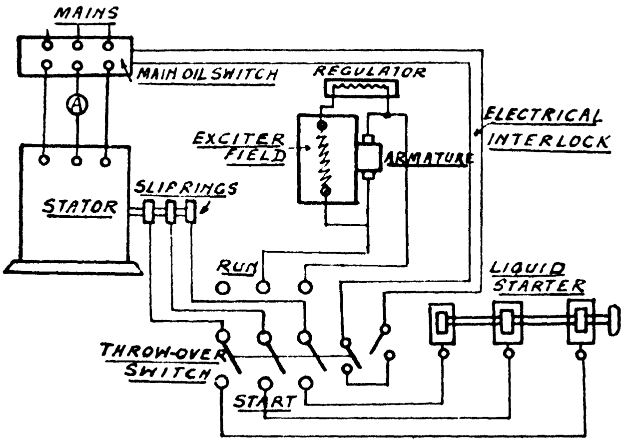


FIG. 121.—Diagram of Connections for Synchronous Induction Motor.

Fig. 121 shows the diagram of connections for a synchronous induction motor.

Mechanical Section

Steam Distribution in Engine Cylinders

3. Describe how steam is distributed in the cylinder of an engine by a slide-valve and eccentric. Make a simple line diagram showing the positions of valve on ports, piston in cylinder, and crank and eccentric in relation on the shaft at the point when cut-off occurs at ¾-stroke. Indicate by an arrow the direction of travel of the piston. (28)

A. The slide-valve, working on the face of the steam ports to the engine cylinder, obtains its reciprocating motion by means of an eccentric fixed on the crankshaft of the engine. This valve governs the distribution of steam to the cylinder by opening the port to steam at the desired part of the stroke, and by allowing the exhaust steam to escape from the cylinder at the same time.

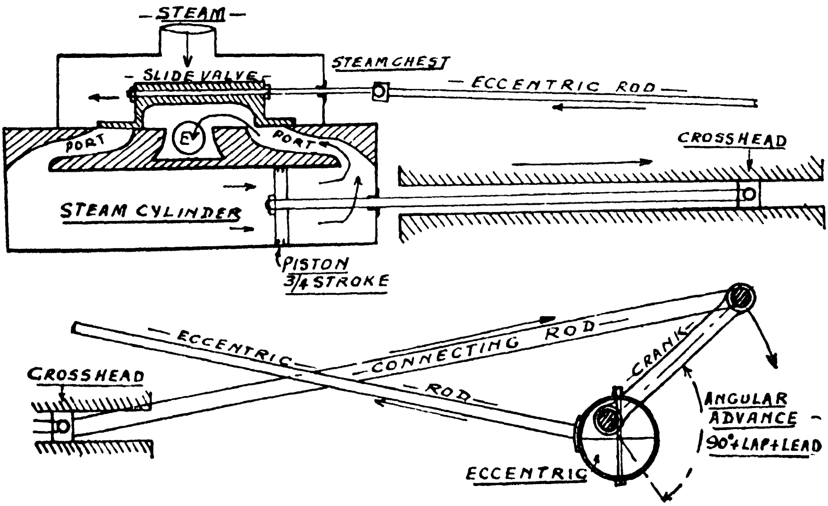


Fig. 122.—Steam Cylinder, Slide-valve, Crank and Eccentric.

It must be of sufficient extent to close both ports when the piston is nearing the end of its stroke.

Fig. 122 shows the drawing asked for in the above question. The piston is at ¾-stroke and is moving towards the right. The slide-valve has just cut off the steam from the left-hand port and is still allowing exhaust steam to leave the cylinder by the right-hand port. The valve is moving towards the left with increasing speed, so as to give steam admission to the right-hand port and an exhaust opening to the left-hand port when the piston reaches the end of the stroke. The motion of the slide-valve caused by the eccentric is similar to that of the piston driven by the steam, but the valve movement is more than 90 degrees in advance of the piston, this being known as the angular advance of the eccentric in front of the crank, so that the admission of steam is obtained at the right time.

Apparatus on Winding Plant for Information and Safety

4. What apparatus and devices are used on a steam winding plant at a deep shaft to afford information and to ensure safety? What are the points that should receive regular attention in keeping these appliances in good working order? (28)

A. The following apparatus and devices should be included in winding plants to afford information and to ensure safety :—

(i) A good type of depth indicator showing clearly the position of the cages in the shaft. Daily examination and frequent checks should be carried out.

(ii) Well-defined markings on the drum cheek to assist the engine-man in changing decks and showing position of mid-shaft insets. Frequent checking should be carried out.

(iii) A warning signal should be given to the engineman when the cage is nearing the surface. Daily examination should be made. All the above apparatus should be maintained in good working order and adjustment. Correct adjustments should be made weekly and when new chains and ropes are fitted.

(iv) Safety hooks should be installed for arresting and suspending cages in the head-gear in the event of an overwind taking place. Refitting should be carried out every three months.

(v) Overwinding gear should be fitted to the winding-engine to prevent overwinding and also overspeeding in the shaft. It should be checked weekly for cage positions in the shaft and by trial in the shaft for speeding.

(vi) A steam brake should be fitted to the winding-engine to be of use to the engineman independent of the overwinding and over-speeding attachment. This brake is necessary to retard the engine and cages near the end of the wind. Its effectiveness should be checked at frequent intervals.

(vii) A reversing handle should be fitted to the engine for use in changing the direction of running of the engine and for controlling the steam to the engine cylinders when combined with a suitable link motion. Its effectiveness should be checked frequently.

(viii) Suitable signalling appliances should be installed for communication between the engineman, banksman, and underground insets. They should be luminous and show clearly when men are in the shaft cages. These devices should be maintained in good working order and adjustment. Daily examination of them should be carried out by a competent person appointed by the Manager. Weekly examinations and adjustments should also be carried out by the same person.

Construction of a Lancashire Boiler

5. Describe the construction of a modern Lancashire boiler for a working pressure of, say, 120 lb. per sq. inch. Deal in detail with the shell and tubes. A list of the fittings and mountings is not wanted. (28)

A. Fig. 123 shows a longitudinal section of a Lancashire boiler, including the details asked for in the above question. Such a

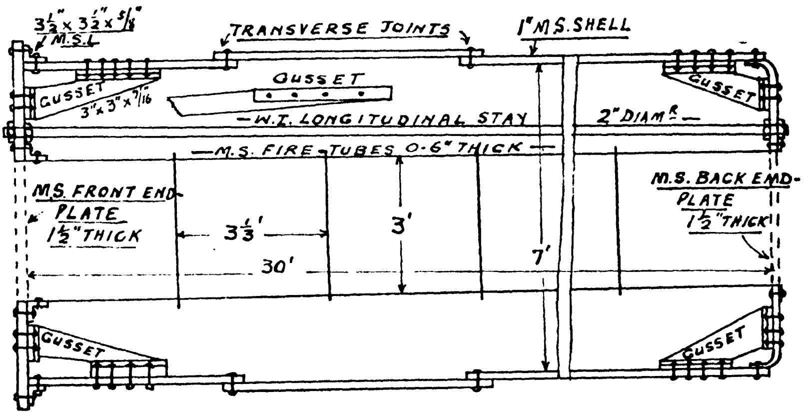


Fig. 123.—Longitudinal Section of Lancashire Boiler.

boiler might be 30 feet long by 7 feet 6 inches diameter, with two fire-tubes 3 feet in diameter. The weakest parts of the boiler are the flat surfaces, the fire-tubes, and the openings in the shell for mountings, manholes, etc.

The boiler shell is made of mild steel plates having single-riveted lap transverse joints, and double-riveted lap longitudinal joints. It is secured to the flat end-plate at the front by an angle ring riveted to the shell and the plate respectively, and to the back end-plate by the plate itself being flanged and riveted to the shell. Additional security is obtained by gusset stays and by longitudinal stay-bolts between the respective end-plates. All joints of the boiler shell and ends are caulked after being riveted, so as to make them water-tight.

Thickness of shell :—

Let P = Working pressure of 120 lb./sq. in.
D = Diameter of shell, 90 ins.
F = Factor of safety, 8.
t = Thickness of shell in inches.
T_s of mild steel = 63,000 lb./sq. in.
e = Joint efficiency, 0·75.

Then working stress × Factor of safety = Safe stress in material × Joint efficiency

$$P \times D \times F = 2t \times T_s \times e$$
$$120 \times 90 \times 8 = 2t \times 63{,}000 \times 0{\cdot}75$$
$$\therefore t = \frac{120 \times 90 \times 8}{2 \times 63{,}000 \times 0{\cdot}75} = \frac{64}{70} \text{ inch, say 1 inch.}$$

The boiler fire-tubes are 3 feet in diameter with section length of $3\frac{1}{2}$ feet. They are secured to the end-plates by turned-up flanges

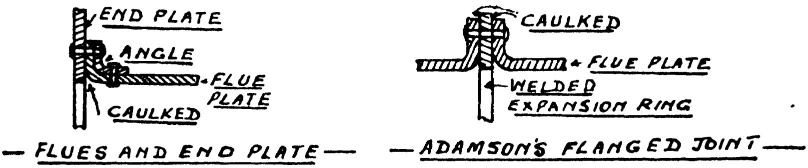


Fig. 124.—Boiler Flue Joints and Connections.

and angle ring to prevent grooving, as shown in Fig. 124. All longitudinal seams of the flues or tubes are solid-welded. Fig. 124 shows also how the different sections of the tubes are connected by Adamson's flanged joint, which allows for expansion and contraction, while at the same time the rivets are not subjected to the direct action of the fire.

Thickness of tubes :—

$$t = \frac{P \times d}{8000}$$
$$= \frac{120 \times 36}{8000} = 0{\cdot}54 \text{ inch, say } \tfrac{9}{16} \text{ inch,}$$

where P = Working pressure, 120 lb./sq. in.
and d = Diameter of flue, 36 inches.

This empirical formula allows for a direct stress of not more than 2 tons per sq. inch on the tube.

For additional details of the Lancashire boiler, see pages 247–8 and Fig. 115.

Steam Sinking Pump

6. Describe a reciprocating steam pump suitable for use in a sinking shaft, and state the provisions that may be made for better dealing with gritty water. What size of pump would you choose for raising 200 g.p.m. to a height of 300 feet with steam at a pressure of 60 lb./sq. in. at the stop-valve ? (28)

A. Fig. 125 shows a sectional elevation of Evans's straight-line differential-ram sinking pump. Such a pump is strong and efficient and is suspended in the pit by strong chains, which also carry stretchers for the various pipes. It is suitable for working where gritty water is to be pumped. A suitable filter-box might be included in the suction branch for extracting grit from the water.

During the downstroke of the large ram, one-half of the delivery water passes through the retaining valve into the rising column, the remaining half being short-circuited into the ram casing to follow the downward movement of the ram. During the upward, or suction, stroke of the large ram, the remaining half of the delivery water is forced through the retaining valve into the rising column by the shoulders on the large ram. This pump therefore pumps an equal volume of water at each stroke and has a capacity equal to that of a double-acting ram pump of the same dimensions as the small ram. It works with a minimum of shock and loss of energy.

Size of pump :—

Gallons per min. $= \dfrac{200 \times 100}{90} = 222$, allowing 10 per cent. slip.

Speed of pump, 120 feet per min.

$$\text{Diameter of large ram} = \sqrt{\frac{\text{Gallons per min.}}{\frac{11}{14} \times \frac{1}{144} \times 60 \times 6\frac{1}{4}}}$$

$$= \sqrt{\frac{222}{0{\cdot}034 \times 60}} = 11 \text{ inches.}$$

$$\text{Diameter of small ram} = \sqrt{\frac{121}{2}} = 8 \text{ inches.}$$

Stroke, 2 feet ; strokes per min., 60.

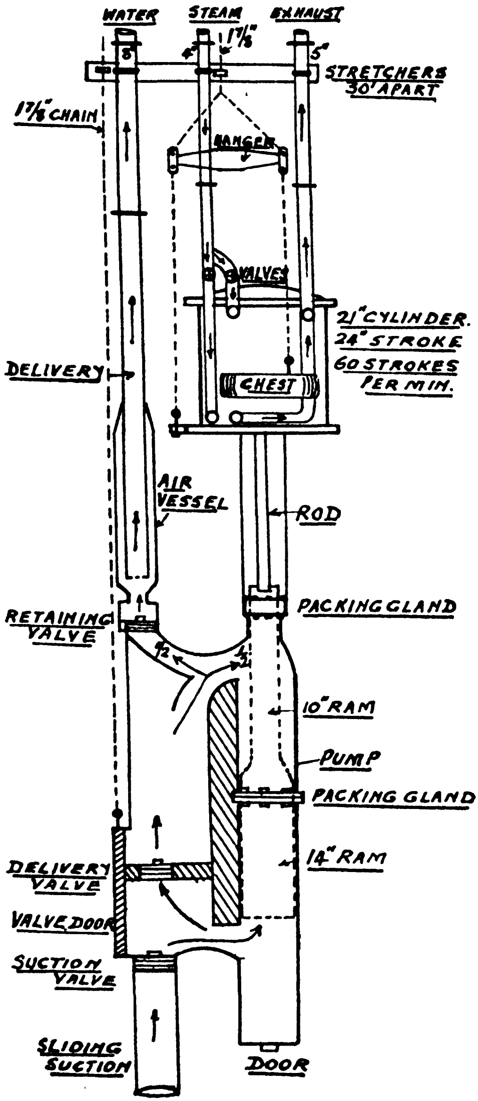


FIG. 125.—Evans's Differential-Ram Sinking Pump.

Size of steam cylinder (mean effective steam pressure 40 lb./sq. in.) :—

Work of engine × Efficiency = Work done in pumping × Friction allowance.

$P.L.A.N. \times$ Efficiency = Gallons per min. × 10 × Head × Friction allowance,

where P = Steam pressure = 40.
L = Length of stroke = 2.
A = Area of cylinder = $D^2 \times \frac{11}{14}$.
N = Number of strokes per min. = 60.

$$40 \times 2 \times D^2 \times \tfrac{11}{14} \times 60 \times \tfrac{70}{100} = 222 \times 10 \times 300 \times \tfrac{100}{85}$$

$$\therefore\ D^2 = 303 \text{ and } D = 17\tfrac{1}{2} \text{ inches.}$$

MAY 1933 EXAMINATION

Electrical Section

Electric Power Calculation

1. A three-phase motor works at 240 B.H.P. when supplied with electric current at 3,000 volts between phases. Its efficiency is 85 per cent. and power factor 0·75. Taking the voltage at the generator terminals as 3,300 volts between phases, what is the output of the generator in kilowatts to supply the motor ? (28)

A. Amperes at motor—

$$= \frac{\text{B.H.P.} \times 746}{\text{Volts} \times \text{Efficiency} \times \text{P.F.} \times \sqrt{3}} = \frac{240 \times 746}{3000 \times 0{\cdot}85 \times 0{\cdot}75 \times 1{\cdot}732} = 54$$

Output of generator—

$$= \frac{V \times A \times \text{P.F.} \times \sqrt{3}}{1000} = \frac{3300 \times 54 \times 0{\cdot}75 \times 1{\cdot}732}{1000} = \underline{231{\cdot}5 \text{ kW.}}$$

Electrical Winding Plant

2. Describe an electrical winding plant suitable for a large output from a deep shaft. What are the adverse effects of a winding load on an electric power system and how may they be mitigated ? (28)

A. Fig. 126 shows a plan view of an electrical winding plant suitable for an output of 600 tons in 8 hours' winding from a depth of 780 feet. The winding cage weighs 30 cwt. and carries two trams, each 16 cwt. gross and 6 cwt. tare. Winding in the shaft is accomplished in 40 seconds. The winding motor is of the three-phase, alternating-current, induction type of 290 E.H.P. normal and 355 E.H.P. at peak load, with a speed of 478 revs. per min.

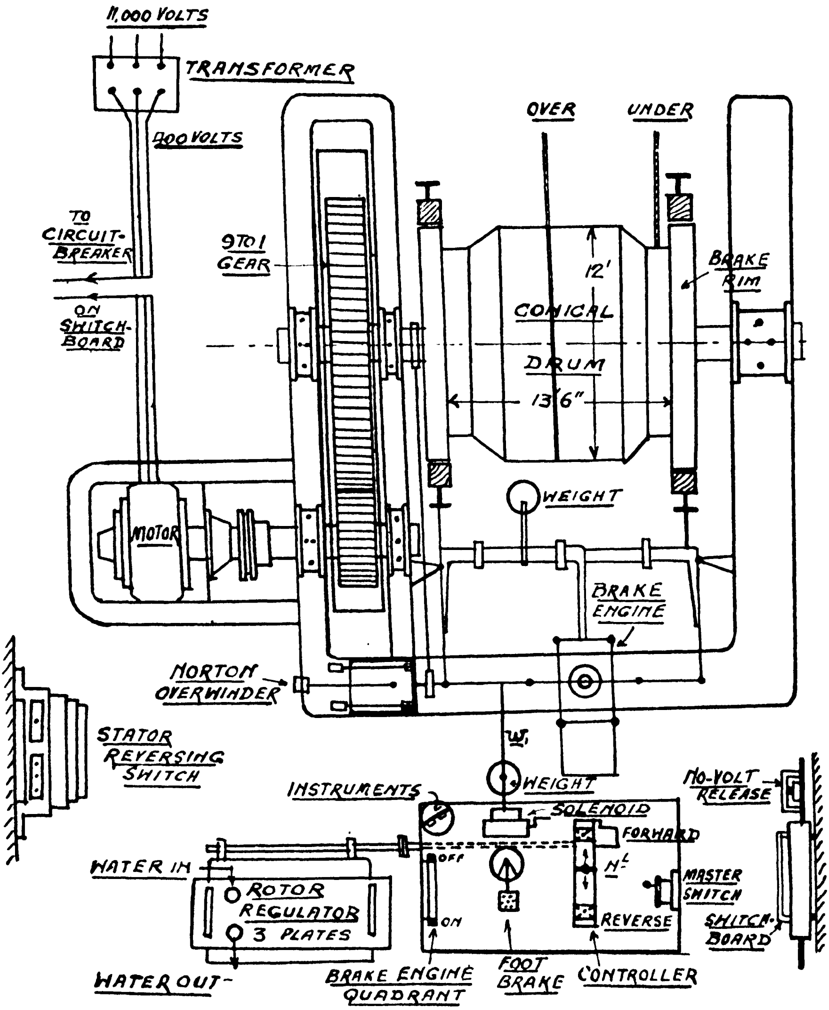


FIG. 126.—Plan View of Electrical Winding Plant.

The motor is geared to the cylindro-conical winding drum by a 9 to 1 totally enclosed gear. The motor is controlled by a simple lever, with central neutral position, which operates the dip plates of the liquid controller, and any speed from dead slow to high can easily be obtained.

During ordinary winding conditions braking is carried out by either the foot-brake or the compressed-air braking system, but not both. Emergency braking consists of (*a*) a master switch for the engineman and a no-voltage automatic switch, a solenoid magnet being used to release the emergency brake lever w_1, and (*b*) an over-speeding and overwinding Norton Controller, which is also connected with the emergency braking system.

Electrical winding plants may frequently consume excessive current if care is not taken in operating them, thus necessitating the installation of more robust electrical fittings, and producing a severe drain on the electric power system. The following points should be properly attended to :—

(i) Single-deck cages should be used, or simultaneous decking of cages with more than one deck, to prevent heavy demands for current.

(ii) Starting and accelerating the load should be so carried out as to keep the current below a predetermined figure.

(iii) Negative loads near the end of the wind should be avoided by counterbalancing the engine by means of suitable drums, such as conical, spiro-conical and cylindro-conical types.

Switchgear for Motor—Definition of Terms

3. In connection with switchgear for starting and controlling a three-phase motor, what is meant by the terms : (*a*) loose-handle switch ; (*b*) automatic overload trip with time-lag device ; (*c*) low-volt (or no-volt) release ; (*d*) potential transformer ; (*e*) rupturing capacity ? (28)

A. A loose handle, or free handle, is fitted to switches and circuit-breakers for closing, and quick breaking, of electrical circuits by hand release. When circuit-breakers are fitted with free handles, it is not possible to hold the breaker in the " on " position if a fault exists in the circuit which it protects.

An *automatic overload coil with time-lag device* is a coil consisting of a few turns of heavy wire capable of conveying the main current and connected in series with each phase of the supply. In the centre of the coil an iron plunger is free to move, so that when the current exceeds a certain amount, the iron plunger is pulled up and trips the switch. The time-lag device is a small piston fixed to the iron plunger, which operates in a small cylinder of oil, its purpose being to retard tripping of the circuit.

A *no-voltage release* consists of a large number of turns of fine wire arranged in a coil and connected across any two lines. When the switch is closed the coil is energised, and an iron core within the coil is pulled down to allow the switch to remain in the " on " position. If the supply fails, or the voltage falls below a certain amount, the coil loses all or part of its magnetism, and by the aid of a small spring the iron core moves upward and trips the switch.

A *potential transformer* is one which is used in conjunction with voltmeters for measuring high voltages. The primary side of the transformer is connected across any two lines of the high voltage, while the secondary terminals of the transformer are connected to the voltmeter at a lower voltage. The voltmeter is calibrated to read the line volts.

Rupturing capacity is a term applied to automatic switchgear and is used to denote the actual kVA. which the switch will rupture. The breaking capacity of a circuit-breaker is influenced by numerous factors, such as material used for the sparking contacts, length of break, speed of break, and number of successive operations. A breaker installed close to the bus-bars of a large station may be called upon to intercept 200 times the full-load current of the circuit it protects. The required rupturing capacity is determined by the location of the breaker relatively to the source of supply, or by the resistance in the circuit between the breaker and the source.

Mechanical Section

Metals used in Construction of Winding-Engine

4. Name the metals and metal alloys that are generally used in the construction of a steam-driven winding-engine. Indicate representative parts of the engine for which each material is suitable, and give reasons for your choice. (28)

A. (*a*) Engine parts which are not subjected to shock and vibration and which are made from castings :—

Engine cylinders Pistons Cylinder covers Guides Slide-valves Eccentric blocks	Cast iron, or sometimes cast steel for greater strength.

(*b*) Piston rings: From castings of cast iron.

(*c*) Engine parts as above, but which require linings of other metals to reduce friction and wear :—

Glands—cast steel, lined with brass or gunmetal bushes.

Slides and slippers—cast iron or cast steel, lined at the base with white-metal.

Eccentric straps—cast iron, lined with white-metal; or cast steel, forged mild steel, gunmetal.

(*d*) Engine parts subject to vibration, shock, twist, thrust and tension :—

Crossheads Piston rods Links, fittings, levers Cranks Gib and cotter	Mild steel.
Crankshafts Connecting-rods Eccentric rods	Forged mild steel.

(*e*) Bearings: Gunmetal, bronze, white-metal, for toughness, even wear and minimum friction.

Brakes for Winding-Engines

5. Describe a good arrangement of brakes and brake-operating gear for a winding-engine at a shaft, say 500 yards deep, where the loaded cage weighs about 5 tons gross and the empty cage about 2 tons. What would be the most severe duty to be performed by the brakes? (28)

A. Fig. 127 shows the arrangement of the *Whitmore brake* for winding-engines. The brake posts with shoes are operated by a brake engine, and they are pressed against the brake rim of the drum to retard its movement.

The position of the piston in the brake engine corresponds exactly with the position of the hand lever in the brake quadrant. Weights and rods are suspended from the crosshead of the brake engine. The weights put on the brake, and steam applied to the underside of the piston takes the brake off. Under working conditions, the full effect is applied partly by the weights and partly by steam to the top of the piston of the brake engine.

The brake lever *E*, in moving down, draws the posts towards the drum, the maximum effort being applied when the spring is fully

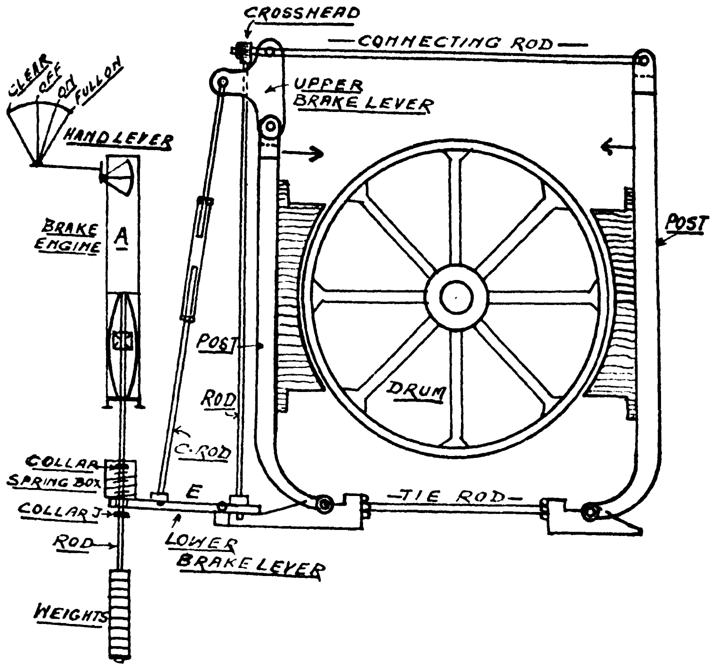


FIG. 127.—Whitmore Brake for Winding-Engine.

compressed, and the minimum when the collar J is in contact with the underside of the lever E.

A ratchet arrangement is fitted on the crosshead of the upper brake lever for automatically adjusting the brake blocks to allow for wear.

The most severe duty to be performed by the brakes is to arrest the motion of the drum quickly in the event of the controller coming into action due to high winding speed or to overwinding.

Calculation of Power of Haulage Motor

6. An incline 500 yards long, dipping 1 in 4, is laid with a single track. Tubs are to be hauled up the incline in sets of eight, using a wire rope $\frac{3}{4}$-inch diameter, weighing 1 lb. per foot. The tubs tare 600 lb. and carry 1,200 lb. of coal. The output in 7 hours is to be 150 tons.

State what diameter of drum you would adopt, what would be the full speed of the tubs, and calculate the brake horse-power of an electric motor to work the haulage. (28)

A. Load for motor

$$= \text{Gravity} + \text{friction} + \text{rope gravity} + \text{rope friction}$$

$$= \frac{8 \times 1800}{4} + \frac{8 \times 1800}{60} + \frac{1500 \times 1}{4} + \frac{1500 \times 1}{20}$$

$$= 3600 + 240 + 375 + 75$$

$$= 4290 \text{ lb.}$$

Tons per hour $= \frac{150}{6} = 25$ (allowing for stops and changing)

$$\text{Sets per hour} = \frac{25 \times 2240}{8 \times 1200} = 6$$

Time to make a single run $= \frac{60}{12} = 5$ mins.

Speed of set = 500 yards in 5 mins. or 300 feet per min. . (1)

Size of drum = 80 times diameter of rope $= \frac{80 \times \frac{3}{4}}{12}$

= 5 feet diameter . . (2)

B.H.P. of motor $= \frac{4290 \times 300}{33{,}000} \times \frac{100}{75}$ (gear efficiency)

$= 52$ . . . . (3)

Furnaces and Grates for Water-Tube Boilers

7. Discuss briefly the types of furnace and of grates that may be used with water-tube boilers, the fuel being coal. (28)

A. *Furnaces for Water-Tube Boilers.*—The ordinary simple spring-arch of firebrick for boiler furnaces has been extensively used in connection with water-tube boilers, but is being rapidly replaced by the suspended-arch type.

The latter consists of a series of standard firebrick blocks, each attached independently to overhead steel girders to form a flat roof. This is now standard practice for water-tube boilers, and various types of wall are available; for example, the "Liptak" double-suspended arch combined with Bigelow walls of various design. The Liptak arch is constructed with a series of strong and heavy steel beams of H-section, fixed across the roof of the combustion chamber, from which are suspended cast-iron brackets and hangar beams for the firebrick blocks, as shown in Fig. 128.

The advantages of the above system are as follows: (1) The margin of safety is increased by having a double row of blocks. (2) Larger blocks have corrugated sides to prevent pieces falling

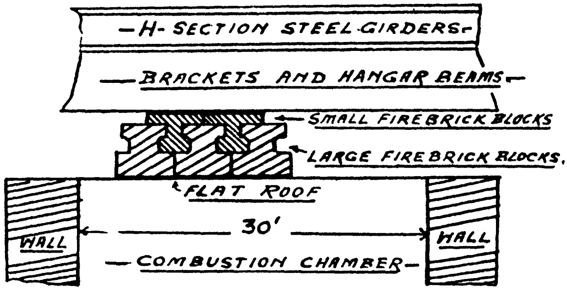


FIG. 128.—Liptak Double-suspended Furnace Arch.

out when broken. (3) The right shape is obtained for efficient and uniform heat radiation. (4) Each block is independent of its neighbours, and no lateral strain is produced. (5) Broken bricks can be removed easily and replaced by new ones.

Grates for Water-Tube Boilers.—Fig. 129 shows the arrangement of the balanced-draught chain-grate stoker as used in connection

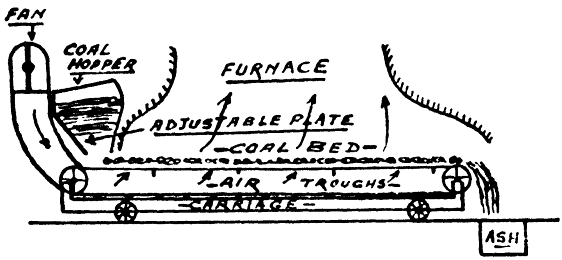


FIG. 129.—Balanced-draught Chain-grate Stoker.

with water-tube boilers. Air troughs exist between the upper and lower sections of the grate, air being supplied to each 2-foot length of grate by a forced-draught fan. The fuel is controlled by the grate travel and a guillotine door at the front. The coal and air can be increased or diminished at will to suit the existing conditions of load.

Fig. 130 shows diagrammatically the arrangement of the Riley Stoker for water-tube boilers. Coal is burned on grates which have a reciprocating motion, making the fuel-bed open, porous, and free from clinkers.

The retorts between the grates give thorough distillation of the volatile compounds of the coal and thus smokeless combustion. The overfeed air supplying the grates burns all the coke out of the

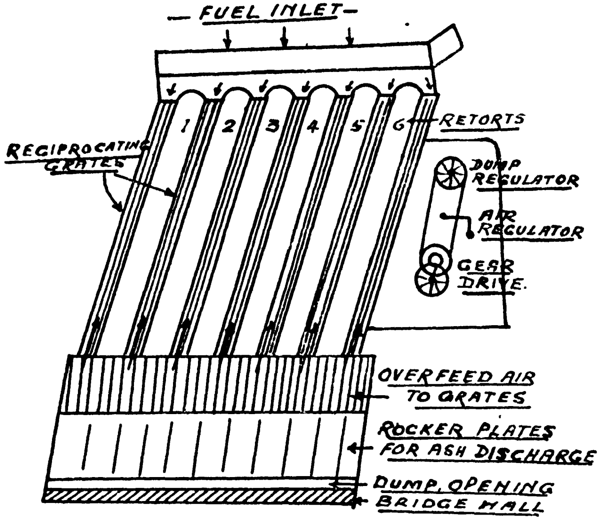


FIG. 130.—Plan View of the Riley Stoker.

fuel before it reaches the discharge opening. The rocker dump plates agitate, crush, and automatically discharge the ash, and at the same time keep the bridge wall clear of clinkers and ash. Forced draught is applied to give a rapid regulation of the fire and a ready response to increased demands for steam. This stoker is suitable for burning bituminous coal containing a high percentage of fine dust. It is not suitable for burning coke breeze or coals containing a low percentage of volatile matter.

Air-Compression

8. In connection with the compression of air, what are the reasons for the adoption of (*a*) Compression in two stages (compound); (*b*) an intercooler between the stages; (*c*) water-jackets on the cylinders? Make a diagrammatic sketch of an intercooler. (28)

A. When air is compressed in a single cylinder its temperature is increased, and more power is required to drive the compressor. The increased temperature cannot be maintained owing to the air cooling-down in transmission, and thus the extra power applied during compression is lost. To reduce this loss the air temperature during compression should be as low as possible, and this is accomplished by compressing the air in stages, say in two or more cylinders, and by having an efficient cooling arrangement between each of the stages. To reduce the temperature as much as possible during compression, the air cylinders and cylinder covers are water-jacketed round the barrels and at the ends. The cooling water is circulated by a small pump.

Fig. 131 shows in elevation (part sectional) the arrangement of

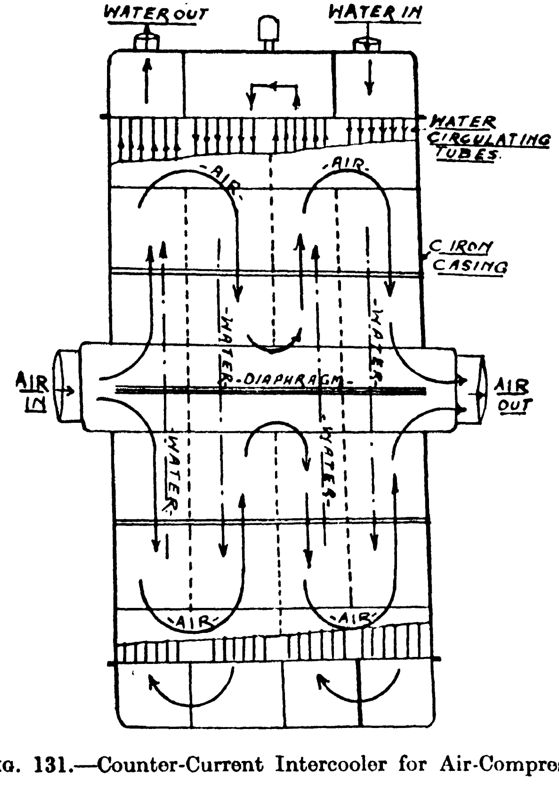


FIG. 131.—Counter-Current Intercooler for Air-Compressor.

the intercooler, in which the air is cooled by coming into contact with tubes through which water is continually circulated. Air enters the cooler at the one side and leaves it at the opposite side after circulation. Water is circulated through 200 tubes of solid-drawn brass, $\frac{3}{4}$-inch diameter, at the rate of 80 to 100 gallons per minute. The capacity of this intercooler is 2,000 cubic feet of free air per minute.

NOVEMBER 1933 EXAMINATION

Electrical Section

Types of Electric Motors

1. In the case of electric motors for three-phase current, distinguish between the following types: (*a*) Squirrel-cage or short-circuited rotor; (*b*) slip-ring or wound rotor; (*c*) synchronous with salient poles; (*d*) synchronous induction. Which types would you use for: (i) a 100 H.P. direct haulage; (ii) a 10 H.P. centrifugal pump; (iii) a large continuous-running fan; (iv) a large air-compressor? (28)

A. (*a*) Squirrel-cage or short-circuited rotor. See page 251.

(*b*) Slip-ring or wound rotor. See pages 236–7 and Figs. 110, 111, and 112.

(*c*) Synchronous with salient poles.—In this motor a system of poles, projecting inwards or outwards from the main yoke ring, is used, similar to those on the field system of a flywheel alternator. The term is used to distinguish such poles from poles which do not project, such as those produced by the stator winding of an induction motor, or by the revolving field of a turbo-alternator with a distributed field winding. A synchronous motor with salient poles is designed to run on an alternating-current supply at a constant speed, depending solely on frequency, without slip. It is usually started by the aid of a pony motor, and in such cases it can only be suitable where a low starting torque is required.

(*d*) Synchronous induction. See pages 259–60 and Fig. 121.

(i) A 100 H.P. direct haulage motor should be of the slip-ring or wound rotor type for safe and easy starting of the load.

(ii) A 10 H.P. centrifugal pump motor should be of the squirrel-cage or short-circuited rotor type for strength, reliability and constant use.

(iii) A large continuous-running fan should be driven by a synchronous induction motor for easy starting under load and constant speed.

(iv) A large air-compressor should be driven by a slip-ring motor with short-circuiting and brush-lifting devices.

Electrical Plant for a Haulage

2. A single-drum direct haulage gear is driven by a 120 B.H.P. motor taking three-phase, 50-cycle, 3,000-volt current, the full-load

power factor being 0·8. The haulage is situated 1,000 yards from the switchboard at the pit bottom, where the voltage is 3,300.

Detail the plant that you would install between the pit bottom switchboard and the motor. It must be possible to run the motor at a creeping speed and at full-load torque for, say, 15 minutes. Take the resistance of 1,000 yards of copper wire, 0·1 square inch section, as 0·258 ohm. (28)

A. The direct haulage gear referred to in the above question would require a slip-ring motor for driving it, and as the voltage of the supply is 3,000, the whole of the apparatus would have to be suitable for high pressure.

Full-load current taken by motor

$$= \frac{120 \times 746}{3000 \times 0{\cdot}85 \text{ (effcy.)} \times 0{\cdot}8 \times \sqrt{3}} = 25{\cdot}3 \text{ amps.}$$

Resistance of transmission $= R$

$$= \frac{\text{Voltage drop}}{\text{Current} \times \sqrt{3}} = \frac{300}{25{\cdot}3 \times \sqrt{3}} = 6{\cdot}8, \text{ say 7 ohms.}$$

$$\text{Sectional area of conductor} = \frac{0{\cdot}1 \times 0{\cdot}258}{7} = 0{\cdot}0037 \text{ sq. in.}$$

This section is too small to carry a current of 25 amperes without heating, and a cable of, say, 0·0225 square inch section should be installed. The class of cable to be installed between the pit bottom and the motor would be 1,000 yards of 0·0225, three-core, paper-insulated, bitumen-sheathed with double-wire armouring. The cable would be protected by means of an automatic circuit-breaker, provided with overload coils, time-lags, and earth-leakage protection, installed at the pit bottom.

At the motor room there would be required another automatic circuit-breaker with isolators, overload coils, time-lags, voltmeter, and ammeter, for receiving the main cable and for controlling the supply to the motor. The starting and controlling of the motor could be accomplished by means of an air-break, flameproof, 3,000-volt controller. The speed control could be obtained by inserting a suitable unbreakable resistance unit in the rotor cirçuit, so designed that a creeping speed at full-load torque of the motor would be obtained without excessive temperature-rise or overheating of the resistance unit. The torque developed by an induction motor is a maximum when the slip is equal to $\frac{\text{rotor resistance}}{\text{rotor reactance}}$. By varying the rotor resistance, the maximum torque may be made to occur at any desired value of slip.

MECHANICAL SECTION

Information for Specification of Motor-driven Centrifugal, or Turbine, Pump

3. Set out the information that should be given to a manufacturer to enable him to offer a centrifugal, or turbine, pump coupled direct to an electric motor for placing at the bottom of a shaft to raise water up the shaft. Deal with both the pump and the motor, but not with switchgear. (28)

A. The following details should be sent to manufacturers to enable them to specify for a pumping plant :—

(1) Plant required. Turbine pumping plant.
(2) Place of erection of plant.
(3) Company.
(4) Purpose for which pump is to be used. Shaft pumping.
(5) Normal discharge in gallons per min. 500
(6) Total head, H, from lowest suction level. 875 feet.
(7) Head, h, from lowest suction level to base-plate. 10 feet.

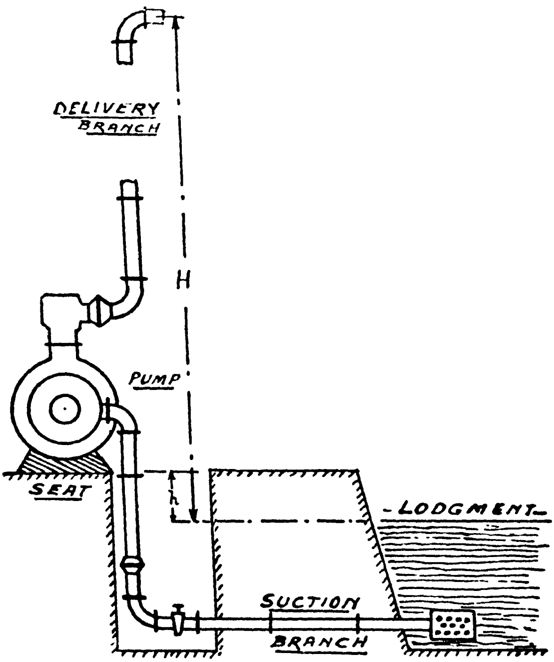


FIG. 132.—Arrangement of Pumping Plant.

(8) Difference between highest and lowest suction levels. 10 feet.

(9) Length, number of bends, and general arrangement of suction and delivery branches. See diagram, Fig. 132.

(10) Details of water to be pumped. Mechanical mixtures; 60° F.; 1·05 sp. gr.; no corrosive action; no chemicals.

(11) Drive and motor. Direct-coupled squirrel-cage motor for 3-phase alternating current, 50 cycles, 1,450 r.p.m., 1,500 volts.

(12) Position of pumping plant. In intake airway; dry and dusty.

(13) Arrangement of plant. See diagram, Fig. 132.

(14) Transport and erection difficulties. None.

Electric Welding

4. Discuss the use of electric welding in connection with machinery used in and about mines, giving instances of parts now commonly made of mild steel welded, that formerly were riveted or were castings. (28)

A. The tendency at the present time is to make use of the very much improved art of welding, and fabricated structures are taking the place of castings and bolted or riveted structures. The result obtained includes greater strength, less liability to work loose and break down, and the absence of projecting parts.

Rivets have been dispensed with in boiler flues, colliery trams, cages, conveyor pans, coal-cutter parts, tanks, casings and barrels for motors, terminal boxes, air mains, steam mains and water mains. Castings have been superseded in the frames of machines and motors, caps and flanges for steel props, and chains are made with welded links.

Fractured parts of machines can be repaired by welding, and shaft journals can be adjusted by welding-on metal and then grinding.

Wire-Rope Constructions

5. Give a list of typical wire-rope constructions and describe them briefly. Indicate the particular purposes for which each construction is most suitable. (28)

A. *Ordinary lay* wire ropes have alternate right- and left-hand twists. The strands in the rope are laid in the opposite direction to the twist of the wires composing the strands. They are mostly used for cranes, tackle, and light loads, and are quite flexible. There is a risk of the wires breaking at the crown of the strands.

Lang's lay wire ropes are uniformly twisted. The wires composing the strands are twisted in the same direction as the strands are laid in the rope. They have a larger wearing surface, and there is much less danger of broken wires in the strands. Ropes of this type are more rigid than ordinary lay ropes. They are greatly used at collieries for hauling and winding purposes.

Flattened-strand ropes are made with one or more flattened surfaces on the strands. They are very flexible, and have a smooth surface exposed to wearing action. Such ropes are sometimes used for winding in shafts.

Galvanised wire ropes are made with wires that are galvanised before being included in the rope ; the core consists also of galvanised wire. They are used in wet winding-shafts, especially where the water has corrosive properties.

Locked-coil wire ropes are specially constructed with interlocking wires, and they resemble a round bar when completed. Owing to their non-rotative action they are greatly used for winding in sinking pits and for cage guide ropes in winding-shafts.

True lay ropes are now being used at collieries for haulage and winding purposes. They are manufactured in the ordinary lay and Lang's lay respectively, as required. The wires forming the strands of true lay ropes do not jump out of position and untwist when cut through for re-capping purposes. See also page 249.

MAY 1934 EXAMINATION

Electrical Section

Electrical Plant for Haulage

1. A supply of three-phase, 50-cycle, 3,000-volt electric power is available from a cable at an underground engine-house. The haulage gear is driven by a 100 H.P., three-phase, 50-cycle, 400-volt reversible motor with wound rotor. Describe the apparatus and switchgear that should be installed for working and controlling the motor and for lighting the engine-house. (28)

A. As the supply cable carries electric power at 3,000 volts, it will be necessary to install a three-phase step-down transformer in order to reduce the 3,000-volt supply to a 400-volt supply for the working of the 100 H.P. haulage motor. The voltage for lighting the engine-house must not exceed 125 volts (alternating current), and it will thus be necessary to make a further reduction in voltage. This could be done by installing a single-phase transformer which would reduce the voltage from 400 to 110 volts. The apparatus and switchgear required are shown diagrammatically in Fig. 133, consisting of an automatic oil-break circuit-breaker, fitted with overload coils and time-lag, for receiving and controlling the 3,000-volt supply to the primary side of the three-phase transformer.

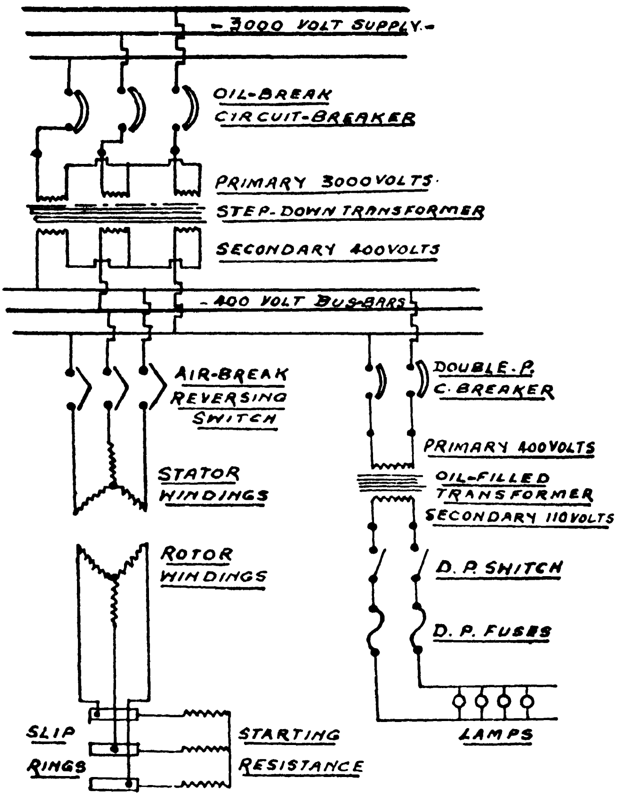


FIG. 133.—Haulage Plant: Diagram of Connections for Apparatus and Switchgear.

By mutual induction the voltage at the secondary terminals of the transformer would be 400, and this supply is connected to the 400-volt bus-bars, which are enclosed in suitable chambers. Mounted on these chambers are two automatic air-break or oil-break circuit-breakers, each fitted with ammeter, voltmeter, overload coils and time-lags, for controlling the supply to the haulage motor and to the single-phase transformer for lighting purposes, as shown diagrammatically in Fig. 134.

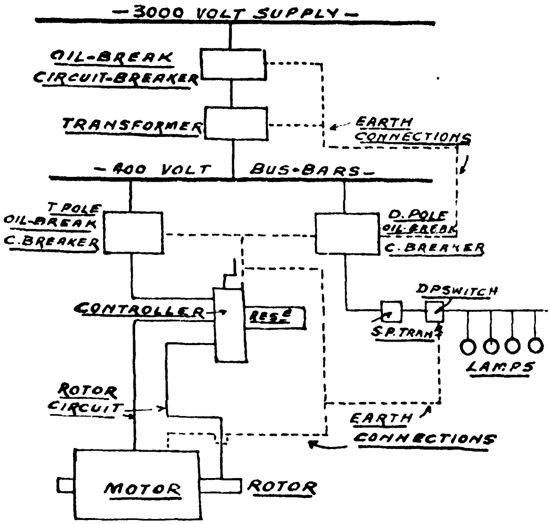


Fig. 134.—Haulage Plant: Diagram of Connections and Earthing Systems.

The main cable from the motor circuit-breaker terminates at a suitable reversing-type controller, which is used for connecting the main supply to the motor stator, and for varying the resistance in the rotor circuit, the rotor and stator being thoroughly insulated from each other. The regulating resistance in the rotor circuit would be of the unbreakable-grid type and would be capable of operating and controlling the speed of the motor without an excessive temperature-rise.

The circuit-breaker controlling the single-phase transformer would be of the double-pole type, connected to the primary side of the transformer. Connection would then be made to the secondary terminals of the transformer, and a supply at 110 volts, through a double-pole switch and fuses, or double-pole automatic circuit-breaker, be obtained for the lighting circuit. Each lamp would be enclosed in a strong fitting, protected by a suitable well-glass and cage. The whole of the apparatus would be earthed to a common earth-bar.

Starters for Induction Motors

2. Three-phase induction motors may be started (*a*) by connecting the power supply direct to the stator through a three-pole switch; (*b*) by using an auto-transformer; (*c*) by using a star-delta

switch ; (*d*) by using a three-pole switch for the stator circuit and a liquid starter for the rotor circuit. Briefly explain each method of starting and say for what purpose each method is suitable. (28)

A. (*a*) This method of starting is suitable only for small motors up to about 5 H.P. The starting current is usually about 4 to 6 times the full-load current, while the torque is from $\frac{1}{2}$ to twice the full-load torque.

(*b*) In the auto-transformer starter, a range of voltage is obtained from the transformer tappings, and this controls the starting current to the motor. This method is suitable for starting large motors, such as those connected with turbine pumps.

(*c*) By using a star-delta starting device the voltage impressed on the motor windings is limited to about 58 per cent. of the line voltage, while at the same time the starting current is also reduced. When the motor attains normal speed, the switch is thrown over to the " running " side, and the motor receives the full line voltage. It is suitable for coal-cutting-machine motors, and endless-rope haulage motors.

(*d*) This method of starting motors is used in conjunction with slip-ring motors. A three-pole switch is used for controlling the supply of current to the stator, while the liquid starter is used for short-circuiting the rotor circuit through the resistance. This method is suitable where a variation in speed is required and also a good starting torque, such as for main rope haulages.

Cable Insulation and Calculation

3. Copper conductors may be insulated either by paper or by bitumen. State which can be worked at the higher current-density, and why. Given that the resistance of 1,000 yards of copper wire, 0·075 sq. inch in section, is 0·32 ohm, what area of conductor would you use to get a 10 per cent. drop in voltage in a three-phase cable one mile long with a current of 100 amperes fed to the cable at 3,300 volts between phases ? (28)

A. Paper-insulated cables can be worked at a higher current-density than bitumen-insulated cables. If the latter are worked at a higher current-density than they are intended for, the bitumen becomes soft by the increased temperature and after a time becomes decentralised. When this occurs, the bitumen no longer retains its property as an insulating material having a high ohmic resistance.

The resistance of a conductor is directly proportional to its length.

Resistance of 1,000 yards of 0·075 sq.-inch conductor $= 0{\cdot}32$ ohm.

$\therefore$ Resistance of 1,760 yards of 0·075 sq.-inch conductor

$$= \frac{0{\cdot}32 \times 1760}{1000} = 0{\cdot}56 \text{ ohm.}$$

Voltage of supply is 3,300 volts, and 10 per cent. of this is 330 volts.

$$\text{Voltage drop } (V) = I \times R \times \sqrt{3}$$

and $$R = \frac{V}{I \times \sqrt{3}} = \frac{330}{100 \times \sqrt{3}} = 1{\cdot}90 \text{ ohms.}$$

If the resistance of 1 mile of 0·075 sq.-inch conductor is 0·56 ohm, the sectional area of a conductor of the same length with resistance of 1·9 ohms will be less than 0·075 sq. inch, the resistance of a conductor being inversely proportional to its cross-sectional area.

$$\therefore \text{ Area of conductor} = 0{\cdot}075 \times \frac{0{\cdot}56}{1{\cdot}9} = \underline{0{\cdot}022 \text{ sq. in.}}$$

MECHANICAL SECTION

Adjustment of Cage Position

4. The length of steel-wire winding-ropes changes, owing to stretch and to cutting when re-capping, making it necessary to adjust the cage positions. Describe methods for doing this. (28)

A. When light drums are used for winding from shallow pits, a number of wooden strips might be nailed to the wooden tread of the winding-drum so as to increase its diameter. This would allow for the stretch of the winding-rope. Sufficient rope laps are usually contained on the drum, when the cage is at the shaft bottom, to allow for cutting the rope when re-capping.

In deep pits, where heavy winding-drums are in use, the adjustments referred to above are usually carried out by fitting cage chains of correct length to suit the markings on the drum cheek, and the depth indicator markings. Various lengths of cage chains are kept in readiness at the colliery workshops for this purpose.

If the stretch of the rope is not enough to warrant the above adjustment, new markings might be made on the cheek of the drum, as a guide for the engineman in decking the cages. A slight adjustment might also be required in the overwinding gear.

Steam-raising Plant

5. Describe a modern steam-raising plant to provide superheated steam at high pressure, and at the rate of, say, 40,000 lb. per hour. (38)

A. A modern steam-raising plant to provide superheated steam at high pressure at the rate of 40,000 lb. per hour might consist of three water-tube boilers of the Stirling or Babcock & Wilcox pattern, as already described. See pages 242–3 and 254–5 and Figs. 113 and 118.

The auxiliary plant might consist of the following :—

(*a*) A water-softening plant to remove some of the chemicals in solution in the feed-water, and thus render frequent cleaning of the boiler tubes unnecessary.

(*b*) A feed-water heater, using exhaust steam, to increase the temperature of the feed-water, and thus avoid rapid changes in temperature inside the boiler.

(*c*) Air pre-heaters for each boiler, using the heat from the waste gases of the boiler, to heat the feed air and thus increase the efficiency of the plant.

(*d*) Mechanical stokers and forced draught for each boiler, to give better combustion and increased efficiency. See Figs. 129 and 130.

(*e*) Liptak suspended-arch furnaces for each boiler to reduce the cost of repairs. See pages 273–4 and Fig. 128.

Injector

6. Describe the action of an injector for feeding water to a boiler. Illustrate your answer by a diagram or sketch. (28)

A. Fig. 135 shows the sectional arrangement of a Davis-Metcalfe Exhaust Steam Injector. Exhaust steam enters the injector through a wing valve at the top, and it is augmented by a supply of live steam before passing to the exhaust steam cone. Supplementary live steam enters the injector at the side and passes to the live steam nozzle inside the exhaust steam cone.

The steam issuing from the main cone meets with the feed-water in the draught cone and is condensed by it, imparting its velocity to the water. The velocity given to the feed-water is converted to

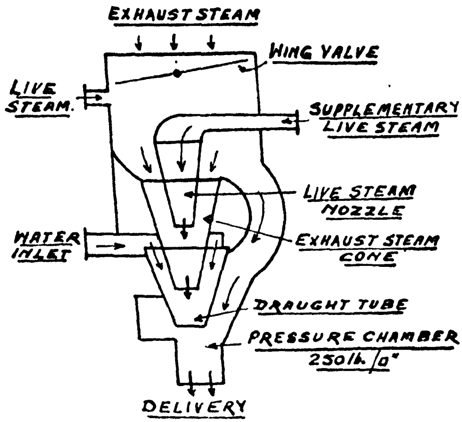


FIG. 135.—Davis-Metcalfe Exhaust Steam Injector.

pressure energy at 250 lb. per sq. inch in the pressure chamber to enable the water to enter the boiler.

Pins in Double Shear

7. Make a sketch to illustrate a pin or rivet stressed in double shear. Given that the maximum permissible stress in double shear for an iron pin is 7 tons per sq. inch, of what diameter would you make a pin to carry a load of 6 tons? (28)

A. Fig. 136 is a sketch illustrating a pin in double shear.
Let f_s = maximum shear stress = 7 tons per sq. inch.

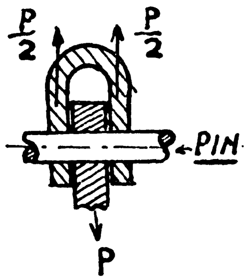


FIG. 136.—Pin in Double Shear.

Then $$P = f_s \times \frac{\pi}{4}d^2$$

$$6 = 7 \times \frac{11}{14} d^2$$

and $$\underline{d = \sqrt{\tfrac{6}{7} \times \tfrac{14}{11}}} = \underline{1{\cdot}05 \text{ or } 1\tfrac{1}{16}}$$ inches diameter.

Centrifugal or Turbine Pumps

8. In connection with multi-stage centrifugal or turbine pumps :—

(*a*) What is the highest suction head that you would arrange for ordinary cold water ?

(*b*) Describe the operation of starting the pump and also of stopping it.

(*c*) What are the points to watch in keeping the pump running well ?

(*d*) What parts of the pump are the more liable to wear ?

(*e*) If the driving electric motor is overloaded, how can the load on the pump be lessened at constant speed ? (28)

A. (*a*) The suction head depends on the water temperature It should not be more than 15 to 18 feet for water temperatures under 100° F.

(*b*) Before starting the pump, the suction leg and the pump casing should be primed by using water from the rising column, and at the same time opening the priming cocks to allow air to escape. When water flows from the cocks they are closed. With delivery closed and suction open, the motor is started gradually by means of the starter, after which the gate valve in the rising column is slowly opened. If the plant is in good order the pump should then function properly.

The pump is stopped, after closing the gate valve in the rising column, by pressing the automatic release button, turning the resistance back to zero, and putting the starter to the "off" position.

(*c*) To keep the pump running well, the foot-valve and strainer must be operating properly to give an adequate suction flow of water. The delivery column must be free and not obstructed by a faulty reflux valve. The pump and motor should be properly lubricated. The balancing disc should be operating properly, and the stuffing-boxes effective to prevent the pump running on air. The pressure-gauge showing head of water should be under constant observation.

(*d*) The parts of the pump most liable to wear are the bushes and linings in contact with moving parts, such as the bushes at the suction side of the impellers and at the periphery of the impellers. The volute passages of the pump casing, and also the impellers, are liable to wear quickly if the water contains sand or grit.

(*e*) If the motor is overloaded, the opening in the gate valve on the delivery column can be reduced by partly closing the valve. This will reduce the volume of water passing and reduce the load on the motor. See page 256.

NOVEMBER 1934 EXAMINATION

Electrical Section

Connections for Three-Phase Motor

1. Draw a diagram of connections showing an electrical three-phase circuit from a three-core cable through high-tension switchgear, step-down transformer, medium-tension switchgear, and stator reversing switch to the motor. The reversing switch is combined with a liquid starter for the rotor circuit. Show automatic overload trips and low-voltage releases where appropriate. Show ammeter and voltmeter on the medium-tension (step-down) side of the transformer. (28)

A. Fig. 137 shows the diagram required by the above question.

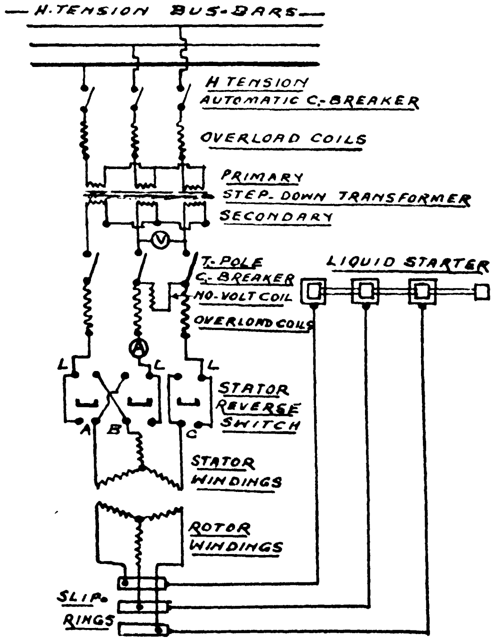


Fig. 137.—Diagram of Connections for Apparatus and Motor.

Supply of Electric Power to a Moving Coal-cutter

2. Discuss the supply of electric power to a moving coal-cutter. Deal with the questions of switchgear, trailing cable and protection from danger to men and from damage to plant. (28)

A. In connection with the above installation the following details should receive attention :—

The switchgear should be of the air- or oil-break type and flameproof. The air-break switch is the better and safer of the two. If the oil level is not maintained, cracking of the oil results, and internal explosions may occur.

The trailing cable should be of the five-core C.T.S. type with copper braiding (see Fig. 153, p. 313).

The protecting apparatus for the plant should be incorporated in the gate-end box, and consist of automatic trip coils with time-lags, and a system of earth-circuit protection.

A systematic inspection, and frequent testing of apparatus, is necessary for the protection of the men, in addition to the equipment already mentioned.

The supply of electric power to a moving coal-cutter would be at medium pressure. The main supply cable would be three-core for three-phase alternating current, insulated with vulcanised bitumen and protected by a single or double layer of wire armouring (Fig. 152, pp. 312–13). The cable would be connected to a suitable circuit-breaker at the main distribution centre. Incorporated in the circuit-breaker would be a core-balance leakage transformer, so that in the event of a short-circuit between the conductors, or an earth fault in any one conductor, the circuit is immediately made dead.

At the point where the flexible trailing cable is joined to the main cable, an automatic air-break gate-end circuit-breaker should be installed ; and incorporated in the gate-end box should be the " Williams Rowley " system of earth-circuit protection to ensure that the coal-cutter is not operated unless properly earthed.

The trailing cable should be of the C.T.S. type, as already stated, with copper braiding and having three current cores, one pilot core, and an earth conductor. At each end of the cable, properly constructed connectors should be fixed, or B.S.I. plugs. A regular systematic inspection of all apparatus, and frequent testing of the same, often prevents damage to plant, and at the same time reduces the risk of fire and shock to a minimum.

Mechanical Section

Specification of Headgear Pulley

3. An ordinary headgear pulley with shaft and bearings is wanted for carrying a 1-inch diameter steel-wire winding-rope. Draw up a specification to be sent to the makers when inviting tenders. Assume your own conditions and include the chief dimensions and particulars. Do not deal with the headgear or the rope. (28)

A. Specification for a Winding Pulley :—

Pulley.—The pulley to be 10 feet diameter on the tread and suitable for a single turn of 1-inch diameter steel-wire rope ; the horns to be 6 inches long. The rim and boss of the pulley are to be made of cast steel, the rim being cast in four segments, and the boss in halves 13 inches long. The joints of the boss and rim are to be machined with a check at each joint, and fitted with two 1-inch diameter bolts having jamb nuts at each end. Weldless mild-steel hoops, bored out, are to be shrunk on machined bosses at each joint of the rim and also round the boss. The rim must be turned in the groove, and the boss bored out and key-seated to suit the shaft. The light arms are to be 3 by 1 inch mild steel of 28 to 30 tons per sq. inch tensile strength, accurately fitted into and bearing solid at the semicircular ends of machined recesses at the rim and boss, the depth of the recesses to be half the thickness of the arm. The arms are to be bolted at each end with two 1-inch diameter bolts having jamb nuts at each end. The above details are sketched out and illustrated in Fig. 138.

Shaft.—The shaft must be made of forged steel, 8 inches diameter, machined all over, with a collar at each end of the seating, so that the pulley cannot slide on the shaft. The shaft must have a feather, or key, fitted to suit the key seat in the pulley. Journals $6\frac{1}{2}$ by 10 inches are to be arranged at each end of the shaft, and each journal must have two spiral oil grooves cut from the inner face of the collar to within 1 inch of the end of the bearing length.

Bearings.—The bearings are to be of cast iron of massive design, with cast-iron covers and bottom-half semicircular gunmetal bushes made of 86 per cent. copper and 14 per cent. tin. Two heavy wrought-iron oiling rings, made in halves, securely bolted at the joints, and having holes round the periphery, are to be fitted in each bearing. Provision must be made for running-off the oil from the oil well when required.

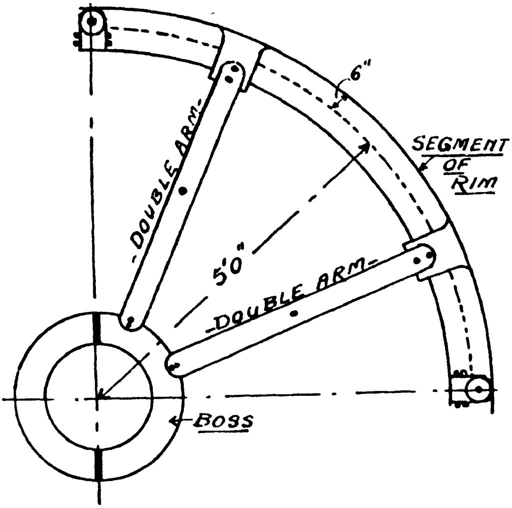


FIG. 138.—Headgear Pulley for Winding.

Sole-plates.—The sole-plates are to be of cast iron, 3½ inches thick, machined on the seating for the bearing, and on the bottom to sit on the headgear beams. Bearing bolts only to be supplied.

Generally.—The whole construction to be a first-class job, and when ready for delivery an inspection of it will be carried out by a competent person representing the Colliery Company.

Three-Deck Cage

4. Describe the construction of a three-deck cage and chains to carry men, or six tubs each to hold 10 cwt. of coal. Give the chief dimensions and state the materials used. An outline drawing of the whole cage and chains, or a detail drawing of a part, should be included. (28)

A. Fig. 139 shows end and side elevations of a winding-cage as required by the above question, suitable sizes and dimensions being included in the drawing.

Size of cage chains for a load of 3 tons cage + 3 tons coals + 1½ ton tubs :—

$d = \sqrt{9W}$, where d = diameter in sixteenths of an inch and W = breaking load.

Four chains taking the load: $d = \sqrt{\dfrac{7 \cdot 5 \times 10}{4} \times 9} = \frac{13}{16}$ or $\frac{7}{8}$ inch.

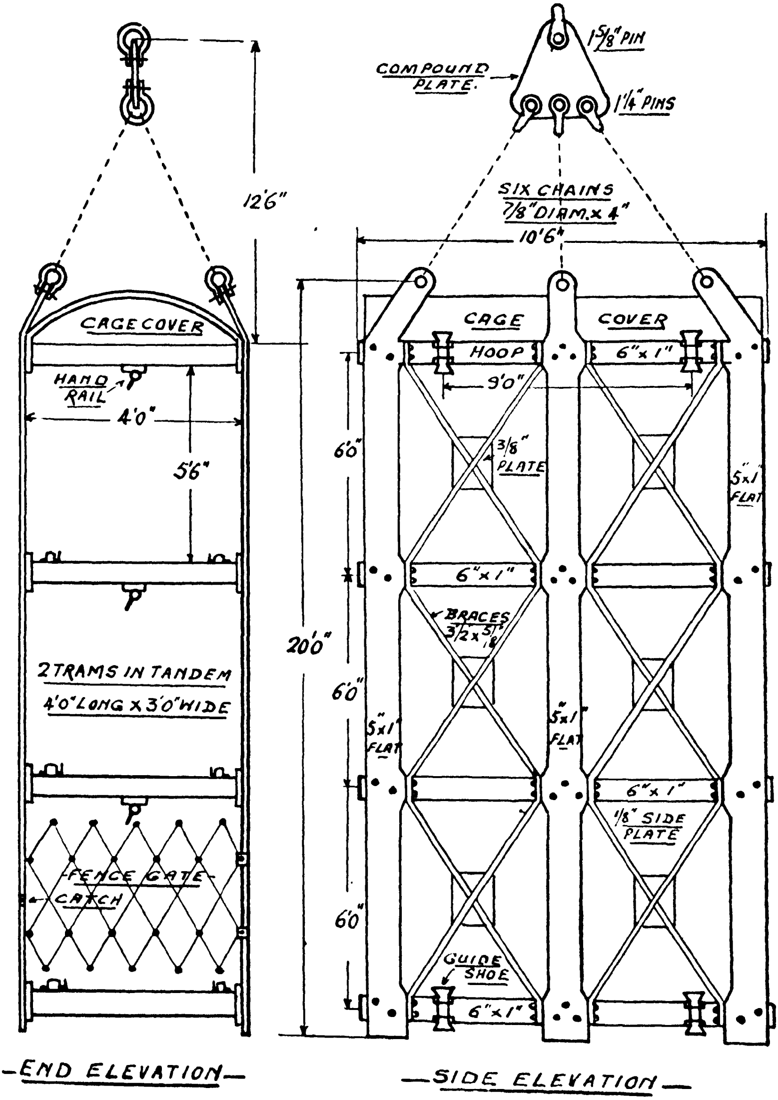


FIG. 139.—Construction of Three-deck Winding-cage.

Two small pins on compound plate taking the load :—

$$d = \sqrt{\frac{7{\cdot}5 \times 10}{2} \times 9} = \tfrac{19}{16} \text{ or } 1\tfrac{1}{4} \text{ inches.}$$

One large pin taking the load: $d = \sqrt{7{\cdot}5 \times 10 \times 9} = \frac{26}{16}$ or $1\frac{5}{8}$ inches.

Four shackle-pins on cage taking the load :—

$$d = \sqrt{\frac{7{\cdot}5 \times 10}{4} \times 9} = \tfrac{13}{16} \text{ or } \tfrac{7}{8} \text{ inch.}$$

Materials of construction :

Cage chains, shackles, pins and hooks.—Wrought iron.
Hoops of cage.—Best weldable mild steel.
Hangers and ties of cage.—Best mild or forged steel.
Compound plate.—Forged mild steel.

Butt Joint for Lancashire Boiler

5. Show by part plan and sectional cross-elevation a butt joint for the longitudinal seam of a Lancashire boiler shell with inside and outside cover-plates and four rows of rivets, the shell plate being $\frac{3}{4}$-inch thick. The working steam pressure is 150 lb. per sq. inch. Show the dimensions of the cover-plates and the size and spacing of the rivets. Draw to a scale of a quarter full size. Rivet holes can be indicated in plan by crosses. The ends of the cover-plates where the shell rings overlap need not be shown. (38)

A. Fig. 140 shows the drawings required by the above question.

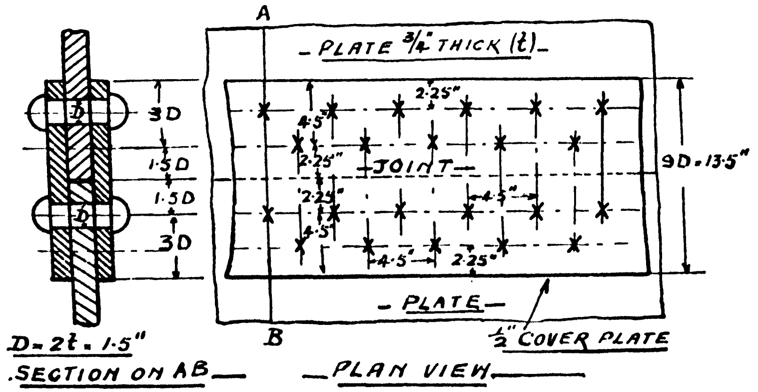


FIG. 140.—Butt Joint for Longitudinal Seam of a Lancashire Boiler.

Steam Turbine and Condenser

6. Describe the construction and working of one type of steam turbine and condenser, say for an output of 1,500 H.P., using steam at about 250 lb. per sq. inch at an inlet temperature of about 600° F. (28)

A. Fig. 141 shows diagrammatically the arrangement of a high-pressure condensing turbine. The high-pressure part of the turbine is of the impulse type, with two or more stages. The low-pressure part is of the reaction type, with 6 to 10 stages. The

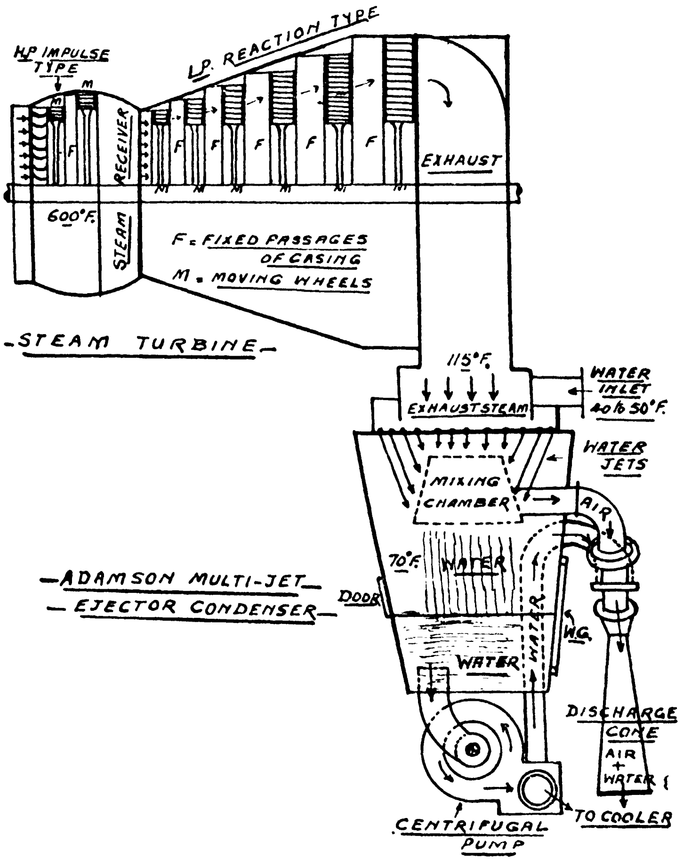


FIG. 141.—Steam Turbine and Condenser.

exhaust steam passing to the condenser is assumed to be at 115° F. and the temperature inside the condenser is 70° F.

In the high-pressure part of the turbine, the steam from nozzles impinges on the blades of the moving wheel of the turbine, and then escapes into fixed passages of the casing, which guide it into the blades of the next moving wheel, and so on, until it finally arrives at a receiver situated between the impulse and reaction parts. In the latter part of the turbine, the wheels are driven round by the reactive properties of the steam as it flows from one to another. The steam pressure is gradually reduced, and with the excessive volume produced thereby, the velocity of the steam is maintained to give the necessary movement of the wheels.

The condenser sketched out in the diagram is known as the "Adamson Multi-jet Ejector Condenser." Water and steam are introduced at the top of the condenser, and the steam is condensed, by sprays of water from nozzles, in the mixing cone. The water is drawn off by a centrifugal pump at the base of the condenser, part going to a cooling arrangement, or to waste, and the remainder passing to a discharge cone for trapping air from the condenser and producing the necessary vacuum. With a steam consumption of 18 to 26 lb. per kW. hour, and a vacuum of 28·5 inches of mercury, 40 to 60 lb. of water would be required in the condenser per pound of steam condensed.

Centrifugal or Turbine Pump—Characteristic Curves

7. Describe the action of a centrifugal, or turbine, multi-stage pump. Draw typical curves connecting the head, rate of flow, power, and efficiency for such a pump. (28)

A. The fundamental principle in the action of centrifugal and turbine pumps is based on Bernouilli's Theorem, which states that the energy in a liquid flowing through a pipe is a constant if friction be neglected, or the pressure energy plus the kinetic energy contained in each pound of water is a constant at any point in the line of flow.

Let V = Final velocity of the water in the pump (high).
V_1 = ,, ,, ,, ,, ,, ,, pressure chamber (low).
p = ,, pressure ,, ,, ,, ,, pump (low).
p_1 = ,, ,, ,, ,, ,, ,, pressure chamber (high).

w = Weight of a column of water one foot head by one square inch = 0·434 lb.
h = Head of water.

Then Pressure energy + kinetic energy = constant,

i.e. $$\frac{p}{w} + \frac{V^2}{2g} = \frac{p_1}{w} + \frac{V_1^2}{2g} = \text{constant};$$

$$\therefore \quad \frac{V^2 - V_1^2}{2g} = \frac{p_1 - p}{w};$$

but as $$\frac{V^2 - V_1^2}{2g} = h$$

$$\therefore \quad h = \frac{p_1 - p}{w}.$$

The head therefore depends upon the velocity given to the water by the rotary action of the pump impellers or wheels.

The foregoing principle is applied practically in the case of the turbine pump as follows :—

(i) Water flows into the pump under atmospheric pressure at a low velocity.

(ii) The velocity is greatly increased by the action of the pump impellers.

(iii) A normal velocity is again produced by means of diffusers having expanding passages and guide blades surrounding each impeller, and finally in a pressure-chamber, so that velocity head is converted to pressure head.

(iv) The impeller vanes are properly designed to reduce shock when the water is entering and leaving the impellers.

(v) Friction varies as V^3, and there is thus a limit to the velocity to obtain a good efficiency. High heads are obtained by keeping the size of the impellers constant, and by placing a number of impellers in series upon a common shaft.

(vi) The impellers are narrower at the tips than at the centre to ensure a constant flow of water through the pump.

(vii) The expanding passages in the pump casing are carefully designed so that no sudden change takes place in either the direction or the speed of the water before it passes to the delivery column.

The following results of a test made with a Sulzer turbine pump, having three impellers $11\frac{3}{4}$ inches diameter, driven at a steady speed of 1,625 revs. per min., can be used to plot the characteristic curves referred to in the above question.

THREE-STAGE TURBINE PUMP TEST AT CONSTANT SPEED AND VARYING HEAD.

Pumping Head.			Revs. per min.	Gallons of Water delivered per min.	Input to Motor (Watts).	Output in Watts : $G \times Head \times 0{\cdot}226$	Brake Horse-power.	Efficiency per cent.
Suction.	Delivery.	Total.						
Feet.	Feet.	Feet.						
5	0	5	1,625	127·3	4,335	142·4	0·19	3·5
5	27	32	,,	120·6	4,166	884·5	1·19	22·1
5	40	45	,,	114·0	4,100	1,166·5	1·56	29·8
5	60	65	,,	102·2	4,000	1,450·0	1·94	38·6
5	76	81	,,	90·4	3,850	1,576·0	2·11	43·0
5	90	95	,,	70·3	3,404	1,481·4	1·99	45·2
5	100	105	,,	57·5	3,187	1,336·0	1·80	44·0

Fig. 142 shows the graphs or characteristic curves of head, efficiency and brake horse-power plotted on a gallons basis. The limits of application of the pump are indicated on the graph.

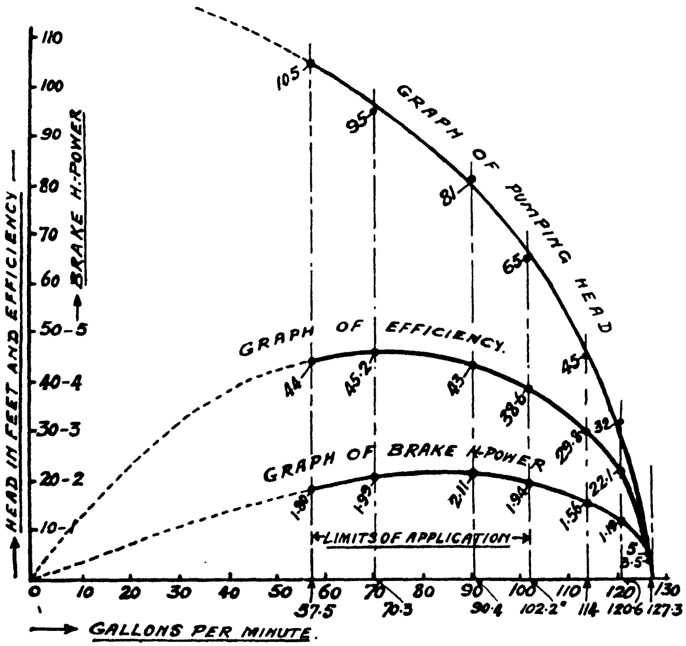


FIG. 142.—Characteristic Curves for Sulzer Three-stage Turbine Pump.

JULY 1940 EXAMINATION

(*Five questions only to be answered.*)

ELECTRICAL SECTION

Mining Type Transformer

1. Describe a transformer, suitable for underground use where there is no firedamp, for obtaining three-phase current at 400 volts between phases from a 3000-volt supply. What is meant when it is said that the windings are delta-connected on the primary side and star-connected on the secondary side? (28)

A. The modern tendency in mining operations is to carry high-tension current as near to the working face as possible, and to install step-down transformers at a safe distance from the actual workings. By doing this the voltage at the various points can be kept up to a higher level, and consequently the whole electrical plant can be worked much more efficiently than by the older system of having a transformer stationed at the shaft bottom and feeding long lines of cables to points inbye at a much lower voltage. To enable transformers to be mobile underground, some manufacturers now specialise in "mining type" transformers, designing them to suit the height and width of the road, and also fixing them on wheels so that they can be transported on the pit-tub track. These transformers can be obtained with fittings for incoming and outgoing cables only, or with incoming control-switch, or again fitted with both incoming and outgoing control-switches, these switches in turn being fitted with overload release trips and earth-leakage trips.

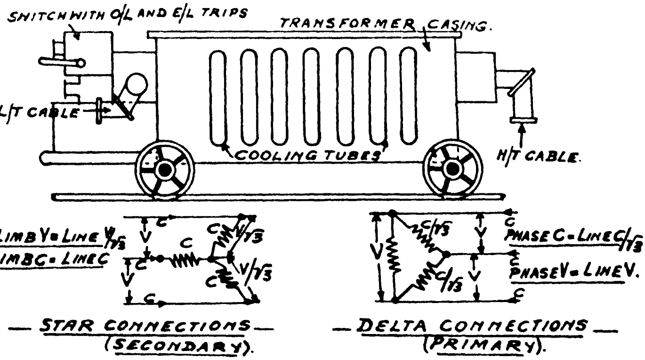


FIG. 143.—Mining Type Transformer.

Fig. 143 shows the general arrangement of a mining type transformer fitted with control-switch and trips on the low-tension side. The Coal Mines Act requires that the L.T. side of the transformer must not be accidentally charged with primary voltage. The neutral point of the three-phase system must therefore be earthed. The transformer cores are fitted inside a casing containing oil for cooling purposes. A fixed oil level is necessary, and there is often an auxiliary upper tank for replenishing the supply when required. The delta connections on the primary windings and the star connections on the secondary windings are shown in Fig. 143.

Starters for Three-Phase Squirrel-cage Motors

2. When a three-phase squirrel-cage motor is connected direct to the power supply, the starting current may be over six times the full-load running current. What devices may be adopted to reduce the demand on the system for current at starting under various conditions ? How do the devices act ? (28)

A. The star-delta starter for squirrel-cage motors of small and medium size—such as those used in mines for coal-cutting machines, endless-rope haulages, auxiliary ventilating fans and turbine pumps—has been described in answering similar questions. See pages 258–9 and 284 and Figs. 119 and 120.

The high-torque rotor with a double set of short-circuited bars for increasing the starting torque and reducing the starting current with squirrel-cage motors has been described on page 251.

When large squirrel-cage motors are in use for hauling and pumping, and where it is necessary to start up gradually under heavy load, the auto-transformer starter is very suitable, as the voltage at starting can be controlled. Fig. 144 shows in diagrammatic fashion the arrangement of such a starter. Each phase is fitted with a transformer, and these are tapped at various points to give the desired voltage, say 1, 2 and 3 in the diagram. When

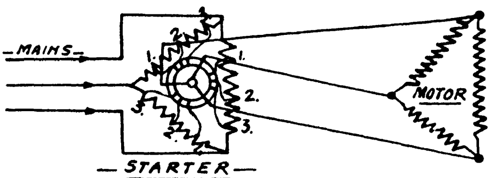


FIG. 144.—Auto-Transformer Starter.

the motor has attained its full speed, the starter is switched direct on to the line, and the transformer is cut out.

A mechanical method of starting squirrel-cage motors is known as the *friction clutch method.* The clutch is similar to that used to control endless haulages in mines. The motor is started up by direct switching from the line, but the load is applied in a gradual way. The friction clutch coupling on the motor shaft allows this to take place when the operating lever is moved from the " off " to the " on " position.

Three-Phase Induction Motor

3. Describe, both mechanically and electrically, a three-phase induction motor with wound rotor suitable for medium tension current. If using 50-cycle current, how is a speed of about 730 revs. per min. (750 synchronons) obtained ? What is the purpose of a wound rotor as distinct from a squirrel-cage rotor ? (28)

A. A three-phase induction motor with a wound rotor is made up mechanically of a mild steel or nickel-steel shaft, one end of which is fitted with slip-rings and the other end with pinion wheel, or coupling, for the load. A cast-iron casing, made cylindrical with two sides, encloses the electrical equipment of the motor, and such a casing might be explosion-proof. It is mounted on a cast-iron or built-up mild steel bed-plate. Fig. 145 shows the general arrangement of the motor.

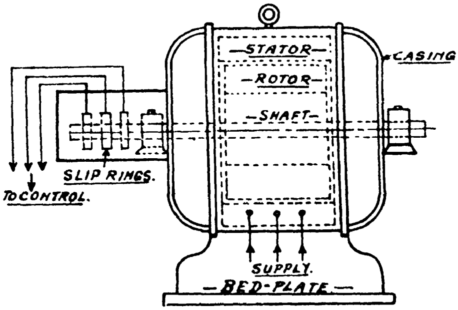


FIG. 145.—Induction Motor.

Electrically the motor has two separate copper circuits, which are known as the primary or " stator," and the secondary or " rotor." The conductors of the primary circuit are disposed in slots in the inner periphery of a hollow laminated iron cylinder forming the stator. The conductors of the secondary circuit are disposed in slots on the outer periphery of a laminated iron cylinder fixed on the motor shaft, so that it is capable of rotating. Three slip-rings, which are insulated from one another and also from the shaft, are connected to the secondary windings at equidistant points. This part is termed the rotor.

The stator windings are connected to the supply by a three-pole direct-starting switch, and a rotating magnetic field is produced within it. A current is induced in the rotor windings by the rotating magnetic field of the stator. The action between the induced current and the rotating magnetic field gives rise to the rotation of the rotor. For speed regulation within wide limits a liquid variable resistance is connected with the rotor circuit. See also pages 236–7.

To determine the speed of an induction motor when the frequency is known, the governing factor is the number of pairs of poles wound in the stator circuit. As there must be both positive and negative poles to obtain a magnetic circuit in the path of the rotor, the poles are taken in pairs.

Let x = number of pairs of poles for a synchronous speed of 750 revs. per min.

$$\text{Then Motor Speed in r.p.m.} = \frac{\text{Frequency or cycles per second}}{\text{Pairs of poles (or } x)}$$

$$\therefore\ 750 = \frac{50 \times 60}{x}$$

$$\text{and } x = \frac{50 \times 60}{750} = \underline{4 \text{ pairs of poles.}}$$

The purpose of the wound rotor is for use with high-power plant, such as direct haulages and winding, where the load is heavy and gradual starting is necessary. A wide range of speed is often necessary in winding.

Mechanical Section

Water-Tube Boilers

4. What distinguishes water-tube boilers of the Babcock type from those of Stirling type? Illustrate your answer by simple sketches. (28)

A. Fig. 118 (p. 254) shows a sectional side elevation of a Babcock & Wilcox water-tube boiler. This boiler has two steam and water drums connected at the ends by water tubes. The feed-water from the back part of the drums passes down headers to banks of inclined tubes which in turn are connected by headers also to the front part of the drums. Steam is produced in the banks of tubes, directly over the fire, just before their junction with the front headers.

Fig. 113 (p. 242) shows a sectional side elevation of a Stirling water-tube boiler. Such a boiler has three steam and water drums and two mud drums. The feed-water circulates from the back drum to the centre one and finally to the front drum by means of banks of water tubes. Steam is drawn from the front and centre drums to a superheater before passing to the steam main.

Winding-Engine Controller and Slow-Banking Arrangement

5. Describe an apparatus suitable for preventing overwinds by a steam-driven winding engine and for preventing over-speed during the wind, especially when the down-going cage is approaching the bottom landing, carrying men. (28)

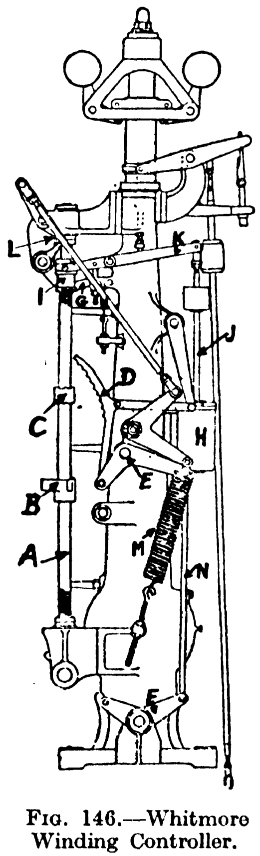


FIG. 146.—Whitmore Winding Controller.

A. Fig. 146 shows the *Whitmore overwind and overspeed controller*. The overwinder consists of two long screws *A*, representing the wind of each of the winding cages. Each screw carries two index nuts *B* and *C*, the lower nut *B*, forming the "first attachment," being provided with two arms and a tooth to engage with speed hooks *D*, which are notched and curved to a special sweep, and directly controlled by a governor. The point at which the nut *B* engages with the hook depends on the amount the latter is pushed forward by the action of the governor. This attachment safeguards the speed in the shaft and therefore ensures retardation at the proper point in the shaft. The notched speed hooks are fitted on a lever keyed on a spindle *E*, which, through a system of levers and trips, pulls the brake gear slowly on, say in 3 or 4 revolutions of the drum, after having first of all tripped and closed the throttle-valve by the lever *F*.

The nut *C*, forming the "second attachment," when the cages are nearing the surface, engages with levers *G*, lifting the same in the continuation of travel and permitting a dash-pot piston working in cylinder *H*, and attached to the rod *J*, to descend at a desired and diminishing rate of fall. The piston is connected through the rod *J* and lever *K* to the sliding collar *I*. In the event of the nut *C*, after engaging with lever *G*, having a higher speed upward than the piston has downward, it will engage with sliding collar *I* and become locked with it, and will thus revolve with the screwed spindle *A*. This action releases the trip *L*, trips the throttle-valve, and applies the brakes by means of spring *M*, rod *N*, and bell-crank lever *F*. The same trip operations would take place in the case of an overwind.

A series of catches, known as the "third attachment," comes into operation during the final 1½ revolutions of the drum at the end of the wind for both coal and men. A spring-loaded trigger is released in the event of overspeed, and the steam brake is applied.

A specially designed spring-loaded trigger forms the "fourth attachment," and this is operated if the engine is started in the wrong direction. When the trigger is released the steam brake is applied.

Slow - Banking Arrangement.—The new regulations of 2nd March, 1937, state that the automatic contrivance to prevent overwinding must also prevent the descending cage from being landed at the pit bottom, or other permanent landing, at a speed exceeding 5 ft. per second, when men are riding. To fulfil these conditions, some type of slow-banking arrangement must be incorporated in overwinders, and this is accomplished by an alteration in the third and fourth attachments already referred to.

Fig. 147 shows the details of the slow-banking arrangement of the Whitmore Controller. The overwinder nuts *B*, during their final 1½ inches of travel, operate on the free end of a specially designed lever *SL*. The lever is made to have a variable fulcrum by the rolling-over of a cam *SC*, thereby lifting the dash-pot piston in the cylinder *H*. If the dash-pot is lifted in excess of the predetermined velocity, the piston and rod *J* are lifted by the fluid in the dash-pot. An arm on rod *J* operates a mercury switch which actuates an electric buzzer and warns the engineman of overspeed. This warning is continuous until the speed is again normal. A spring-loaded trip gear has now been fitted instead of the "third attachment" to prevent increase of speed at, say, 5 feet from the end of the wind. A collar on rod *J* operates a bell-crank lever when the rod lifts. The other end of the bell-crank lever supports

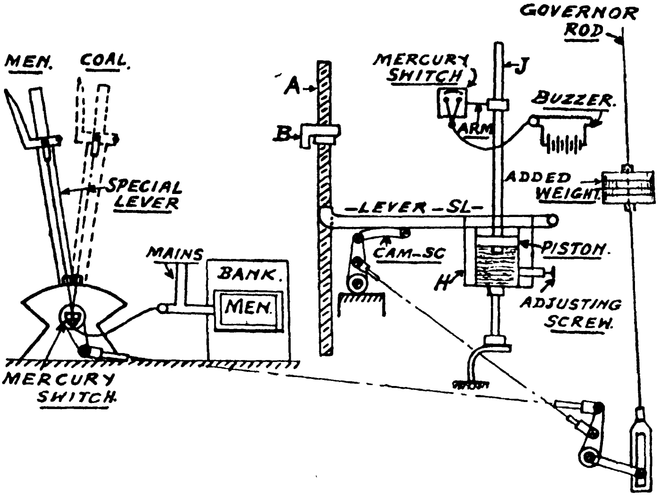


FIG. 147.—Whitmore Slow-Banking Arrangement.

a spring-loaded hammer on a catch. If the bell-crank lever is moved sufficiently, the hammer is released and, striking the lever *K*, operates the trip.

To differentiate between the speeds of winding for coal and men respectively, the cam is lowered at one end so as not to interfere with normal coal-winding speed. This is accomplished by fitting a special lever on the driver's platform, as shown in Fig. 147, and putting the lever into the forward or "Coal" position. The action of bringing the lever to the "Men" position rocks a mercury switch on the end of the shaft to a closed position, and a panel showing "Men" is illuminated at the pit-head. At the same time the cam *SC* takes up a special position under the operating lever *SL* to regulate the winding speed at the end of the wind. The governor rod is also loaded with additional weights to ensure that the maximum speed in mid-shaft is less for men than for coals. The apparatus is easily adjusted to any deceleration required by means of the adjusting screw on the dash-pot, or by varying the viscosity of the oil.

Winding-Ropes—Treatment when in Use

6. Discuss the choice and use of a rope for colliery winding plant. What working conditions would you regard as helpful to the rope, and what conditions would operate to shorten the life of the rope? (28)

A. This question has been answered when dealing with similar topics ; see pages 249–50 and 280–1. In addition to the daily and weekly examinations carried out in connection with winding-ropes, it is very important to have periodical examinations of the rope for wear. This can be carried out successfully by cleaning the surface of the rope and using proper gauges.

Drysdale Snorer Turbine Pump for Headings

7. Describe a pump suitable for draining the advancing end of a dipping incline, so as to keep it as dry as possible for the workmen. Water collects at the rate of about 150 gallons per min., and has to be raised a maximum of 100 feet to a sump higher up the incline. Deal with both the driving and water-handling parts of the pump, but not with any auxiliary plant. Either compressed air or electric power is available. (28)

A. Assuming that normal conditions exist in the heading, and that the water is not particularly gritty, a pump of the Drysdale Snorer type could be used. Such a pump would be light in weight, of small dimensions, and easily mounted on a bogie to advance with the working face. This type of turbine pump will run on air without stopping, and would have decided advantages over other types under the given conditions.

To the back end of the pump is attached a separate chamber containing a valve-plate and air rotor, both made of stainless steel, the latter being keyed on the pump shaft. The seal between this air-extraction chamber and the pump is maintained by means of an intermediate bush which is lubricated. The air rotor revolves in its circular casing in which it is eccentrically fitted. The casing is partly filled with water which revolves with the rotor but follows the outer edge of the casing, due to the centrifugal force imparted to it by the rotor. Once in each revolution the water alternately recedes from and re-enters the rotor, and, acting like a piston, compresses the air and expels it from the chamber. The air rotor revolves with a normal clearance of four-thousandths of an inch from the valve-plate, and adjustment is necessary to take up wear, which becomes excessive with gritty water.

Chief dimensions of pump required.—Allowing for stopping of pump and for friction, a quantity of 175 gallons per min. against a head of 120 feet should be provided for.

Area of pump outlet (allowing discharge velocity of $5\frac{1}{2}$ ft. per sec.)

$$= \frac{175}{5\frac{1}{4} \times 60} \times \frac{144}{5\frac{1}{2}}$$

$$= 14\tfrac{1}{2} \text{ sq. inches}$$

$$\text{Diameter of pump outlet} = \sqrt{\frac{14{\cdot}5 \times 14}{11}} = 4\tfrac{1}{4} \text{ inches}$$

$$\text{Diameter of pump impellers} = 8\tfrac{1}{2} \text{ inches}$$

$$\text{Width of impellers} = \frac{8\frac{1}{2}}{20} = 0{\cdot}4 \text{ inch}$$

Speed of pump is 1440 revs. per min. with 4 poles and 50-cycle current.

Now
$$\text{r.p.m.} = \frac{\sqrt{\frac{v^2}{n}} \times 60}{\text{Circ. of Impeller}}$$

$$1440 = \frac{\sqrt{\frac{64 \times 120}{n}} \times 60}{\frac{8}{12} \times \frac{22}{7}} \qquad (v^2 = 2gh)$$

$$\therefore\ 1440 \times \sqrt{n} \times 8 \times 22 = 8 \times \sqrt{120} \times 60 \times 84$$

$$\text{and } \sqrt{n} = 1{\cdot}75$$

$$\therefore\ n = 3 \text{ stages.}$$

$$\text{B.H.P. of squirrel-cage motor} = \frac{175 \times 10 \text{ lbs.} \times 120 \text{ ft.}}{33{,}000} \times \frac{100}{75}$$

$$= \underline{\text{almost } 9}$$

$$\text{Electrical H.P. of motor} = 9 \times \frac{100}{90} = \underline{10}$$

The overall length of the pump and motor would be 6 feet 6 inches, its width 1 foot 6 inches and its height 2 feet 6 inches.

Fig. 148 shows a plan view of the pump installed at the face of the dipping incline, on a special track at one side of the road. It is moved forward every 10 feet of advance of the face, the time taken for this operation being about 15 to 20 minutes.

Fig. 148 also shows in elevation the pump sole-plate mounted on an undercarriage for sliding on the special track referred to above, and for keeping the pump in a horizontal position. The rail skids or runners are $2\frac{1}{2}$ inches by $\frac{1}{2}$ inch and they are turned round in the form of hooks at the top end. A steel bar $1\frac{1}{2}$ inches diameter is passed through the hooks and projects 3 inches on

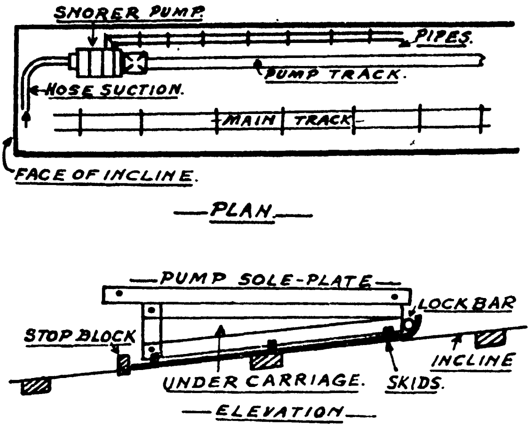


FIG. 148.—Arrangement of Portable Pump for Incline.

each side. By applying pinches to the ends of the steel bar at the top end of the undercarriage, the unit is skidded down the incline against a special stop-block bolted to the rails. A safety chain should be used during these operations.

Drawing a Pump Valve from given Details

8. A gunmetal valve seat (in the valve box of a pump) is 4 inches internal diameter, 3 inches deep, and at the top is bevelled or coned to a diameter of 5 inches at an angle of 45 degrees from the vertical. The cover of the valve box comes to 2 inches above the top of the valve seat. Design and draw a gunmetal mushroom valve to fit the valve seat. The lift of the valve is to be limited by the top of the valve, when open, coming into contact with the box cover. Show the valve, full size, by plan and sectional side elevation, and give the leading dimensions, including the lift allowed. (38)

A. Fig. 149 shows the plan and sectional side elevation of the mushroom valve referred to above, the drawing being on the half-size scale. The valve is guided on its seat by means of guide rings and wings fitting into the valve box. The four wings extend from the base of the valve to guide rings at the top and base of the cage. The wings are $\frac{1}{2}$ in. × $\frac{3}{8}$ in. × $1\frac{3}{4}$ ins. long, and the guide rings are $\frac{3}{8}$ in. × $\frac{1}{4}$ in. The lift of the valve is a quarter the diameter, or $1\frac{1}{4}$ inches.

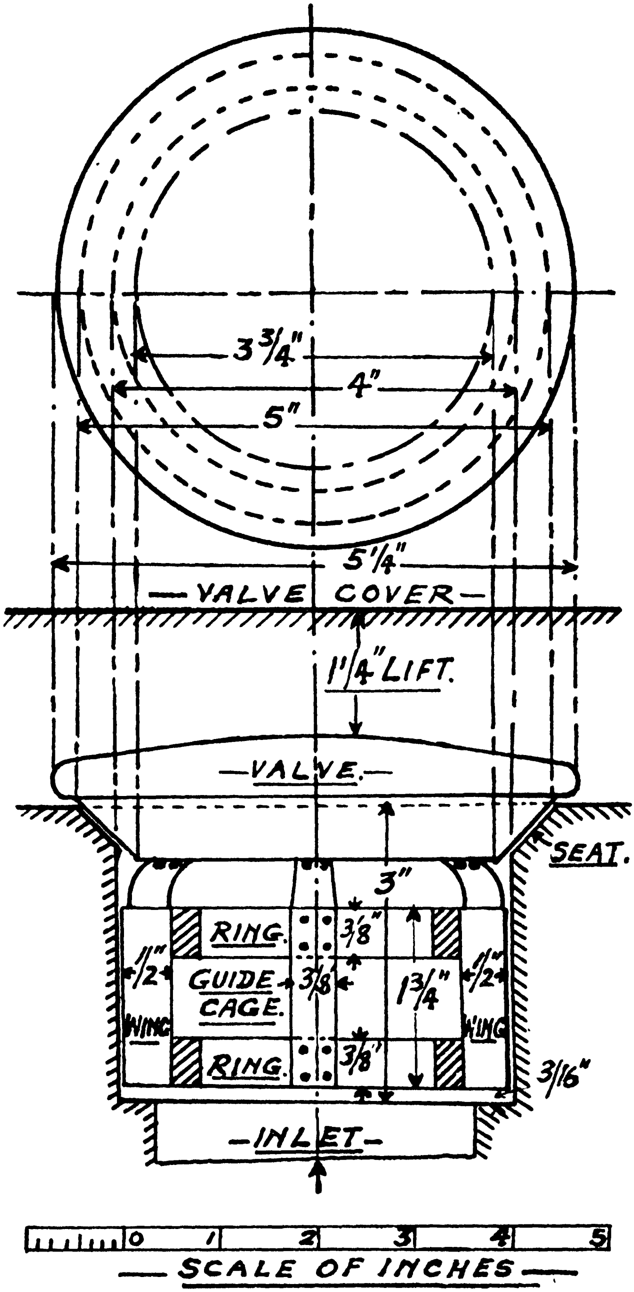


FIG. 149.—Plan and Sectional End Elevation of Mushroom Valve and Guide Cage.

JULY 1941 EXAMINATION

(*Five questions only to be answered.*)

ELECTRICAL SECTION

Transformer, Mercury-Arc Rectifier, and Static Condenser

1. Describe briefly the following apparatus and the uses that may be made of them about a colliery: (*a*) a transformer, (*b*) a mercury-arc rectifier, (*c*) a static condenser. (28)

A. A mining type transformer has been described previously; see pages 299–300 and Fig. 143.

Fig. 150 shows in diagrammatic fashion the general arrangement

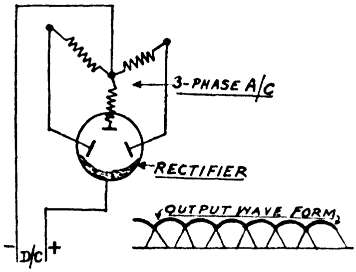


FIG. 150.—Mercury-Arc Rectifier.

of a *mercury-arc rectifier* installed to convert alternating-current to direct-current electricity. Rectifiers are now being used extensively in preference to rotary converters, where an alternating-current motor is coupled to a direct-current generator to convert the electrical energy from A.C. to D.C. With the converter set, a lower efficiency is obtained than with the rectifier. The transformation of the current produces considerable heat, and it is

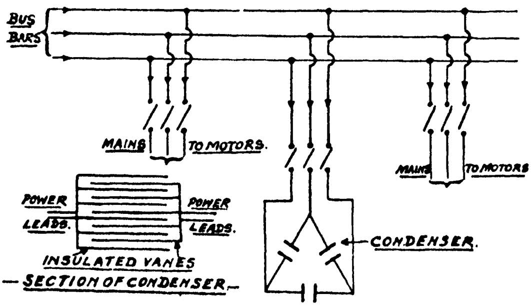


FIG. 151.—Static Condenser.

necessary to cool the rectifier by a current of air from a fan or by a continuous flow of water.

Fig. 151 shows by a simple line diagram the arrangement of a *static condenser* installed in alternating-current circuits to improve the power factor. The latter can reach a very low figure where the carrying capacity of cables is low, or where motors are run on a load very much below their normal rating. This gives rise to the current and the E.M.F. getting out of step, and consequently wattless current exists in the circuit. To restore the balance of E.M.F. and current in the circuit, static condensers are installed. These condensers are possessed of high-capacity effect, and they absorb the wattless current. When a demand is made for extra current in the circuit, the stored-up energy in the condenser is given out to it, and in this way a better balance of E.M.F. and current is obtained, thus improving the power factor.

Three-phase Overhead Bare Copper Transmission Line Losses

2. Quote the formula connecting current (I), voltage or pressure drop (V), and resistance (R) in a three-phase overhead bare copper transmission line. What other electrical property may play a part in the transmission of energy in a system with a voltage of, say, 6,600 volts between phases ? Given that the resistance of each conductor of a three-phase transmission is 1·5 ohms, what will the terminal voltage be if a voltmeter at the generator end reads 3,300 volts and the current is 100 amperes ? (28)

A. The formula usually applied for calculating voltage drop in three-phase alternating-current transmission lines is :—

$$\text{Voltage drop } (V) = \text{Current } (I) \times \text{Resistance } (R) \times \sqrt{3} \qquad (1)$$

The other electrical property which may play a part in the transmission of energy in an A.C. system is *Inductive Reactance*, due to the angle of lag.

The formula given above is only true where the applied voltage and the current are in step with each other, so that they both reach their maximum values and both their zero values at the same time. There is in this case no angle of lag, and no inductive reactance takes place, but only ohmic resistance. In most cases, however, the A.C. does not keep in step with the applied voltage, so that the maximum current does not flow just at the instant that the voltage is a maximum. There is in this case an angle of lag, and " self-inductance " takes place in addition to ohmic resistance. To include this effect, the formula becomes :—

$$V \text{ (drop)} = \text{Current } (I) \times [R \cos \theta + X \sin \theta] \times \sqrt{3}, \qquad (2)$$

where R = ohmic resistance ; X = inductive reactance in ohms $= 2\pi f L$, f being frequency and L henries ; θ = angle of lag, and $\cos \theta$ the power factor.

The maximum voltage drop is given by the formula :—

$$\left.\begin{aligned} V \text{ (drop)} &= \text{Current } (I) \times \text{Impedance} \times \sqrt{3} \\ &= \text{Current } (I) \times \sqrt{R^2 + X^2} \times \sqrt{3} \end{aligned}\right\} \qquad (3)$$

Neglecting the effect of inductive reactance and using formula (1),

$$\begin{aligned} V \text{ (drop)} &= \text{Current } (I) \times \text{Resistance } (R) \times \sqrt{3} \\ &= 100 \times 1{\cdot}5 \times \sqrt{3} \\ &= 259{\cdot}8, \text{ say } 260 \text{ volts.} \end{aligned}$$

∴ Terminal voltage is 3,300 − 260, or 3,040 volts.

Construction of Cables for Roads and Working Faces

3. In connection with electric coal-cutting machines and conveyors, all driven by three-phase alternating current on a longwall face, describe the cables to be used : (*a*) To lead the current along roads to switchgear at the face ; (*b*) to connect the face switchgear and coal-cutting machines ; and (*c*) to connect the face switchgear and motors of the conveyors. (28)

A. (*a*) Fig. 152 shows in section the construction of a cable

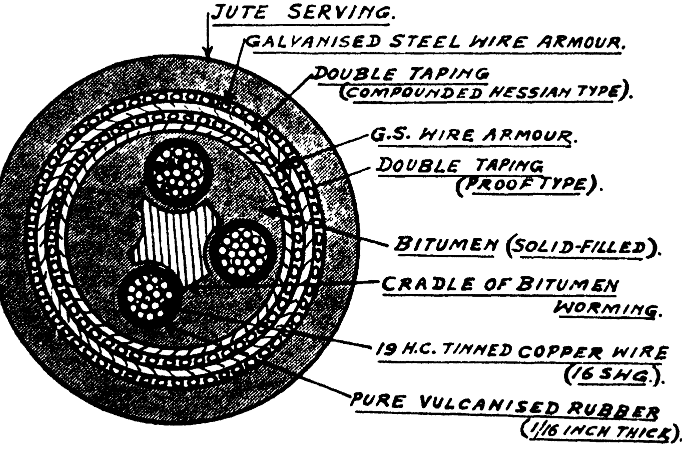


FIG. 152.—Section of Mine Road Cable.

for conveying current along roads to switchgear at the face at a pressure of 650 volts. Such a cable is of the double-wire armoured type with three cores of 19/0·064 high-conductivity tinned copper wires, enclosed in vulcanised rubber and solid-filled with bitumen. A cable of this design can be used for pressures up to 3,000 volts. It is suitable for supplying current to a double-unit face where coal-cutters, coal conveyors, loader, and drilling-machines are in use.

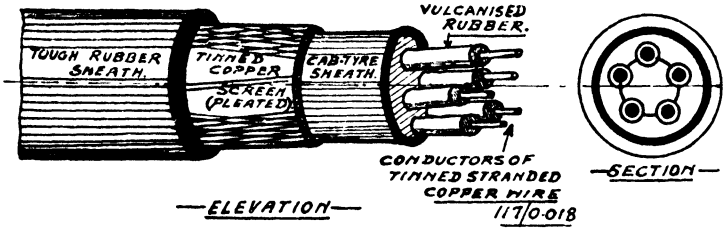


FIG. 153.—Trailing Cable for Coal-cutter for Remote Control.

(*b*) To connect the face switchgear and the motor of the coal-cutting machine a trailing cable of the cab-tyre-sheathed (C.T.S.) type is generally installed. Such a cable is shown in section and elevation in Fig. 153. It has three power cores, one pilot core, and one earth core, all of equal cross-sectional area and all suitably protected by a pleated and tinned copper screen, covered externally by a tough rubber sheath.

(*c*) To connect the face switchgear and the motors of the conveyors a pliable armoured cable is generally installed. This type of cable is shown in cross-section and elevation in Fig. 154. It

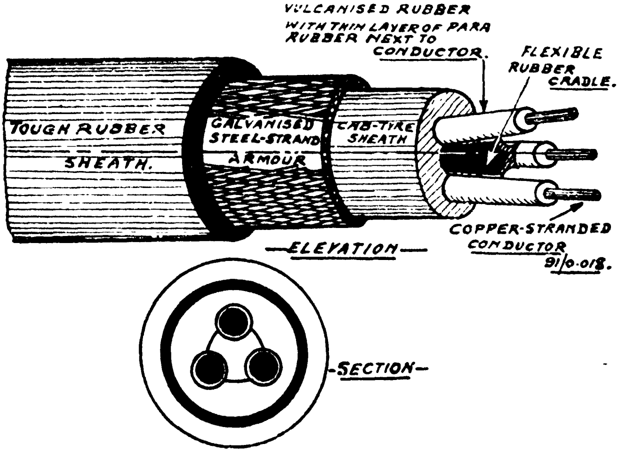


FIG. 154.—Pliable Armoured Cable for Conveyors and Loaders.

has three power cores in a cab-tyre sheath, protected by galvanised steel-strand armouring and outside by a tough rubber sheath. The three cores are separated by an internal flexible rubber cradle.

MECHANICAL SECTION

Receiver in a Compressed-Air System

4. What is the use of a receiver in a compressed-air system? How would you arrive at the desirable capacity of a receiver, or receivers, for a given system in which air is compressed at the surface and distributed underground? What fittings and mountings would you have on an air receiver? (28)

A. A receiver is used in connection with a compressed-air system to act as a reservoir of air power between the air-compressor and the air-motors in the mine, so that the latter do not slow down at frequent intervals under load, but tend to keep up a steady speed. Pipe-line losses by cooling and leakage are made good to a certain extent by the receiver, and a steady pressure of air is maintained at the motors. Pulsating effects of the compressor pistons are damped out by the receiver, and they do not reach the air-mains. The air-temperature is reduced in the receiver and water is precipitated. There should be a water separator fixed in the main pipe near the receiver. In order to prevent loss of pressure and freezing of moisture at the air-motors, the receiver should not be less than 10 to 15 times the volume of the air-cylinders or motors working from it. A compressed-air receiver should be installed on the surface for easy access.

Fig. 155 shows an outline sketch of a receiver for compressed

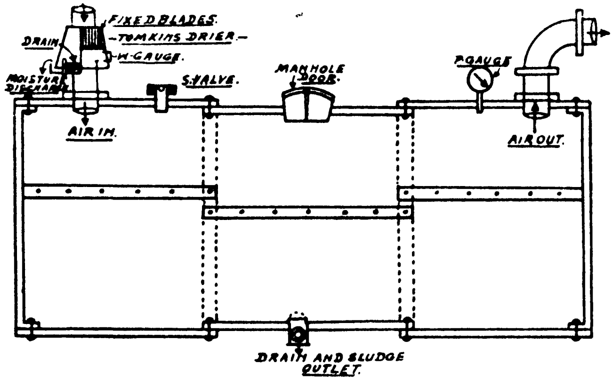


FIG. 155.—Arrangement of a Compressed-air Receiver.

air with the necessary fittings, such as safety-valve, pressure-gauge, drain and sludge cock, inlet and discharge branches, and manhole for examination and cleaning. A Tomkins dryer is fitted to the inlet branch, and the air is caused to rotate by fixed blades of turbine pattern, so that moisture, oil, dirt and solids are thrown off by centrifugal action, to be discharged through the outlet without loss in power.

Uses of Ball-bearings, Roller-bearings and Thrust-bearings

5. Draw freehand sketches to illustrate (*a*) a ball-bearing for carrying, say, the shaft of an electric motor; (*b*) a roller-bearing for a like purpose; and (*c*) a thrust ball-bearing. In an electric motor there is often one ball-bearing and one roller-bearing. Why is this combination adopted, and which of the two types of bearing is used at the driving end of the shaft? (28)

A. Fig. 156A shows in end elevation and section the arrangement of a ball-bearing of medium type, provided with ball cage to prevent the balls crowding together during rotation and thus causing sliding friction. The outer race is grooved internally to provide a rolling track for the balls, and is securely fixed in a housing to keep it stationary. The inner race is also grooved internally and is a tight fit on the shaft with which it revolves. The outside edges of the outer race and the inside edges of the inner race are slightly bevelled to facilitate fitting.

Fig. 156B shows in end elevation and section the arrangement of a roller-bearing of medium type, provided with a roller-cage for the same purpose as stated above. Such a bearing is used when heavy loading on a shaft occurs. A larger bearing surface takes the load, and less wear results.

Fig. 156C shows in sectional elevation a ball thrust-bearing consisting of two races grooved on one side for the balls, which are contained in a cage to facilitate assembling. The upper race fits easily on the end of the shaft, which is turned down so as to form a shoulder. The lower race should have a spherical base and rest in a corresponding seating in the housing, so that it can centre itself automatically. Sometimes thrust-bearings are made as complete units, in which a spherical seating is combined with a protective casing to give a most desirable improvement.

Generally motors are fitted with ball-bearings at both ends of the shaft, but roller-bearings might be used at the driving end of the shaft to meet some special circumstance, such as the motor being used to drive a heavily-loaded transmission belt. The roller-

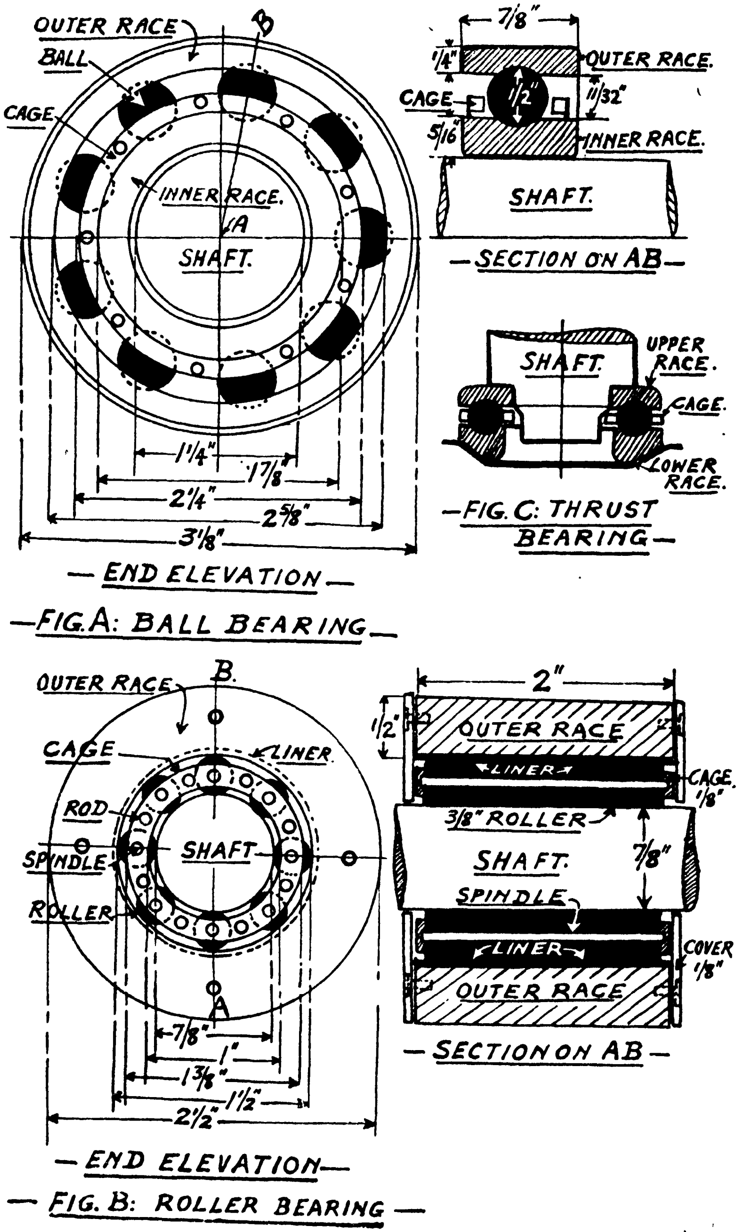


FIG. 156.—Ball, Roller, and Thrust-Bearings.

bearing has a greater bearing surface than the ball-bearing, and less trouble is experienced in maintaining it in condition.

Where a thrust may have to be counterbalanced on the end of a motor shaft, say, when a worm on the shaft is driving a worm-wheel on a shaft at right angles, the thrust is taken up by fixing a thrust-bearing at the end of the shaft, at the worm end, rather than by using the ordinary motor bearings. A similar thrust-bearing is generally installed on the worm-wheel shaft at the end nearest the wheel.

Specification of a Rising Main in a Shaft

6. Give a specification of a rising main of about 6 inches bore for a vertical shaft, 1,000 feet deep, to carry non-corrosive water. Deal with (*a*) the materials to be used for pipes and joints, and (*b*) the design of the joint. If the rising main is serving a centrifugal pump, for about what quantity of water, in gallons per minute, would you consider the main suitable? (28)

A. The rising main referred to in the above question might comprise the following: (*a*) Reflux valve and gate valve at the pump end, and (*b*) delivery branch of suitable pipes, with two cast-iron stools, one for each 500 feet of pipe-line, and collars at every third pipe, fixed to shaft buntons or walling. Fig. 157 shows the arrangement of the stool piece and the collars for the pipes.

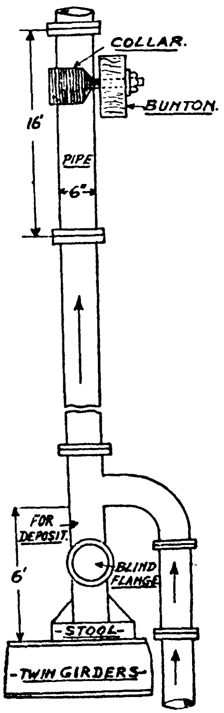


FIG. 157. — Stool and Clamps for Rising Main of a Pump.

Specification :—

(i) The pipes of the rising main are to be in 16-foot lengths of mild steel, the number required being approximately 63.

(ii) The internal diameter of the pipes is to be 6 inches and the external diameter $6\frac{1}{2}$ inches (see Calculation for thickness on page 318).

(iii) The weight of the pipes is approximately 8 tons (see Calculation for weight on page 319).

(iv) At the shaft bottom and at mid-shaft the rising main must rest on suitable stools, and every third pipe must be clamped to the shaft buntons or walling.

(v) The pipe joint is shown in section and elevation in Fig. 158. This loose-flange joint is the design of Messrs. Stewart and Lloyd,

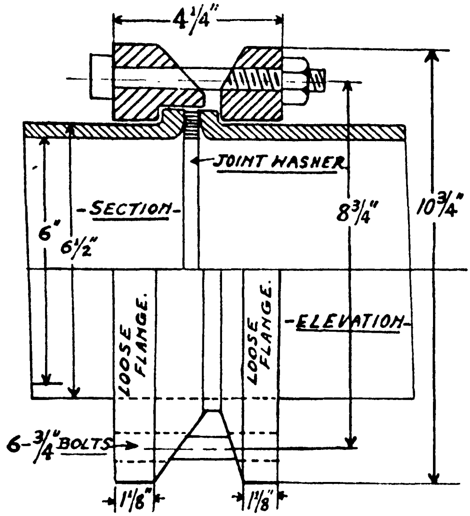


FIG. 158.—Section and Elevation of Loose-flange Pipe Joint.

Ltd., of Glasgow. Stout rings are welded on to the ends of the pipe, and the joint is completed by washers, mild-steel loose flanges, and bolts. This pipe joint is made to withstand great weight, and it is suitable for pipe columns in shafts.

Thickness of Pipe required :—

Let P = Pressure per sq. inch due to static head.
D = Internal diameter of pipe in inches.
F = Factor of Safety (8).
T = Thickness of pipe in inches.
T_s = Tensile strength of mild steel =

35,000 lbs. per sq. inch for pipes under 4 ins.
40,000 to 45,000 ,, ,, ,, ,, ,, 4 ins. to 7 ins.
50,000 ,, ,, ,, ,, ,, over 7 ins.

Then Tensile stress in pipe = Tensile strength of material

$$P \times D \times F = 2T \times T_s.$$

$$\therefore T = \frac{P \times D \times F}{2 \times T_s}$$

$$= \frac{1000 \times 0{\cdot}434 \times 6 \times 8}{2 \times 45{,}000}$$

$$= \underline{0{\cdot}23 \text{ inch}}, \text{ say } \tfrac{1}{4} \text{ inch.}$$

Weight of Pipe-line :—

Length is 1000 ft. + $\frac{1000}{16}$ for flanges, or 1063 feet.

$$\text{Weight} = \frac{(D^2 - d^2) \times L \times 2{\cdot}64 \text{ for mild steel}}{2240}$$

$$= \frac{(6\tfrac{1}{2} - 6) \times (6\tfrac{1}{2} + 6) \times 1063 \times 2{\cdot}64}{2240} = \underline{8 \text{ tons}} \text{ (almost).}$$

Flow of Water through Six-inch Pipes :—

Gallons per min. = area in sq. ft. × water velocity in ft. per min. × $6\frac{1}{4}$ gallons per cu. ft.

$$= \frac{6 \times 6 \times \frac{11}{14}}{144} \times 180 \times 6\tfrac{1}{4}$$

$$= 6 \times 6 \times 0{\cdot}034 \times 180$$

$$= \underline{220 \text{ gallons per min.}}$$

Characteristic Curves of Turbine Pump at Steady Speed

7. In the case of a centrifugal or turbine pump, state the relations between head, quantity, power, and efficiency, at a given steady speed. Illustrate the relations by drawing approximate characteristic curves. (38)

A. See a previous answer, pp. 297–8, and Fig. 142, showing characteristic curves for a turbine pump running at a steady speed.

As the gallons of water are increased by opening the gate valve the head falls, and the efficiency and B.H.P. rise steadily to a maximum value and then fall gradually. Between the limits taken there is a range of practical application of the pump which is marked on the diagram.

Troubles by Bad Feed-water in Lancashire Boilers—Remedies

8. What are the troubles that may occur in Lancashire boilers due to the quality of the feed-water ? What steps can be taken to prevent damage to the boilers ? (28)

A. See pages 239 and 286 for the answer given to a similar question.

PART V.—SURVEYING, LEVELLING AND DRAWING

MAY 1931 EXAMINATION

Contour Plan—Plotting of Sections

1. The accompanying plan (Fig. 159) shows the centre-line of part of a proposed colliery railway, contour lines of adjoining ground, and the level of the surface of the ground at each peg. The formation level of the railway at peg 10 is 20 feet above datum, and the gradient is 1 in 100, rising in the direction of peg 15. Plot a longitudinal section from peg 10 to peg 15 on a scale of 1 inch to 100 feet for horizontals, and 1 inch to 10 feet for verticals of section. Show the formation level and mark the depth of cutting at each peg. Plot cross-sections at pegs 12, 13 and 14 to a scale of 1 inch to 10 feet, and calculate therefrom the amount of cutting in cubic yards between pegs 12 and 14. Assume the width on formation level to be 15 feet and the angle of the slopes 1 vertical to $1\frac{1}{2}$ horizontal. (30)

A. Fig. 159 shows the contour plan referred to in the above question, also the longitudinal section from peg 10 to peg 15. Fig. 160 shows the cross-sections at pegs 12, 13 and 14.

Area of cross-section at peg 12 $= \dfrac{45 \times 5}{2} = 112$ sq. ft. (1)

" " " " 13 $= \dfrac{36 \times 8}{2} = 144$ sq. ft. (2)

" " " " 14 $= \dfrac{45 \times 11{\cdot}5}{2} = 259$ sq. ft. (3)

Amount of cutting between pegs 12 and 14

$$= \frac{A_{12} + 4A_{13} + A_{14}}{6} \times \frac{\text{length}}{27}$$

$$= \frac{112 + 4(144) + 259}{6} \times \frac{200}{27} = \underline{1{,}170 \text{ cubic yards}} \qquad (4)$$

Note.—Under Examination conditions, *five* questions only are to be answered from this Section. The figures in brackets indicate the maximum marks allotted to each question.

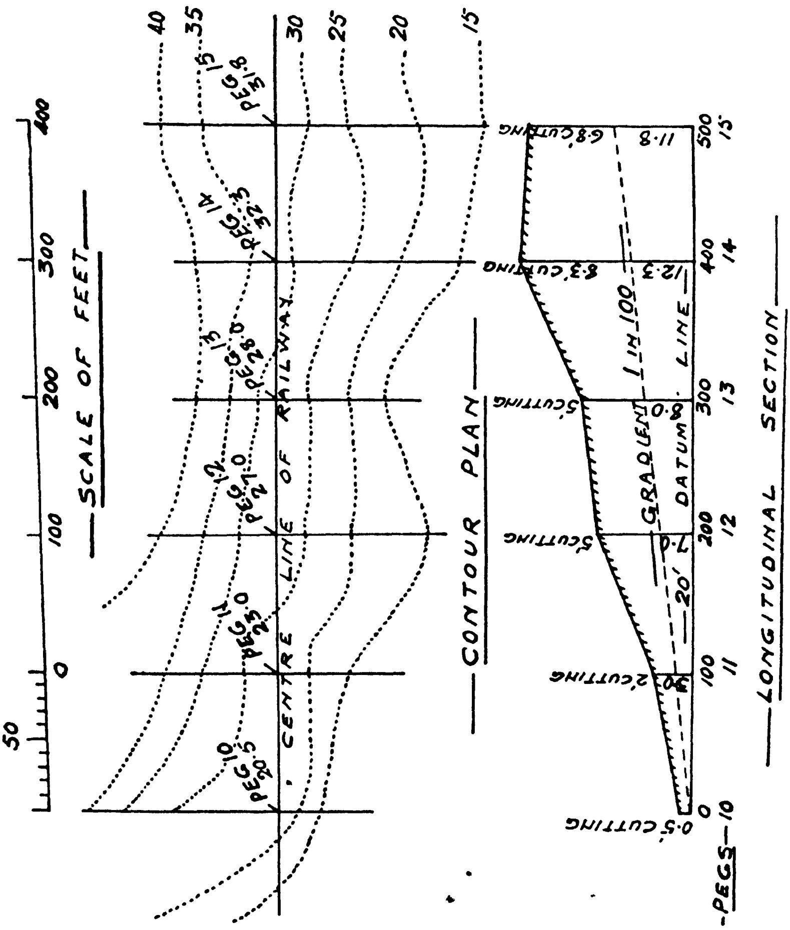
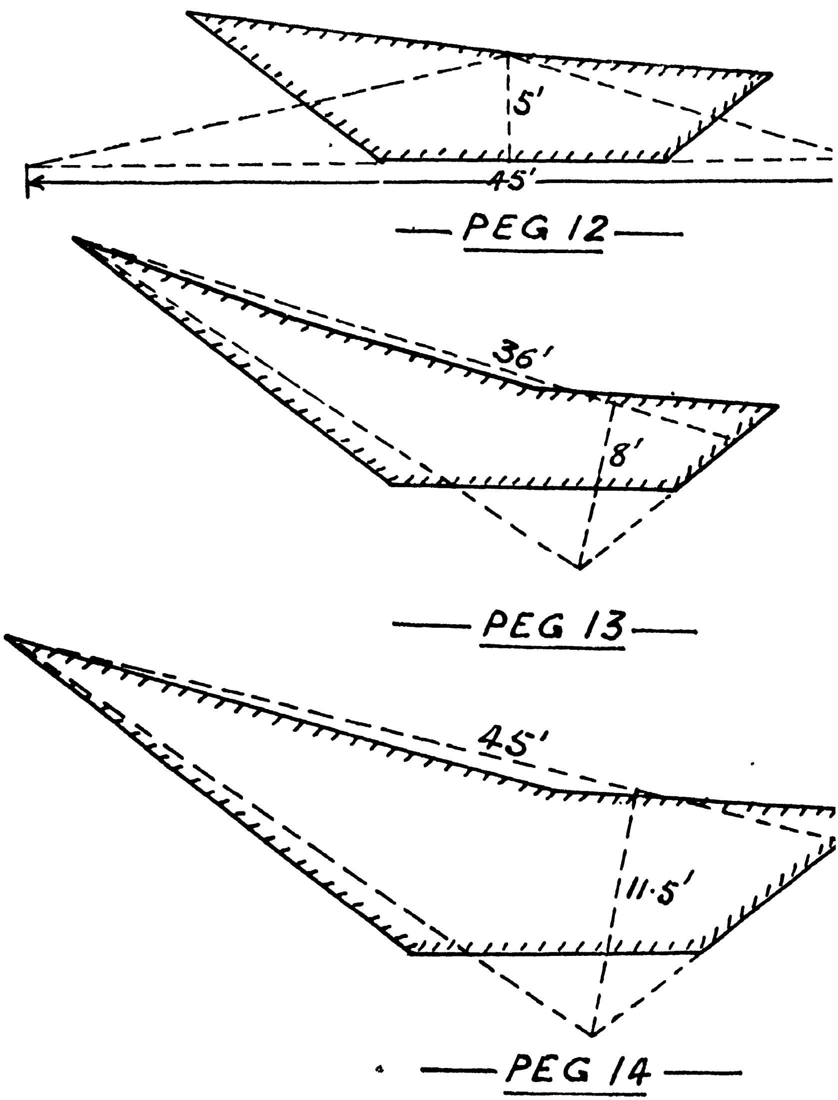


FIG. 159.—Contour Plan and Longitudinal Section of Railway.



Scale 1 inch = 10 feet.

FIG. 160.—Cross-sections at Pegs 12, 13 and 14 in Plan shown in Fig. 159.

Coal under Leasehold Intersected by a Fault

2. A colliery leasehold measures 1 mile square in area. It is intersected by a large fault which runs parallel to two of its sides, and has a vertical displacement of 80 yards. The hade of the fault is 30 degrees from the vertical. Calculate the quantity of coal within the leasehold, assuming a total thickness of 15 feet, with specific gravity 1·25, and that the strata dip 1 in 6. If you were asked to estimate the sale product of the leasehold, what further deduction, in addition to the large fault, would you make from the above quantity ? Assume longwall working throughout. (30)

A. Width of "want," or barren ground, caused by fault is $80 \times \tan 30°$, or 46·19 yards.

Area of barren ground $= 46{\cdot}19 \times 1760 = 81{,}290$ sq. yards.
Area of coal-bearing strata $= 1760^2 - 81{,}290 = 3{,}006{,}310$ sq. yards or 621 acres . . . (1)

Assuming 1 acre of water 1 inch deep to weigh 100 tons,
Tons contained $= 621 \times 15 \times 12 \times 100 \times 1{\cdot}25$
$= 13{,}972{,}500$ or almost 14,000,000 tons . . . . . (2)

$$\text{Tons contained allowing for inclination} = 14{,}000{,}000 \times \frac{\sqrt{37}}{6}$$
$$= 14{,}186{,}600, \text{ say } \underline{14{,}200{,}000 \text{ tons}} \quad (3)$$

Further deductions from the above might include 5 per cent. for loss in working and 5 per cent. for bad coal, stones and dirt, a total of 10 per cent.

Calculation of Co-ordinates of Mid-point of a Line

3. The co-ordinates of two points A and B, in relation to a common point or origin, are as follows :—

A : Latitude, North 188·6 feet ; departure, East 922·4 feet.
B : ,, South 495·4 feet ; ,, East 58·6 feet.

Calculate the co-ordinates of a point P which lies midway between A and B. Check your result by plotting on a scale of ½ inch to 100 feet. (20)

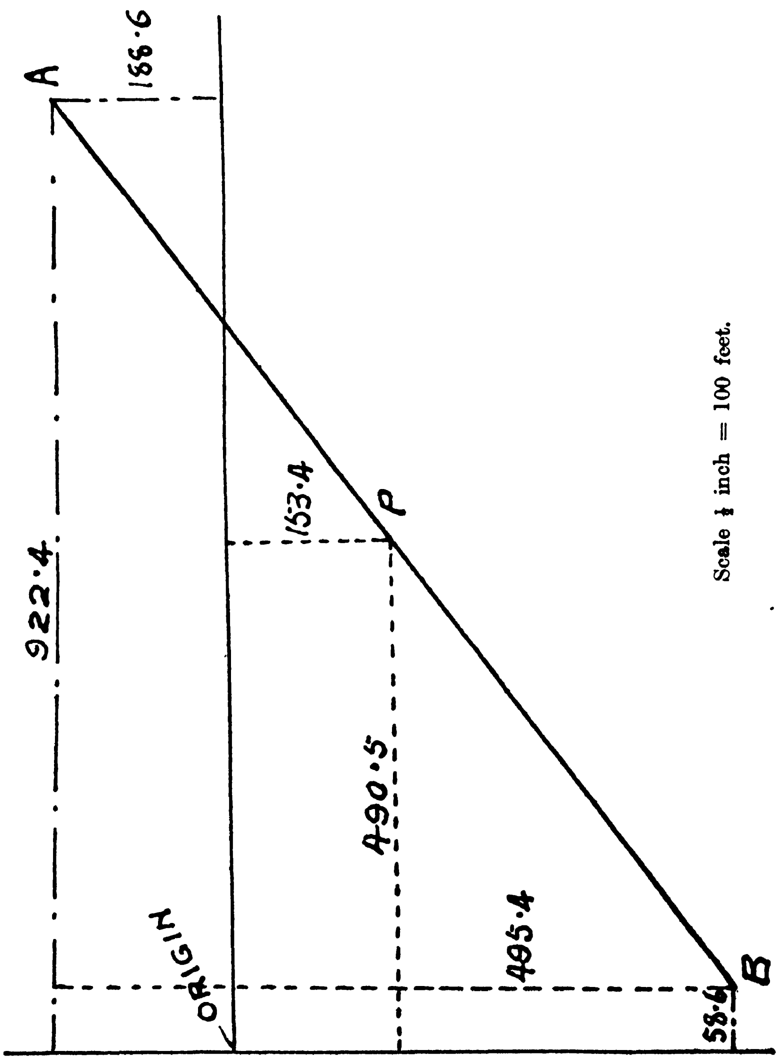
A. Latitude of P

$$= \frac{188{\cdot}6 + 495{\cdot}4}{2} - 188{\cdot}6 = \underline{153{\cdot}4 \text{ feet South}} \quad . \quad (1)$$

Departure of P

$$= \frac{922 \cdot 4 - 58 \cdot 6}{2} + 58 \cdot 6 = \underline{490 \cdot 5 \text{ feet East}} \quad . \quad . \quad (2)$$

Fig. 161 shows a plan plotted to the given scale and checking the above results.



Scale ¼ inch = 100 feet.

FIG. 161.—Plan showing Co-ordinates of Mid-point of Line.

Plotting a Short Traverse—Calculation of Area of Closed Figure

4. Plot the following short survey, or traverse, to a scale of 1 inch to 100 feet :—

From A :	N. 38° E. 215 ft. level		Join A and B, and mark the quadrant bearing and distance from B to A. Calculate the enclosed area by reducing it to a single triangle, giving the answer in acres and decimals. (30)
	N. 25° W. 250 ft. rising 1 in 2		
	N. 85° E. 325 ft. level		
	S. 40° E. 134 ft. dipping 1 in 3		
	S. 10° W. 262 ft. level to B		

A. Plotting distance of road 250 feet rising 1 in 2,

$$= \frac{250 \times 2}{\sqrt{5}} = 223 \cdot 6 \text{ feet.}$$

Plotting distance of road 134 feet dipping 1 in 3,

$$= \frac{134 \times 3}{\sqrt{10}} = 127 \text{ feet.}$$

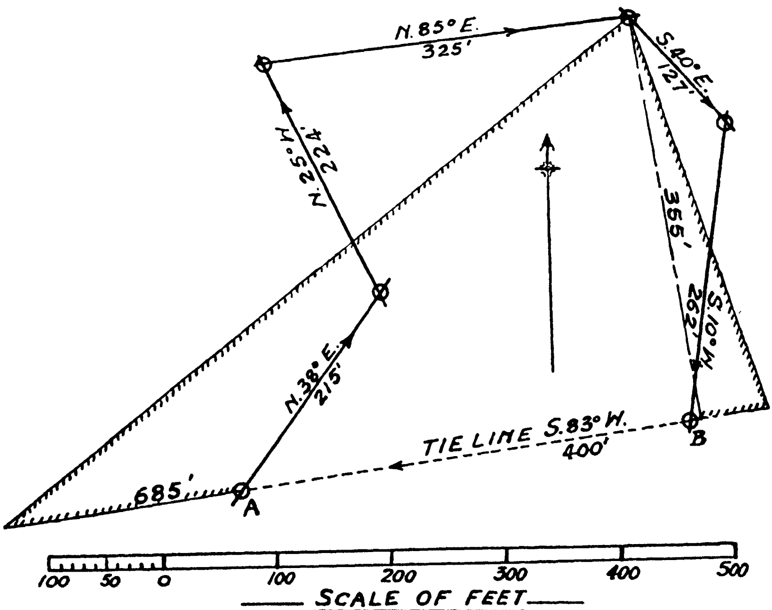


FIG. 162.—Plotting of Survey and Area Enclosed.

Fig. 162 shows the plotting required by the above question.

The tie line from B to A is S. 83° W. quadrant bearing, and the distance 400 feet.

Area of enclosed figure by reducing to a single triangle, shown shaded,

$$= \frac{685 \times 355}{2} = 121{,}588 \text{ sq. ft., or } \frac{121{,}588}{43{,}560} = \underline{2 \cdot 79 \text{ acres.}}$$

Definition of Terms connected with Levelling

5. State briefly what is meant by the undernoted terms, all of which are used in connection with levelling: (*a*) contour line; (*b*) datum line; (*c*) back sight; (*d*) fore sight; (*e*) line of collimation. (20)

A. (*a*) A *contour line* is an irregular line marked on a plan, which has the same height above or below datum throughout the whole extent of its path.

(*b*) A *datum line*, as used in levelling, has a value greater or less than Ordnance Datum Line, which is the height of the general mean level of the sea at Newlyn, Cornwall. Such a line is used as a starting-line in levelling.

(*c*) A *back sight* is the first sight taken by the levelling instrument in taking a series of levels. It is the starting sight with the staff placed on a known datum line, and from it the level values of all intermediate readings are deduced.

(*d*) A *fore sight* is the last sight taken by any setting of the levelling instrument, and it denotes the finishing staff reading before the instrument is removed to another station.

(*e*) A *line of collimation* is formed by joining the intersection of the cross-fibres of the diaphragm to the centre of the object-glass of the telescope; or it is the optical centre-line of the telescope. Such a line, extended beyond the limits of the telescope, and by which staff readings are taken, is known as the collimation line.

NOVEMBER 1931 EXAMINATION

Determination of Line of Outcrop

1. The plan (Fig. 163) shows an area of coal workings in a shallow seam with levels therein, and several contour lines on the surface in the vicinity of the coal face. Lay down on the plan a line representing as much of the inferred outcrop of the seam as the contour

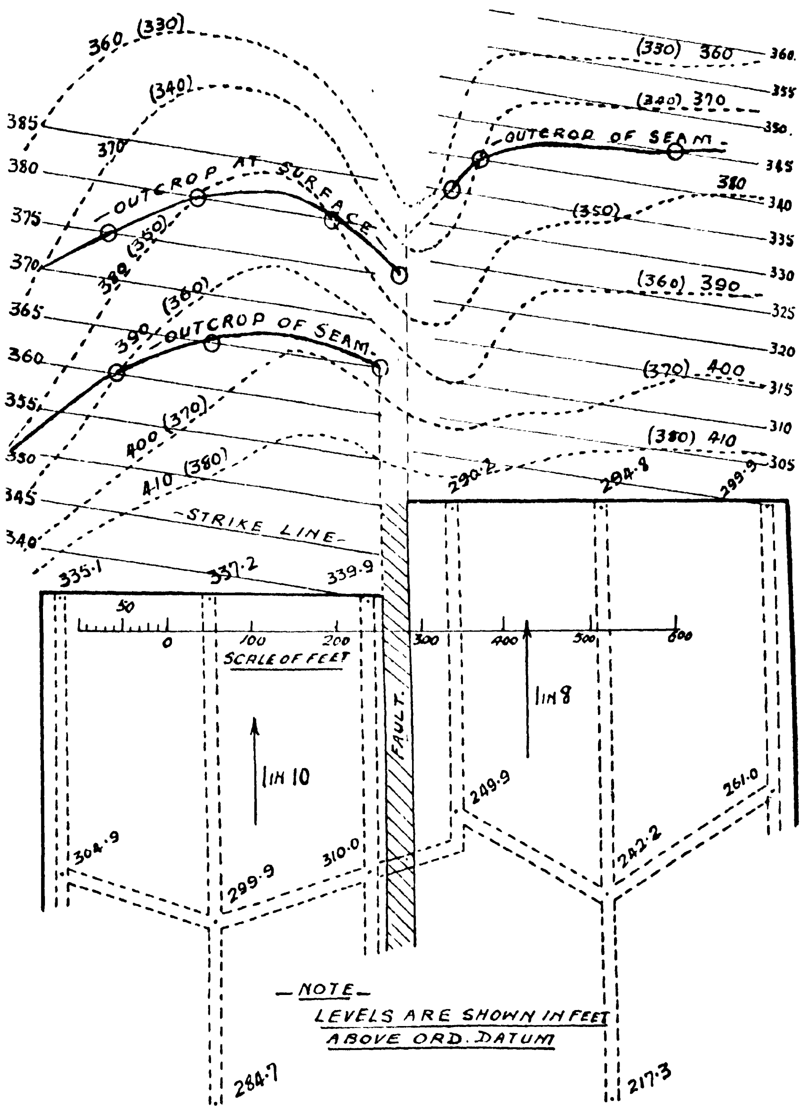


FIG. 163.—Contour Plan and Outcrop of Coal Seam.

lines shown will allow. Assume an average thickness of 30 feet of alluvium overlying the rockhead. (30)

A. Fig. 163 shows the plan referred to in the above question. The gradient of the seam on the left-hand side of the fault is $\frac{335{\cdot}1 - 304{\cdot}9}{300}$, or 1 in 10. Strike lines 50 feet apart and 5 feet difference in level are constructed as shown on the plan. The outcrop of the seam is obtained at points where a contour line meets a strike line of the same level. Such points are shown by small circles on the plan. The level values in brackets represent heights to the rockhead.

The gradient of the seam on the right-hand side of the fault is $\frac{290{\cdot}2 - 249{\cdot}9}{320}$, or 1 in 8. Strike lines 40 feet apart and 5 feet difference in level are constructed as shown. The small circles again determine the points of outcrop under the 30 feet of alluvium.

Calculation of Length of Roadway

2. Two underground roadways, *AB* and *AC*, diverging from a common point *A* and bearing N. 30° W. and N. 30° E., respectively, are connected by another roadway *BC* which is 600 feet in length and bears from *B* to *C* in the direction S. 75° E. Calculate the length of the roadway *AC*. (30)

A. Fig. 164 shows the triangle formed by the three lines:—
Angle A = 60°; angle B = 45°; angle C = 75°.

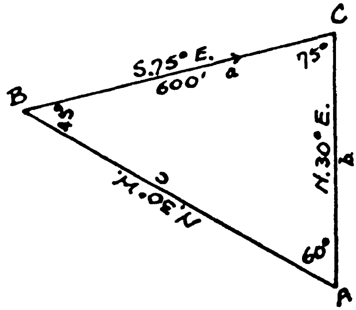


FIG. 164.—Direction of Underground Roadways.

By the sine rule: $\dfrac{a}{\sin A} = \dfrac{b}{\sin B} = \dfrac{c}{\sin C}$

so that $\dfrac{600}{\sin 60^\circ} = \dfrac{b}{\sin 45^\circ}$

$$\therefore b = \frac{600 \times \sin 45^\circ}{\sin 60^\circ} = \frac{600 \times \frac{1}{\sqrt{2}}}{\frac{\sqrt{3}}{2}} = \frac{600 \times 2}{\sqrt{2} \times \sqrt{3}} = \frac{1200 \times \sqrt{6}}{6}$$
$$= \underline{490 \text{ feet.}}$$

The distance or length of the roadway AC is thus 490 feet.

Calculation of Co-ordinates

3. Calculate the co-ordinates of the undernoted survey or traverse, and the bearing and length of the line from A to B:—From A—S. 41° W., 180 feet; S. 48° E., 160 feet; E., 200 feet; N. 50° E., 140 feet; N. 40° W., 200 feet—to B. (30)

A. Calculation of co-ordinates:—

S. 41° W., 180 feet. Latitude = 180 × cos 41° = 136 feet
Departure = 180 × sin 41° = 118 „
S. 48° E., 160 feet. Latitude = 160 × cos 48° = 107 „
Departure = 160 × sin 48° = 119 „
E., 200 feet. Latitude = nil.
Departure = 200 feet.
N. 50° E., 140 feet. Latitude = 140 × cos 50° = 90 „
Departure = 140 × sin 50° = 107 „
N. 40° W., 200 feet. Latitude = 200 × cos 40° = 153 „
Departure = 200 × sin 40° = 128 „

The following table shows the booking of the survey:—

Line.	Bearing.	Distance.	Latitude.		Departure.	
			N.	S.	E.	W.
A—1 . .	S. 41° W.	180 feet	—	136	—	118
2 . .	S. 48° E.	160 „	—	107	119	—
3 . .	E.	200 „	—	—	200	—
4 . .	N. 50° E.	140 „	90	—	107	—
5—B .	N. 40° W.	200 „	153	—	—	128
		Totals	243	243	426	246

The North and South values balance, and the East value exceeds that of the West by 426 – 246, or 180 feet.

The bearing of the tie line from A to B is East, and its distance is 180 feet.

Measuring and Checking a Survey Line over Steep Ground

4. Describe briefly how you would accurately measure a survey line over steep and rough ground. A survey line across a fairly deep ravine has been measured as accurately as possible. Describe, with sketches, how you would check the measurement so obtained by using (a) chain and ranging poles only; (b) theodolite, chain and ranging poles. (20)

A. Fig. 165 (a) shows how the accurate measurement of a line might be determined over steep and rough ground. The measuring-

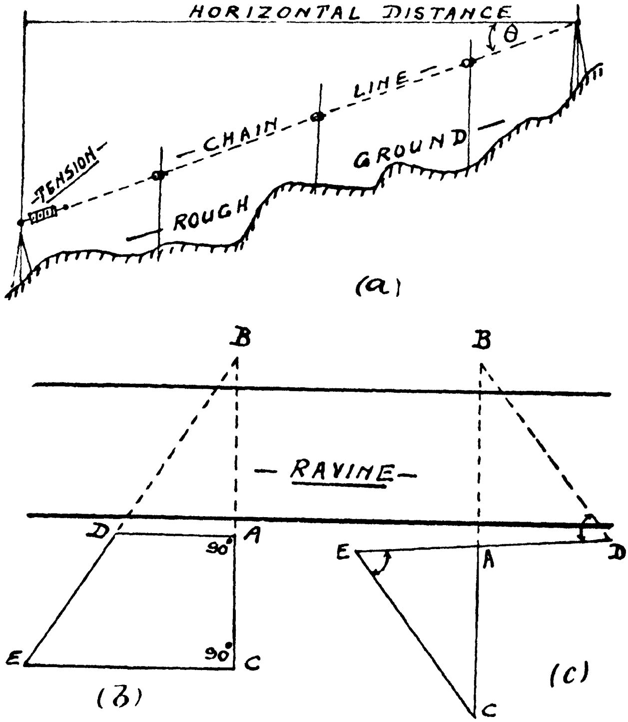


FIG. 165.—Measurement of Distances over Steep and Rough Ground and of Inaccessible Distances.

chain is fixed up to tripods at the respective ends and is supported by poles and rings at several intermediate points. At one end the chain is properly tensioned by a spring-balance so as to reduce sag. The correct horizontal distance is obtained by taking the angle of inclination of the chain and multiplying the inclined distance by the cosine of this angle :—

Correct horizontal distance = Inclined distance $\times \cos \theta$

Fig. 165 (*b*) shows how the distance across a deep ravine can be checked by using a chain and ranging poles. The line BAC is ranged out as shown, and right-angles are set out at the points A and C by using 30, 40 and 50 links to form a right-angle. The line BDE is then ranged out and AD, CE and AC are measured. To calculate the inaccessible distance AB the following ratios about equal angles of similar triangles are taken :—

$$\frac{AB}{AD} = \frac{AB + AC}{CE}$$

$$\therefore\ AB \times CE = AD(AB + AC)$$

$$AB \times CE = (AD \times AB) + (AD \times AC)$$

$$AB(CE - AD) = AD \times AC$$

$$\text{and } AB = \frac{AD \times AC}{CE - AD}$$

Fig. 165 (*c*) shows how the foregoing inaccessible distance can be checked by using theodolite, chain and poles. The line BAC is ranged out as before, C being a temporary point on the line to be definitely fixed later by using the theodolite. The line DAE is then ranged so that $AD = AE$. With the theodolite at D the angle ADB is taken. The instrument is then transferred to the point E, and the angle AEC is made equal to that taken at the point D. This operation definitely fixes the point C on the line BAC. The constructed triangles are equiangular and have $AD = AE$; they are therefore equal in all respects, and $AB = AC$.

Scale Equivalents and Calculation of Plan Area

5. State the equivalent in inches to a mile of the following scales : $\frac{1}{10560}$, $\frac{1}{1584}$, $\frac{1}{500}$. If the area of a field marked on a map drawn on the first-mentioned scale is 1·75 square inches, what is its area in square inches on maps drawn to the second and third-mentioned scales respectively ? (20)

A. A scale of $\frac{1}{10560}$ means that 1 inch on the plan is equal to 10,560 inches of ground.

$\therefore$ 1 sq. in. on plan is equivalent to $(10{,}560)^2$ sq. in. of ground.

$$\text{Inches on plan to a mile} = \frac{1760 \times 3 \times 12}{10{,}560} = 6 \text{ ins.} \quad . \quad (1)$$

$$\text{Scale of } \tfrac{1}{1584} : \text{Inches to a mile} = \frac{1760 \times 3 \times 12}{1584} = 40 \text{ ins.} \quad . \quad (2)$$

$$\text{Scale of } \tfrac{1}{500} : \text{Inches to a mile} = \frac{1760 \times 3 \times 12}{500} = 126{\cdot}72 \text{ ins.} \quad (3)$$

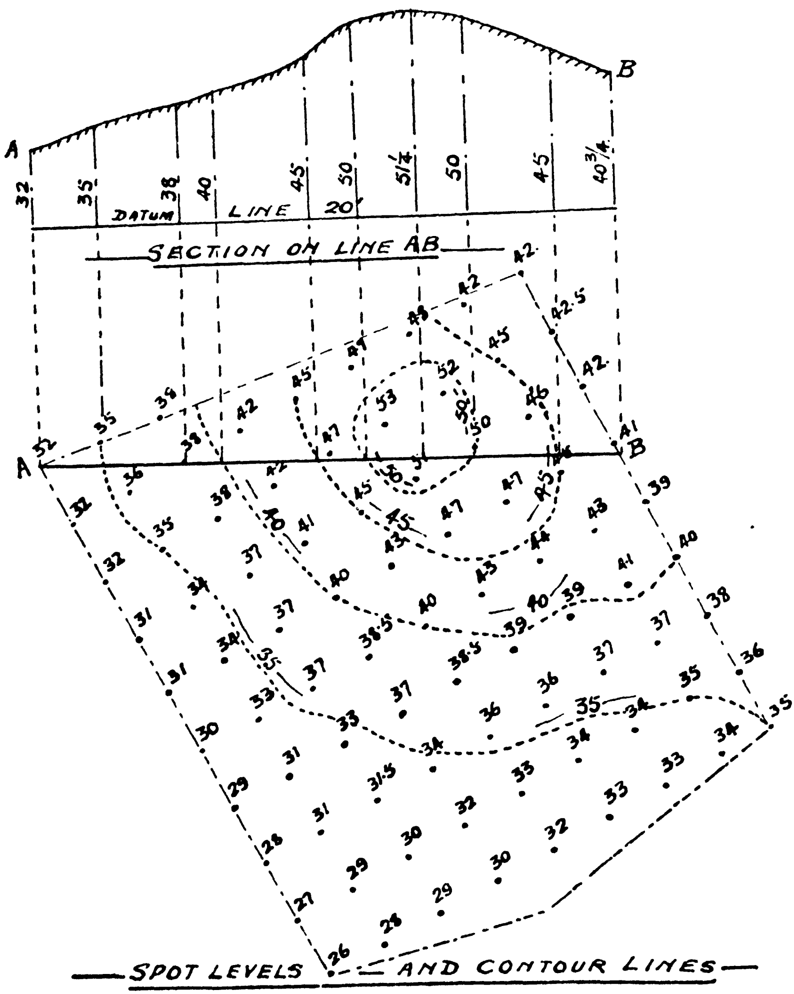


FIG. 166.—Drawing Contours from Spot Levels and Plotting a Section.

$$\text{Area for scale of } \tfrac{1}{1584} = 1{\cdot}75 \times \left(\frac{10{,}560}{1584}\right)^2 = 1{\cdot}75 \times \left(\frac{40}{6}\right)^2$$

$$= \underline{77\tfrac{7}{9} \text{ sq. in.}} \quad . \quad . \quad . \qquad (4)$$

$$\text{Area for scale of } \tfrac{1}{500} = 1{\cdot}75 \times \left(\frac{10{,}560}{500}\right)^2 = 1{\cdot}75 \times \left(\frac{126{\cdot}72}{6}\right)^2$$

$$= \underline{780{\cdot}59 \text{ sq. in.}} \quad . \quad . \quad . \qquad (5)$$

MAY 1932 EXAMINATION

To Draw Contour Lines on a Plan and to Plot a Section

1. The accompanying plan, drawn to a scale of $\frac{1}{2}$ inch to 100 feet, shows an enclosure on the surface of which spot levels are indicated in feet above ordnance datum. Sketch on the plan contours at vertical intervals of 5 feet as accurately as the levels shown on the plan will allow. Take the lowest contour at 35 feet above datum. Thereafter plot a section along the line *AB* by direct projection from the plan, making the scale for verticals ten times that of the plan. (20)

A. Fig. 166 shows the plan referred to in the above question; the contour lines at 5-foot intervals are shown by the dotted lines for values from 35 to 50 feet above datum. The section on the line *AB* is also shown.

To Determine the Length of an Inaccessible Line

2. Fig. 167, not to scale, shows by the letters *CAB* the proposed line of an aerial ropeway, part of which it is impossible to measure, as it crosses an area of soft marshy ground. Given that the lines

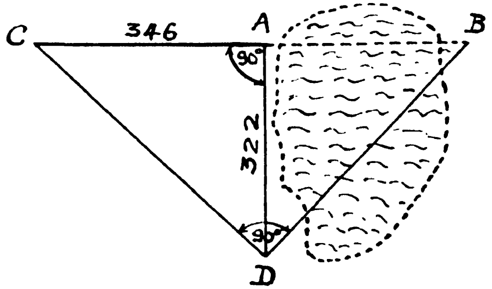


FIG. 167.—Measurement of Inaccessible Distance.

AC and AD measure 346 and 322 feet respectively, and the angles CAD and CDB are right-angles, calculate the length of the line AB. (30)

A. The triangles ACD and ABD have a common side AD, equal angles at the point A, and angle B equal to the angle ADC. These two triangles are thus equiangular and therefore similar.

Ratios about equal angles are $\frac{AC}{AD}$ and $\frac{AD}{AB}$.

Because the angles are equal the ratios are also equal;

$$\therefore \qquad \frac{AC}{AD} = \frac{AD}{AB}$$

and $$AB = \frac{AD^2}{AC} = \frac{322 \times 322}{346} = \underline{299{\cdot}6 \text{ feet.}}$$

Geographical Azimuth of Underground Survey Line from given Observations

3. The geographical azimuth of a line joining two triangulation stations on the surface is 357° 59′ 30″. By observation at 10 a.m. on consecutive days the magnetic azimuth of the line has been found to be 15° 55′ 00″. At the same hour on the two succeeding days the magnetic azimuth of an underground survey line was observed to be 58° 58′ 00″. What is the geographical azimuth of the underground survey line? (20)

A. Magnetic declination $= (360° - 357° \, 59' \, 30'') + 15° \, 55' \, 00''$
$= 2° \, 0' \, 30'' + 15° \, 55' \, 00''$
$= 17° \, 55' \, 30''$ . . . . (1)

Geographical azimuth of survey line $= 58° \, 58' \, 00'' - 17° \, 55' \, 30''$
$= \underline{41° \, 02' \, 30''}$ . . (2)

NOVEMBER 1932 EXAMINATION

To Calculate Length of Line Measured by a Stated Scale

1. The length of a surface line on a plan drawn to a scale of $\frac{1}{2500}$ is measured with a scale of $\frac{1}{2}$ inch = 1 chain, and found to indicate 1,650 links. What is its actual distance? (20)

A. Scale of $\frac{1}{2500}$ is 1 inch to 2,500 inches.
Scale of $\frac{1}{2}$ inch to 1 chain is 1 inch to 1,584 inches.

The distance measured from the plan is therefore too short, owing to the larger scale being used.

$$\text{Actual distance} = 1650 \times \frac{2500}{1584} = \underline{2,604 \text{ links.}}$$

To Calculate the Throw of a Fault and Length of a Stone-drift

2. A roadway advancing on the line of full dip in a seam dipping due South at 30 degrees, is intersected by an up-throw fault bearing East and West with a hade of 30 degrees from the vertical.

From the point where the fault is struck, a stone-drift, bearing in the same direction as the roadway, is driven to prove the fault, and reaches the seam at a point 60 feet—measured along the seam—from the plane of the fault. The levels at the ends of the stone-drift are 284 feet and 352 feet respectively above Ordnance Datum.

Calculate the throw of the fault, and the length and inclination of the stone-drift, assuming that the direction and rate of dip of the seam have been unaltered by the fault. (30)

A. Fig. 168 shows the above details diagrammatically. AB is

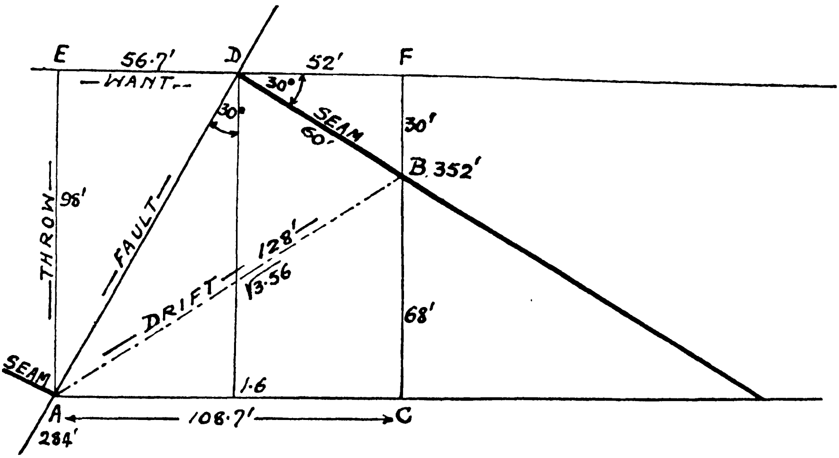


FIG. 168.—Showing Length and Inclination of Stone-drift.

the drift from the low side of the fault to the seam at the top side of the fault.

BC = Difference in level of drift = 352 − 284 = 68 feet.
BD = 60 feet and angle BDF = 30 degrees.
$FB = 60 \times \sin 30° = 30$ feet
∴ Throw of fault, FC, = EA = <u>98 feet</u> . (1)
$DF = 60 \times \cos 30° = 52$ feet
Want ED = Throw × tan 30°
$= 98 \times \tan 30° = 56{\cdot}7$ feet

$$\therefore \text{ Inclination of drift} = \frac{68}{52 + 56{\cdot}7} = \frac{1}{1{\cdot}6} \text{ or } \underline{1 \text{ in } 1{\cdot}6} \quad . \quad (2)$$

$$\text{Length of drift} = \frac{108{\cdot}7 \times \sqrt{(1{\cdot}6)^2 + 1^2}}{1{\cdot}6} = \underline{128 \text{ feet}} \quad . \quad (3)$$

Principle of the Vernier—Application to a Mining Dial

3. Explain the principle of a vernier, and describe briefly, with a sketch, its application to a mining dial. Draw your sketch to show such a vernier indicating a reading of 130° 35′ azimuth. (30)

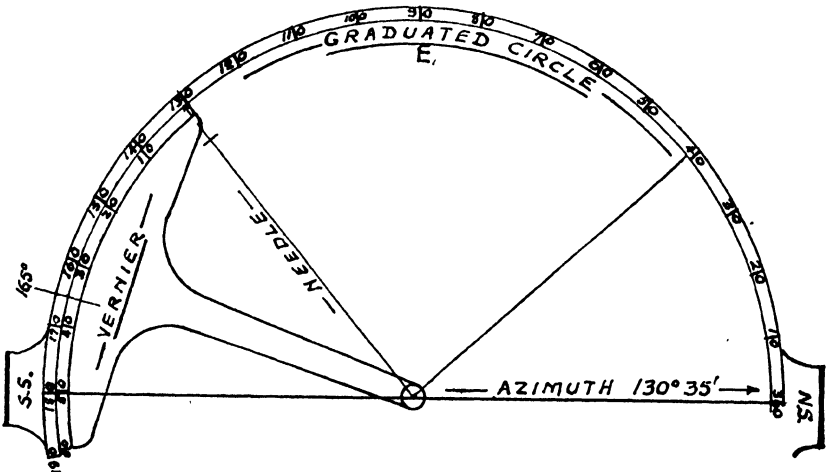


FIG. 169.—Compass showing Vernier Reading of 130° 35′.

A. A vernier scale is a small scale which is made to slide on the edge of a larger scale, so as to obtain a more accurate reading of the larger scale.

Fig. 169 shows the construction of a compass with moving graduated circle and fixed vernier. Such a vernier is constructed to read to 1 minute, by taking 59 divisions on the graduated circle and dividing the same into 60 divisions for the vernier scale, each division of the vernier scale representing $\frac{1}{60}$ of a degree or 1 minute.

The sketch shows the vernier reading an azimuth of 130° 35′, the 35-division mark of the vernier exactly coinciding with the 165-degree division on the graduated circle.

To Calculate the Bearing and Length of a Line joining the Ends of a Traverse

4. Undernoted are details of a short traverse which was made for the purpose of determining the bearing and length of a level cross-measure drift, proposed to be driven from the point *B* to another point vertically under the centre of the shaft *A*, with which it is to be connected by piercing upwards.

Calculate, as accurately as the undernoted tables will allow, the bearing and distance from the point *B* to the starting-point of the proposed shaft, and the extent of the shaft to be pierced up.

From centre of shaft *A*—	East . . .	195 ft.	level.
	S. 9° E. . .	621 ,,	dipping 33°
	S. 8° E. . .	554 ,,	,, 26°
	N. 87° W. . .	371 ,,	level to *B*.

(30)

A. Reduction of Inclined Distances to Plotting Distances :—

621 feet dipping 33° $= 621 \times \cos 33° = 521$ ft. plotting distance.
and $621 \times \sin 33° = 338{\cdot}2$ ft. difference in level.
554 feet dipping 26° $= 554 \times \cos 26° = 498$ ft. plotting distance.
and $554 \times \sin 26° = 242{\cdot}9$ ft. difference in level.

Calculation of Co-ordinates :—

Latitude	$521 \times \cos 9°$	$= 514{\cdot}59$	feet.
,,	$498 \times \cos 8°$	$= 493{\cdot}17$	,,
,,	$371 \times \cos 87°$	$= 19{\cdot}40$	,,
Departure	$521 \times \sin 9°$	$= 81{\cdot}48$	,,
,,	$498 \times \sin 8°$	$= 69{\cdot}32$	,,
,,	$371 \times \sin 87°$	$= 370{\cdot}48$	,,

BOOKING OF SURVEY DETAILS.

Bearing.		Distance.	Latitude.		Departure.	
			N.	S.	E.	W.
From A :	East. .	195 ft.	—	—	195·00	—
	S. 9° E. .	521 „	—	514·59	81·48	—
	S. 8° E. .	498 „	—	493·17	69·32	—
To B :	N. 87° W.	371 „	19·40	—	—	370·48
		Totals	19·40	1007·76	345·80	370·48

$$\text{Tangent of Bearing} = \frac{\text{Difference in departure}}{\text{Difference in latitude}}$$

$$= \frac{370{\cdot}48 - 345{\cdot}80}{1007{\cdot}76 - 19{\cdot}40} = \frac{24{\cdot}68}{988{\cdot}36} = 0{\cdot}025$$

$\therefore$ Bearing is N. 1° 26′ E. from B to A . . (1)

Distance is latitude $\times$ secant of 1° 26′

$$= 988{\cdot}36 \times \frac{1}{0{\cdot}999} = \underline{990 \text{ feet}} \quad . \quad . \quad (2)$$

Extent of shaft to be pierced upward = 338·2 ft. + 242·9 ft.

= 581·1 feet . . (3)

MAY 1933 EXAMINATION

Plotting Cross-sections of Roadway from a Contour Plan

1. The plan, Fig. 170, shows an area of ground with contour lines at vertical intervals of 10 feet. It is proposed to construct a roadway, of which the centre-line AB is shown on the plan. The roadway is to be 30 feet in width, with a uniform fall of 1 in 30 from A to B. The formation level at A is 100 feet.

Draw, on your answer book, to the scale of the plan, two sections, one on the line CD and another on the line EF, shown on the plan, and from them calculate the cross-sectional area of excavation and embankment respectively for the completed roadway. Assume that the slopes batter $1\frac{1}{2}$ horizontal to 1 vertical. (30)

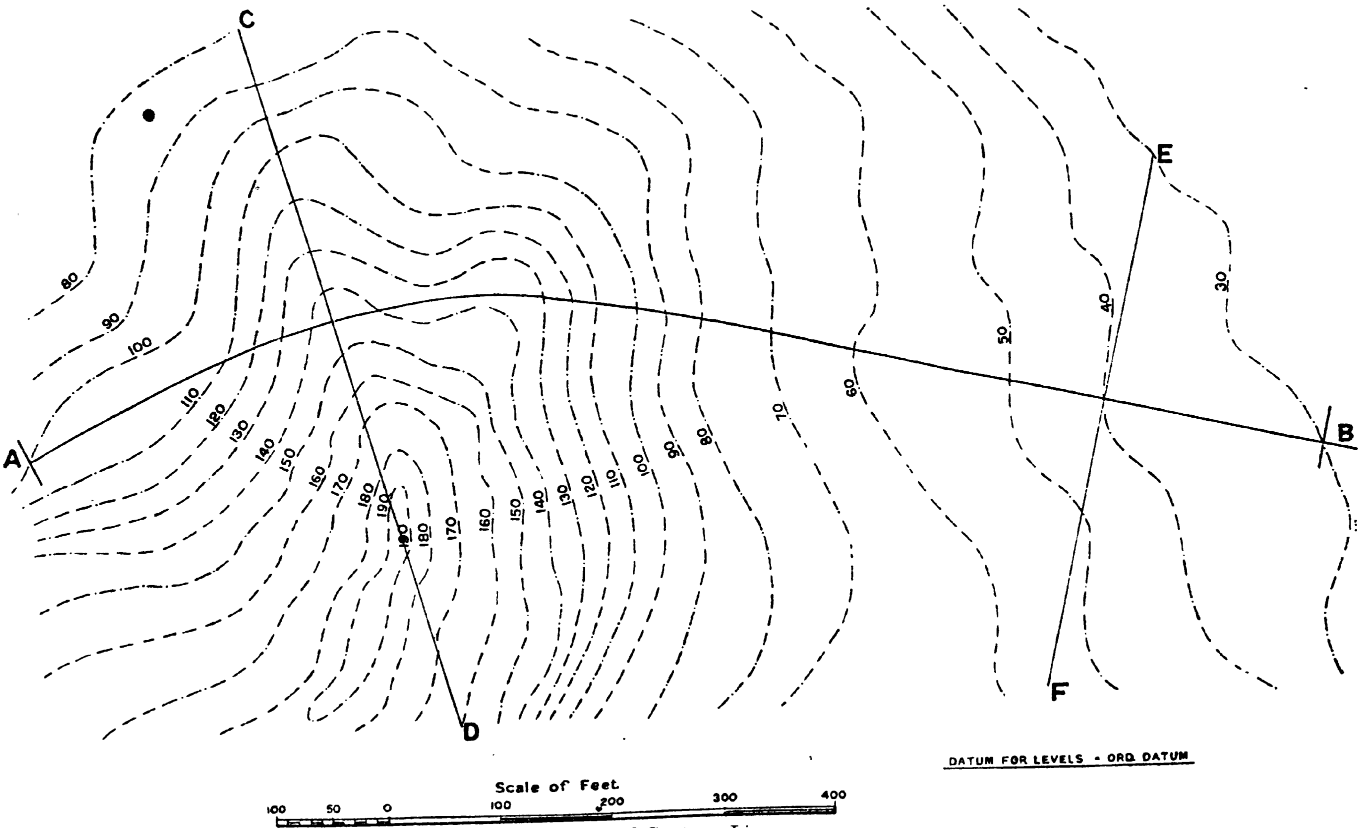


FIG. 170.—Plan of Contour Lines.

A. Fig. 171 shows the cross-sections on the roadway referred to in the above question at the points *CD* and *EF*.

From the point *A* to the centre of the road at the line *CD* the measurement is 300 feet. The gradient dips at 1 in 30, and therefore the centre of the roadway at the line *CD* is 10 feet below *A*, or at 90 feet level.

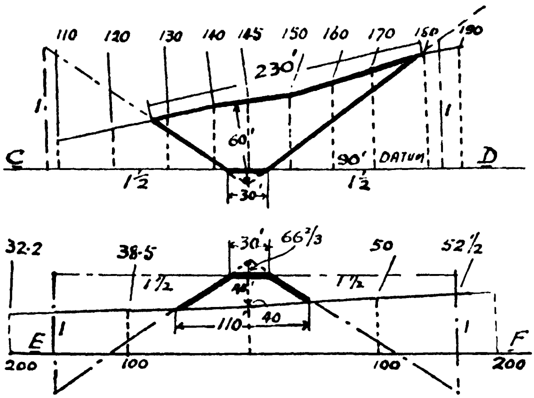


FIG. 171.—Cross-sections of Roadway at Points *CD* and *EF*.

The cross-sectional area of excavation of the roadway on the line *CD*

$$= \frac{230 \times 60}{2} - \frac{30 \times 10}{2}$$

$$= \underline{6{,}750 \text{ sq. ft.}} \quad . \quad . \quad . \quad . \quad . \quad (1)$$

From centre of roadway at *CD* to centre of roadway at *EF* measures 700 feet. The gradient dips at 1 in 30, and therefore centre of roadway at line *EF* is $\frac{700}{30}$ or $23\frac{1}{3}$ feet below the level of the roadway at line *CD*.

The centre of roadway level at the line *EF* is thus $90 - 23\frac{1}{3}$ or $66\frac{2}{3}$ feet. The cross-sectional area of embankment of the roadway on the line *EF*

$$= \frac{110 \times 40}{2} - \frac{30 \times 10}{2}$$

$$= \underline{2{,}050 \text{ sq. ft.}} \quad . \quad . \quad . \quad . \quad . \quad (2)$$

Reducing the Field Notes of a Levelling and Plotting a Section

2. Reduce the following field notes of a levelling, and thereafter plot a section to a scale of $\frac{1}{600}$ for horizontals and $\frac{1}{120}$ for verticals.

Figure the section to show clearly the depth of cutting or filling at each successive 50 feet necessary to make a uniform gradient from A to B. The formation level at each interval is to be calculated, not measured, from the section.

Distance, Feet.	Back Sight.	Inter Sight.	Fore Sight.	Rise.	Fall.	Reduced Level.	Remarks.
← Given Details →				← Calculated Details →			
0	13·95			—		0	Station A.
50		9·71		4·24	—	4·24	
100		5·05		8·90	—	8·90	
150	13·73		0·32	13·63	—	13·63	
200		13·84		—	0·11	13·52	
250		14·55		—	0·82	12·81	
300		15·00		—	1·27	12·36	
350	14·32		11·12	2·61	—	16·24	
400		11·73		2·59	—	18·83	
450		10·65		3·67	—	19·91	
500			5·56	8·76	—	25·00	Station B.

(20)

A. *Check of calculated details :—*

Back sights total = 42·00 feet
Fore sights total = 17·00 „

Difference = 25·00 feet

The above figure agrees with the final reduced level of 25·00 feet.

Fig. 172 shows the section as plotted from the reduced levels and distances. The cutting and filling figures at the respective points are included in the diagram.

Calculation.—The uniform gradient from A to B rises 25 feet in 500 feet, or 2½ feet in 50 feet. The cutting and filling figures are obtained as follows :—

At A – 0.
„ 50 feet = 4·24 – 2·50 = 1·74 feet cutting
„ 100 „ = 8·90 – 5·00 = 3·90 „ „
„ 150 „ = 13·63 – 7·50 = 6·13 „ „
„ 200 „ = 13·52 – 10·00 = 3·52 „ „
„ 250 „ = 12·81 – 12·50 = 0·31 „ „
„ 300 „ = 12·36 – 15·00 = 2·64 feet filling
„ 350 „ = 16·24 – 17·50 = 1·26 „ „
„ 400 „ = 18·83 – 20·00 = 1·17 „ „
„ 450 „ = 19·91 – 22·50 = 2·59 „ „
„ 500 „ – 0 at B.

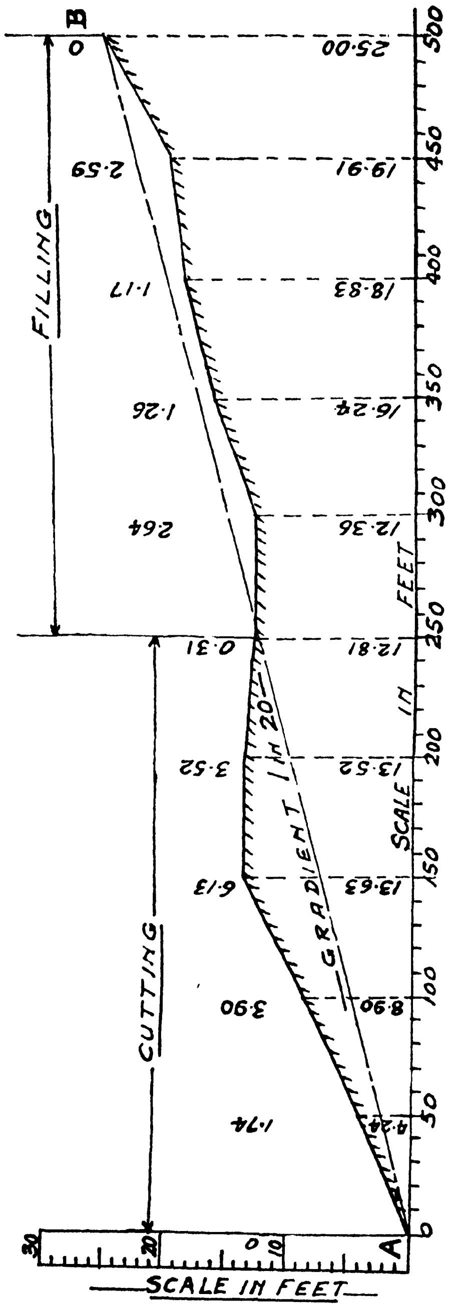


Fig. 172.—Section Plotted from Levelling Details.

Calculation of Royalty Payable from Given Data

3. The output of coal from a mine during the year has been obtained from three separate areas. Undernoted are details obtained from the plan, showing the extent of the three areas worked and the average thickness of the seam worked in each area.

Area	(1)	7·515	acres at	39·5	inches average thickness.
,,	(2)	10·310	,, ,,	36·4	,, ,, ,,
,,	(3)	1·214	,, ,,	59·1	,, ,, ,,

Royalty is payable at the rate of £2 15*s*. per inch per acre on the average thickness over the whole area worked each year. Calculate the average thickness and the amount of royalty payable for the year. (30)

A. To find average thickness of coal :—

$$7{\cdot}515 \times 39{\cdot}5 = 296{\cdot}8425$$
$$10{\cdot}310 \times 36{\cdot}4 = 375{\cdot}2840$$
$$1{\cdot}214 \times 59{\cdot}1 = 71{\cdot}7474$$

$$\text{Totals} = \underline{\underline{19{\cdot}039}} \qquad \underline{\underline{743{\cdot}8739}}$$

$$\text{Average thickness of coal} = \frac{743{\cdot}8739}{19{\cdot}039} = \underline{39{\cdot}071 \text{ inches}} \quad . \quad (1)$$

$$\text{Royalty payable for the year} = 19{\cdot}039 \times 39{\cdot}071 \times £2\ 15s.$$
$$= \underline{£2{,}045\ 13s.\ 1d.} \quad . \quad . \quad (2)$$

Plotting the Notes of a Traverse by Co-ordinates

4. The undernoted short traverse was made in a crooked underground roadway joining two shafts A and E :—

Line	*AB*.	Azimuth	210°.	Length	120 feet.
,,	*BC*.	,,	135°.	,,	405 ,,
,,	*CD*.	,,	240°.	,,	451 ,,
,,	*DE*.	,,	330°.	,,	330·5 ,,

It is proposed to connect the shafts by driving a new roadway direct from A to E. Plot the survey by co-ordinates on a scale of 1 inch = 100 feet, and calculate the length and approximate azimuth of the line AE. (30)

A. The following are the details of bearings and co-ordinates in connection with the given data.

Line.	Bearing.	Distance, feet.	Latitude (Dist. $\times \cos\theta$).	Departure (Dist. $\times \sin\theta$).	Total Latitude.	Total Departure.
AB .	S. 30 W.	120	$120 \times \frac{\sqrt{3}}{2} = 103{\cdot}92$	$120 \times \frac{1}{2} = 60{\cdot}00$	103·92 S.	60·00 W.
BC .	S. 45 E.	405	$405 \times \frac{1}{\sqrt{2}} = 286{\cdot}38$	$405 \times \frac{1}{\sqrt{2}} = 286{\cdot}38$	390·30 S.	226·38 E.
CD .	S. 60 W.	451	$451 \times \frac{1}{2} = 225{\cdot}50$	$451 \times \frac{\sqrt{3}}{2} = 390{\cdot}57$	615·80 S.	164·19 W.
DE .	N. 30 W.	330·5	$330{\cdot}5 \times \frac{\sqrt{3}}{2} = 286{\cdot}21$	$330{\cdot}5 \times \frac{1}{2} = 165{\cdot}25$	329·59 S.	329·44 W.

With the point A as origin, the co-ordinates of the point E are Latitude 329·59 South, and Departure 329·44 West.

The natural tangent of the bearing $AE = \frac{\text{Dep.}}{\text{Lat.}} = \frac{329{\cdot}44}{329{\cdot}59} = 1$ approx.

The bearing AE is therefore S. 45° W. and the azimuth of the same line is 225° . . . . (1)

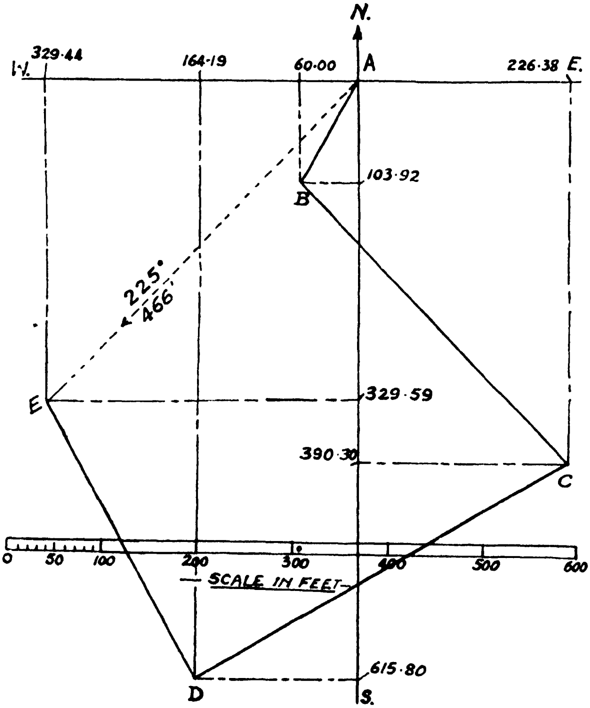


FIG. 173.—Plotting a Survey by Co-ordinates.

$$
\begin{aligned}
\text{The distance } AE &= \text{Latitude} \times \text{secant of } 45^\circ \\
&= 329{\cdot}59 \times \sqrt{2} \\
&= \underline{466 \text{ feet}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)
\end{aligned}
$$

Fig. 173 shows the plan of the survey and tie line plotted by co-ordinates to the given scale.

List of Plotting Instruments—Allowance for Change in Magnetic Meridian

5. State what instruments you would use in plotting mine surveys, and how you would allow for change in the Magnetic Meridian. How would you transfer the plotted survey from the draft to the clean plan ?

How would you observe the changes of Meridian ? If your plans were 25 years old, and a certain area of the workings was approaching the boundary line, what steps, if any, would you deem it necessary to take ? (20)

A. The instruments commonly used for plotting mine surveys are :—A protractor of good size to give direction of survey lines ; a parallel ruler, or two large set-squares, for paralleling lines across the plan ; and a good ivory or boxwood scale for measuring off the distances of the various lines.

To allow for the change in the Magnetic Meridian, the Meridian line should be drawn on the plan with a pencil, and it should be properly dated. Its position should be corrected every 2 or 3 years so as to make it coincide with the direction of the magnetic needle of the compass in use for making mine surveys.

The plotted survey is usually transferred from the draft plan to the clean plan by the use of thin carbon paper, which should not soil the latter. The new workings are thus transferred from one plan to the other by tracing them through with a hard lead pencil.

The changes of Meridian might be determined by taking readings every 2 or 3 years from a fixed surface mark to a well-defined reference mark, such as a church spire, or a lightning conductor on a chimney or other similar place. The difference in the readings divided by the number of years will give the annual variation of the magnetic needle for the instrument in use.

If the plans are 25 years old, care is necessary to check the position of the working faces when they are approaching the boundary. The chief source of error to be guarded against is that caused by shrinkage of the old plan. This should be carefully determined by a surface survey, including as many fixed points as possible on the old plan and also the boundaries. A skeleton survey might be made from the shafts to the working faces, and in plotting this survey the shrinkage allowance and correction of magnetic meridian should not be neglected. In this way the correct position of the working faces in relation to the boundaries would be determined.

Recovery of a Seam beyond a Fault

6. A roadway bearing due West on the strike of a coal seam, which dips North, strikes a fault A running due North and South with a hade of 60° to the horizontal, which throws down the seam 60 feet to the West. The strata to the East of the fault dip at 1 in 8, and to the West of it at 1 in 5.

At a point 100 yards East from A a slant road in the seam, bearing S. 45° W., leaves the level roadway, and the direction and gradient of the slant are to be maintained across the fault until the seam to the West thereof has been intersected.

Draw a plan of the roads and fault on your answer book to a scale of 1 inch = 100 feet, and find and mark the point at which the slant will strike the seam. (30)

A. Fig. 174 is the plan required by the above question; all details are included in it to show that the slant will reach the coal on the left-hand side of the fault after being driven a horizontal or plan distance of 1,131 feet, at the point marked X.

The gradient of the slant driven at S. 45° W. is $\dfrac{1}{8 \text{ sec. } 45°}$ or 1 in 11·31.

The strike lines on the left-hand side of the fault are 100 feet apart for vertical increments of 20 feet. Those on the slant road are 20 × 11·31, or 226·2 feet apart, for the same vertical increment.

The level at the point X is 200 feet in the seam and 200 feet in the slant, thus giving the point of intersection of the two.

The want of the fault at the point $A = 60 \times \tan 30° = 34{\cdot}6$ feet. The want of the fault at the point B is determined by allowing for the difference in gradients of the seam at the two sides of the fault, which are 1 in 5 and 1 in 8 respectively.

$$\therefore \text{ Want at } B = 34{\cdot}6 - \left[\left(\frac{160}{5} - \frac{160}{8}\right) \times \tan 30°\right] = 27{\cdot}6 \text{ feet.}$$

The want of the fault therefore narrows down towards the South.

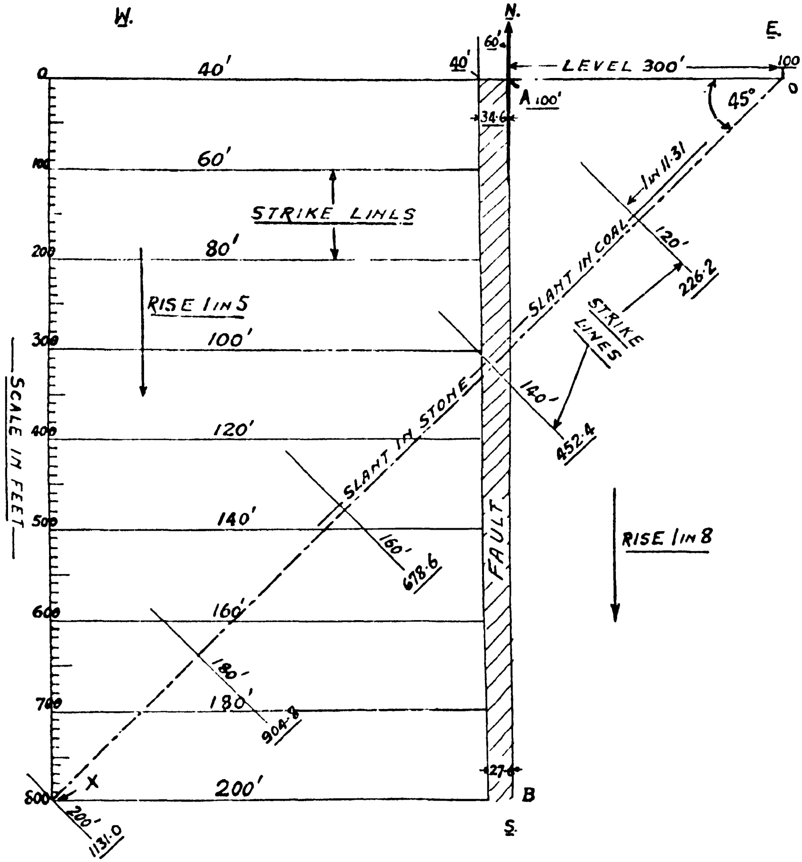


FIG. 174.—Plan showing Length of Slant Road to Intersect Seam.

NOVEMBER 1933 EXAMINATION

Outline of Area of Protection for a Property—Quantity of Coal Contained

1. Part of a colliery leasehold, in which the surface and strata are level, contains two seams of coal, known as the Upper and Lower Seams, which lie 50 yards apart.

Workings in each seam have proved the line of a large fault, as shown on the plan. The school shown thereon has to be protected against subsidence by leaving unworked the coal within such an area of the Upper Seam as lies within a lateral distance from the building equal to one-third of the depth to that seam from the surface.

From the details given on the plan, lay down thereon the margin of the protecting pillar, and calculate the quantity of coal therein. Assume the seam to be 3 feet 3 inches thick, with specific gravity 1·28. (30)

A. Fig. 175 is the plan referred to in the above question. The datum for levels is Ordnance datum. The distance between the fault lines on the plan is 100 feet, and the distance from the fault line in the lower seam to the borehole is 60 feet. From these

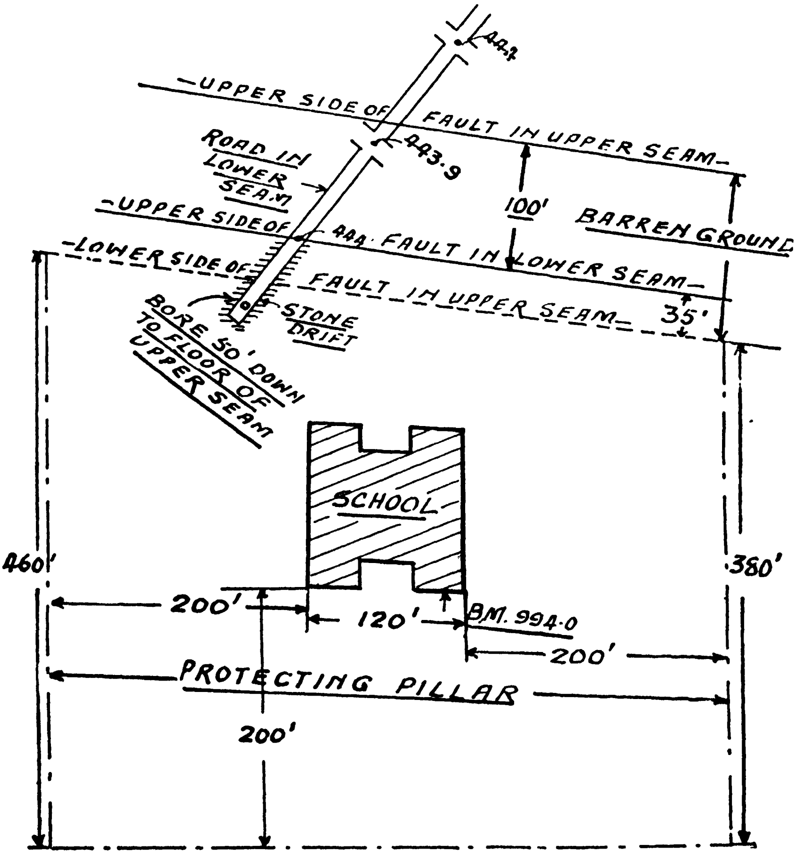


FIG. 175.—Plan showing Protecting Coal Pillar for School.

details a section through the fault can be drawn to determine the lower side of the fault in the upper seam. Fig. 176 shows such a

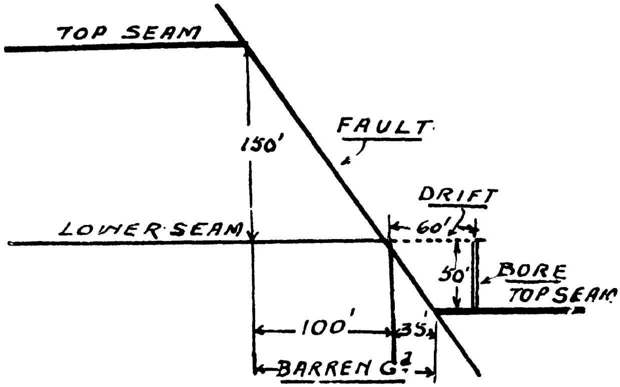


FIG. 176.—Section through Fault showing Width of Barren Ground.

section, and the required distance amounts to 35 feet, as shown.

The lower side of the fault in the upper seam, and the protecting pillar of coal for the school, are shown by dotted lines on the plan, Fig. 175. The pillar terminates at the lower side of the fault in the upper seam, owing to the barren ground, as shown on the plan.

Level of lower seam at upper side of the fault = 444 feet.
" " upper " " lower " " " " = 394 feet (by borehole).

∴ Depth of upper seam at lower side of the fault

$$= 994 - 394 = 600 \text{ feet} \quad (1)$$

$$\text{The allowance for draw} = \tfrac{600}{3} = 200 \text{ feet} \quad (2)$$

Area of protecting pillar for school

$$= \frac{460 + 380}{2} \times 520 = 218{,}400 \text{ sq. ft.} \quad (3)$$

Tons of coal contained in protecting pillar

$$= \frac{218{,}400}{43{,}560} \times 39 \times 101 \times 1{\cdot}28$$

$$= \underline{25{,}280 \text{ tons}} \quad (4)$$

Determination of Extent of Shallow Workings to be worked free from Statutory Restrictions

2. Part of a colliery leasehold has a surface composed of moss and water-bearing alluvial of considerable and variable thickness,

and workings in a seam situated at comparatively shallow depth are advancing thereto. State how you would determine the area of the seam that could be worked free from the restrictions of the General Regulations dealing with workings under moss, etc. (30)

A. The part of the leasehold with soft surface beds and water-bearing beds should be surveyed and a large-size plan prepared. Such a plan should contain contour lines in correct relation to Ordnance datum, as determined by spot-levelling at distances of about one chain apart. At the points where spot levels are taken, borings should be made to the rockhead from the surface and the depths recorded. In this way the level of the rockhead at the various points in relation to Ordnance datum could be determined, and the contour lines on the plan would have reference to the rockhead instead of the actual surface.

The same plan should contain the outline of the underground workings with correct levels in relation to Ordnance datum. The difference between underground levels and rockhead contour lines would give the thickness of solid cover over the seam at any point on the plan. When this difference is reduced to 60 feet, or 10 times the thickness of the seam, whichever is the greater in any area, the workings must be stopped so as to comply with the General Regulations of the Coal Mines Act.

To Calculate the Length of a Rising Cross-Measure Drift

3. A roadway dipping 1 in 8 in the direction of full dip of a seam strikes an upthrow fault, bearing at right-angles thereto, which is subsequently found to have thrown up the seam a distance of 90 feet, measured from floor to floor, on the angle of the hade,

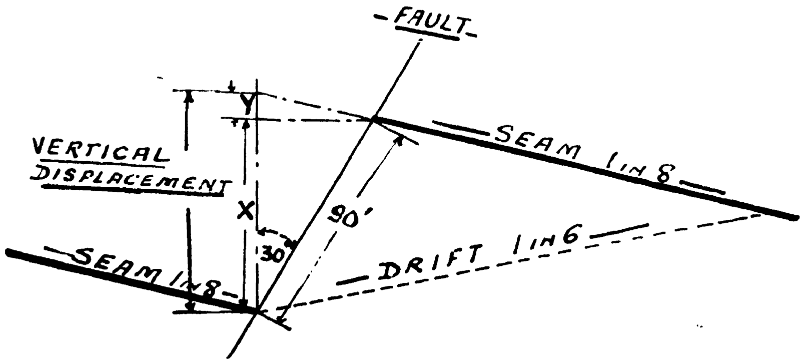


FIG. 177.—Sections of Fault, Seams and Drift.

which is 30 degrees from vertical. Calculate the length of a cross-measure drift to win the seam, commencing at the lower side of the fault and rising 1 in 6 in the same direction as the roadway. (30)

A. Fig. 177 is a section as sketched out from the given details.

$$\begin{aligned}
\text{Vertical displacement of seam by fault} &= X + Y \\
&= 90 \cos 30° + \frac{90 \sin 30°}{8} \\
&= 77{\cdot}94 + 5{\cdot}63 \\
&= 83{\cdot}57 \text{ feet} \quad . \quad . \quad (1) \\
\text{Effective grade of seam and drift} &= \tfrac{1}{6} + \tfrac{1}{8} = \tfrac{7}{24} \quad . \quad . \quad (2) \\
\text{Plan measurement of drift at 1 in 6} &= 83{\cdot}57 \times \tfrac{24}{7} \\
&= 286{\cdot}5 \text{ feet} \quad . \quad . \quad (3) \\
\text{Inclined measurement of drift at 1 in 6} &= \frac{286{\cdot}5 \times \sqrt{37}}{6} \\
&= \underline{290{\cdot}5 \text{ feet}} \quad . \quad . \quad (4)
\end{aligned}$$

Calculation of Excavation for a Pond

4. Calculate the excavation in cubic yards for a pond, the bottom area of which is 75 by 60 feet, and the angle of the slopes, one vertical to two horizontal. The original surface is level, and the depth of the pond is $10\frac{1}{2}$ feet. (20)

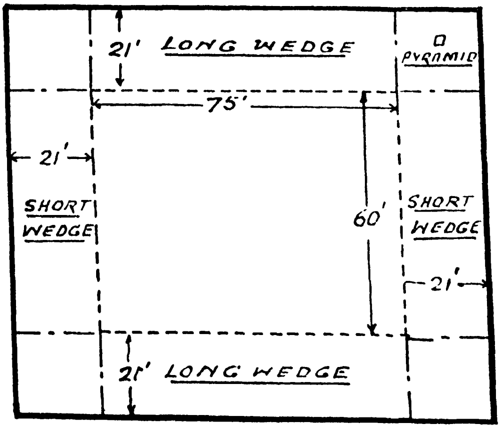


FIG. 178.—Plan of Boundaries of Pond.

A. Fig. 178 is a sketch plan showing the boundaries of the pond. The top boundary is increased by $10\frac{1}{2} \times 2$, or 21 feet, as compared with the base dimensions. The cubical capacity of the pond is most easily obtained by dividing the figure into nine parts and applying simple calculations, as follows :—

Volume of central parallelopiped $= 75 \times 60 \times 10\frac{1}{2} = 47{,}250$ cu. ft.

„ „ 2 long wedges $= 2 \times \frac{21 \times 10\frac{1}{2}}{2} \times 75 = 16{,}537$ „ „

„ „ 2 short wedges $= 2 \times \frac{21 \times 10\frac{1}{2}}{2} \times 60 = 13{,}230$ „ „

„ „ 4 square pyramids $= 4 \times 21 \times 21 \times \frac{10\frac{1}{2}}{3} = 6{,}174$ „ „

Total $= \underline{83{,}191}$ cu. ft.

Excavation in cubic yards $= \frac{83{,}191}{27} = \underline{3{,}081 \text{ cubic yards.}}$

Filling in Imaginary Staff Readings of Levelling with the Dumpy Level

5. Sight rails, or cross-heads, are to be fixed at *A* and *B*, 20 chains apart, on the line of an embankment of a colliery railway in course of construction. The formation level at *A* is 301·25, and there is a fall of 1 in 80 in the gradient from *A* to *B*. Starting from an Ordnance B.M. on a level of 284·3, with the object of taking flying levels towards *A*, successive readings of the staff are as follows :—

Back Sight.	Fore Sight.	
14·93	—	B.M. 284·3
13·99	5·88	
12·91	4·30	

What must be the next fore-sight reading on the staff held on the sight rail at *A* ? Assume you are thereafter to continue the levelling to *B*, so as to fix the cross-rail there at its proper level, and give imaginary readings which you might take in doing so. (20)

A. The following table shows the completed levelling details in connection with the above question :—

Back Sight.	Fore Sight.	Rise.	Fall.	Reduced Level.
14·93	—	—	—	*284·30* B.M.
13·99	*5·88*	9·05	—	293·35
12·91	*4·30*	9·69	—	303·04
12·30	14·70	—	1·79	*301·25* at *A*.
10·50	17·30	—	5·00	296·25
12·50	17·50	—	7·00	289·25
—	17·00	—	4·50	284·75 at *B*.

The reduced level at A is given as 301·25, and the difference between this level and the previous reduced level is 1·79. The fore sight referred to in the question must therefore be 12·91 + 1·79 = 14·70.

The reduced level at B = 301·25 at $A - \frac{20 \times 66}{80}$, or 284·75.

Bookings are filled in to obtain this figure, as shown in the above table.

MAY 1934 EXAMINATION

Calculation of True Azimuth

1. In making a correlation of underground and surface surveys at a mine by means of plumb-lines A and E hanging in separate shafts, the following data were obtained from an underground traverse made with the theodolite :—

Line.	Assumed Azimuth.	Distance, feet.
AB	150°	1,010
BC	90°	500
CD	45°	1,215
DE	300°	210·5

If the true azimuth of the line AE at the surface is 333° 31′, calculate the true azimuths of the lines in the underground traverse as accurately as the undernoted information will allow :—

TABLE OF NATURAL SINES, ETC.

Angle.	Sin.	Tan.	Cotan.	Cos.	Angle.
1°	0·0175	0·0175	57·143	0·9998	89°
2°	0·0349	0·0349	28·653	0·9994	88°
3°	0·0523	0·0524	19·084	0·9986	87°
4°	0·0698	0·0699	14·306	0·9976	86°
5°	0·0872	0·0875	11·428	0·9962	85°
	Cos.	Cotan.	Tan.	Sin.	Angle.

(30)

A. The following table shows the calculated co-ordinates of the various points in the underground traverse, from which the bearing and distance of the line connecting the two shafts is obtained.

BOOKING OF TRAVERSE SURVEY AND CO-ORDINATES.

Line.	Azimuth.	Bearing.	Distance, feet.	Latitude.	Departure.
AB	150°	S. 30° E.	1,010	$1{,}010\times\frac{\sqrt{3}}{2}=874{\cdot}7$ S.	$1{,}010\times\frac{1}{2}=505{\cdot}0$ E.
BC	90°	E.	500	—	500·0 E.
CD	45°	N. 45° E.	1,215	$1{,}215\times\frac{1}{\sqrt{2}}=859{\cdot}1$ N.	$1{,}215\times\frac{1}{\sqrt{2}}=859{\cdot}1$ E.
DE	300°	N. 60° W.	210·5	$210{\cdot}5\times\frac{1}{2}=105{\cdot}3$ N.	$210{\cdot}5\times\frac{\sqrt{3}}{2}=182{\cdot}3$ W.
			Totals	964·4 N. 874·7 S.	1,864·1 E. 182·3 W.
			Co-ordinates of E = Difference	89·7 N.	1,681·8 E.

Natural tangent of bearing of line $AE=\frac{\text{Departure}}{\text{Latitude}}=\frac{1681{\cdot}8}{89{\cdot}7}=18{\cdot}75.$

$\therefore$ Bearing of line AE = N. 86° 57′ E. [87° from given table.]
and Azimuth of line AE = 86° 57′.

Difference between surface and underground lines connecting the two shafts = $(360° - 333°\ 31') + 86°\ 57' = 113°\ 26'$.

True azimuth of line AB = $150° - 113°\ 26' = 36°\ 34'$.

" " " " BC = $90° - 113°\ 26' + 360° = \underline{336°\ 34'}$.

" " " " CD = $45° - 113°\ 26' + 360° = \underline{291°\ 34'}$.

" " " " DE = $300° - 113°\ 26' = \underline{186°\ 34'}$.

Determining Tail of Slopes of Debris Heap

2. The accompanying plan shows an area of ground with contour lines which is about to be used as a dump for colliery debris. It is proposed to deposit the debris by means of a self-tipping hopper operating on a rail-track with 5-foot gauge and rising 1 in 2½ from the point A on the line AB. Show on the plan, by a dotted line, the tail of the slopes of the debris heap to the North of the line CD when the rail incline has reached a point 215 feet to the rise of the starting-point A. Assume the angle of repose of the material to be 40 degrees. (30)

A. Fig. 179 is the contour plan referred to in the above question, the tail of the slopes of the heap being shown by the dotted line.

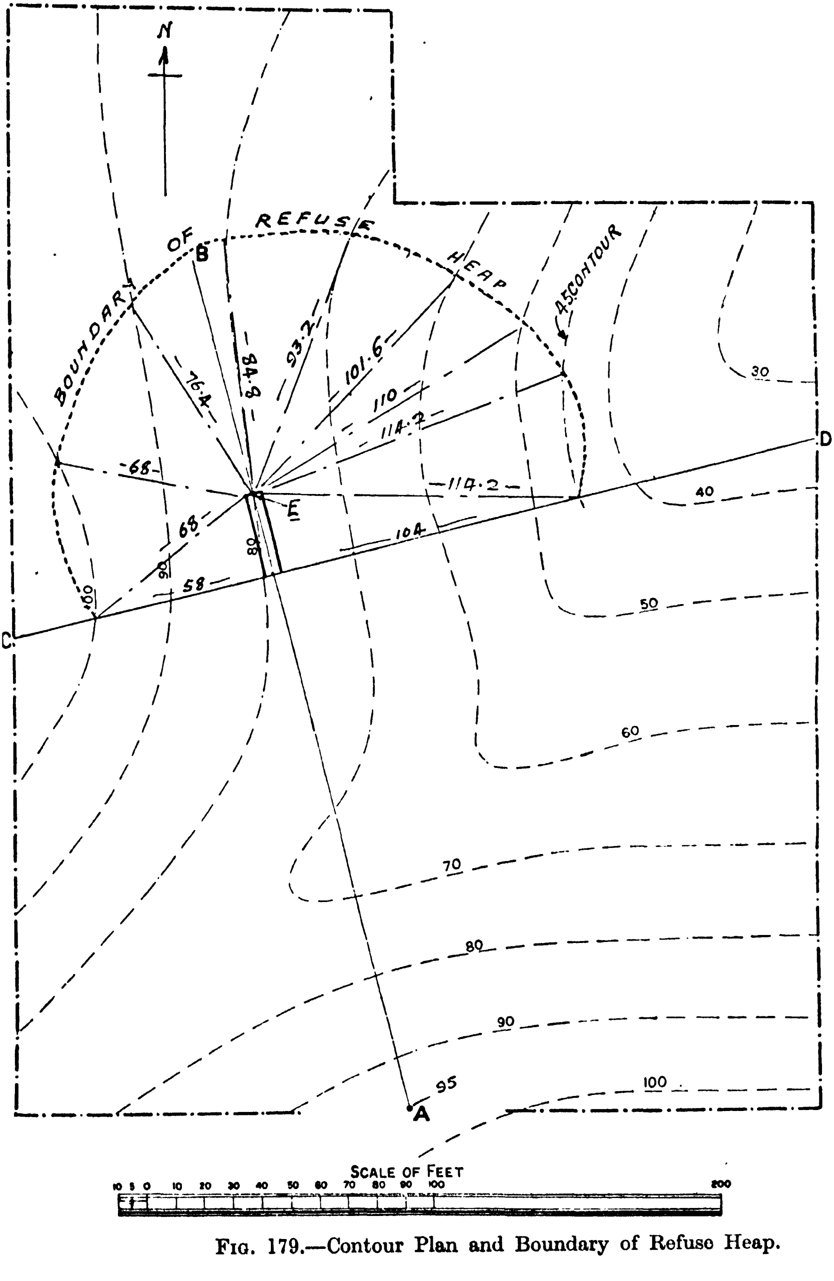


FIG. 179.—Contour Plan and Boundary of Refuse Heap.

Assuming that the 215 feet is plan measurement, the terminus of the incline is at the point E, vertically above the 80-foot contour line.

Height of the point E above datum

$$=\text{Height at } A\ (95)+\frac{215}{2\frac{1}{2}}=181 \text{ feet.}$$

Tail of slope at 100 contour line	$=(181-100)$	$\times \tan 40°$	$=68$ feet.
,, ,, ,, ,, 90 ,,	,, $=(181-90)$	,,	$=76{\cdot}4$,,
,, ,, ,, ,, 80 ,,	,, $=(181-80)$	,,	$=84{\cdot}8$,,
,, ,, ,, ,, 70 ,,	,, $=(181-70)$	,,	$=93{\cdot}2$,,
,, ,, ,, ,, 60 ,,	,, $=(181-60)$	,,	$=101{\cdot}6$,,
,, ,, ,, ,, 50 ,,	,, $=(181-50)$	,,	$=110{\cdot}0$,,
,, ,, ,, ,, 45 ,,	,, $=(181-45)$	,,	$=114{\cdot}2$,,

The above distances might also be determined by plotting the heights to a suitable scale, as shown in Fig. 180.

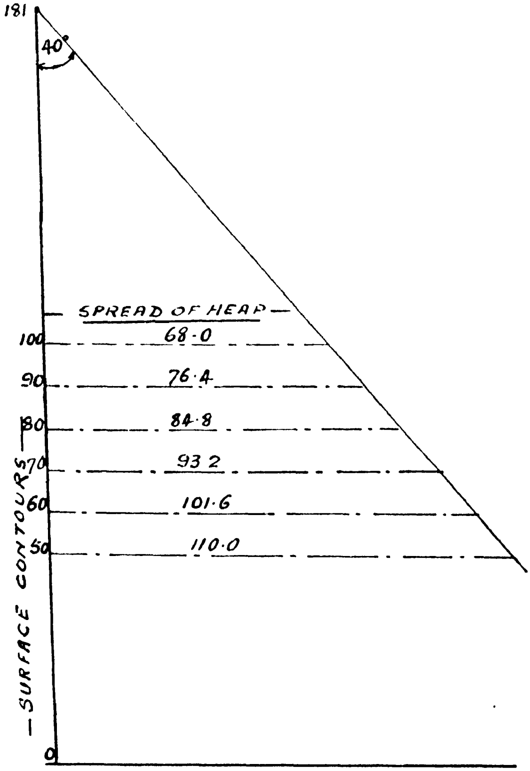


Fig. 180.—Section determining Boundary of Refuse Heap.

To obtain the correct termination of the slopes at the line CD, the height of the heap at that point is $95 + \frac{185}{2\frac{1}{2}} = 169$ feet.

$\therefore$ Tail of slope at 100 contour line $= (169-100) \times \tan 40° = 58$ feet.

„ „ „ „ 45 „ „ $= (169-45)$ „ $= 104$ „

Calculating the Area of a Triangular Enclosure

3. The azimuths of the three sides of a triangular enclosure, of which the side AC measures 940 feet, are respectively as follows :—

$$\begin{aligned} AB &= 347° \ 25' \ 30'' \\ BC &= 77° \ 25' \ 30'' \\ CA &= 227° \ 25' \ 30'' \end{aligned}$$

Calculate the length of the sides AB and BC, and the area of the enclosure in acres, roods and poles. (20)

A. By drawing out a diagram roughly, and marking in the given details, the following results are easily obtained :—

$$\begin{aligned} \text{Angle at } B &= 360° - 347° \ 25' \ 30'' + 77° \ 25' \ 30'' = 90°. \\ \text{,, ,, } C &= 77° \ 25' \ 30'' + 180° - 227° \ 25' \ 30'' = 30°. \\ \text{,, ,, } A &= (360° - 347° \ 25' \ 30'') - (227° \ 25' \ 30'' - 180°) = 60°. \end{aligned}$$

$$\left.\begin{aligned} &\text{Side } BC : \frac{BC}{\sin 60°} = \frac{940}{\sin 90°}, \text{ and } BC = 940 \times \sin 60° = \underline{814 \text{ feet}} \\ &\text{Side } AB : \frac{AB}{\sin 30°} = \frac{940}{\sin 90°}, \text{ and } AB = 940 \times \sin 30° = \underline{470 \text{ feet}} \end{aligned}\right\} \quad . \quad (1)$$

$$\text{Area of enclosure} = \frac{814 \times 470}{2} = 191{,}290 \text{ sq. ft.}$$

$$\text{,, ,, ,,} \quad = \frac{191{,}290}{43{,}560} \text{ ac.} = \underline{4 \text{ ac. } 1 \text{ rd. } 22{\cdot}6 \text{ poles}} \quad . \quad (2)$$

Calculating Volume of an Irregular Heap

4. The undernoted figure represents the plan of a heap of small coal deposited on level ground which is 100 feet above datum. The figures on top of the heap are spot levels stated in feet above the same datum. Transfer the figure to your answer book by pricking through, and thereafter calculate the contents of the heap in cubic yards. (30)

A. Fig. 181 is the plan referred to in the above question; letters of notation and dimensions have been added to it. The heights given are underlined.

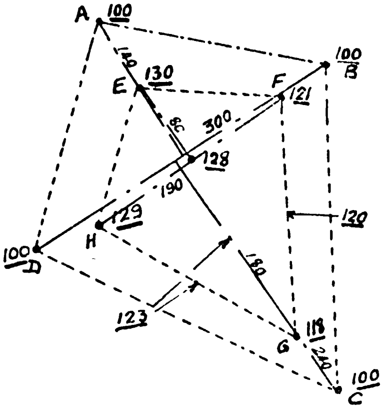


FIG. 181.—Plan of Coal Heap.

Area of ⏢ $ABCD$ at base of heap $= 300\left(\dfrac{140+240}{2}\right)$

$=57{,}000$ sq. ft.

Area of ⏢ $EFGH$ at top of heap $= 190\left(\dfrac{80+180}{2}\right)$

$=24{,}700$ sq. ft.

Mean area of heap $= \dfrac{57{,}000 + 24{,}700}{2} = 40{,}850$ sq. ft. . (1)

Mean height of heap $= \dfrac{30 + 21 + 20 + 18 + 23 + 29 + 28 + 23}{8}$

$= 24$ feet . (2)

Cubic contents of heap $= \dfrac{\text{Mean area} \times \text{Mean height}}{27}$

$= \dfrac{40{,}850 \times 24}{27} = \underline{36{,}310 \text{ cubic yards}}$. (3)

NOVEMBER 1934 EXAMINATION

Protecting Pillar for Railway Viaduct

1. The accompanying plan shows workings in two seams situated 30 yards vertically apart in the strata, which are brought

opposite each other at the point A by a fault marked AB. The railway viaduct shown to the South of the workings is to be protected against subsidence by leaving unworked such an area of coal in each seam as lies within a lateral distance from the viaduct equal to one-third of the depth thereto from the surface.

Draw the outline of the protecting barrier for the upper seam on the plan. The surface level at the viaduct is 989·0 above Ordnance datum, and the hade of the fault is 60 degrees. (30)

A. Fig. 182 is the plan referred to above, and the figures given

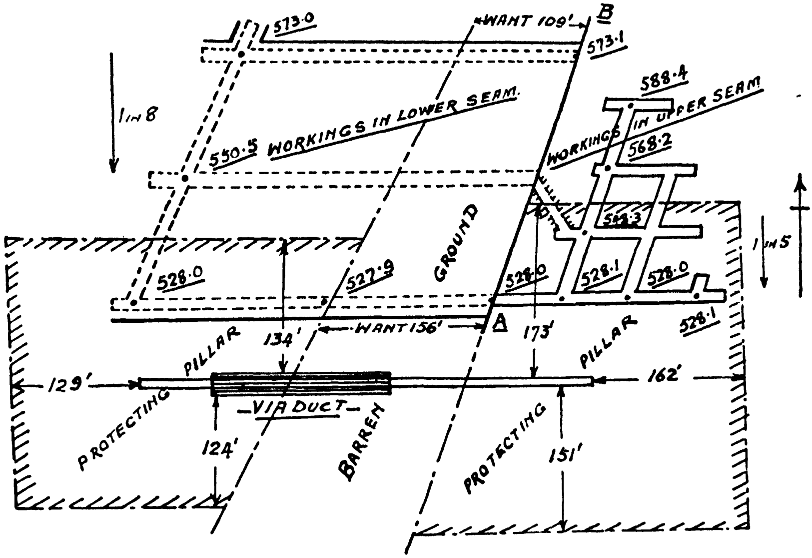


FIG. 182.—Plan of Workings showing Protecting Barrier for Viaduct.

are underlined. The details of fault, barren ground, and protecting pillar are shown on the plan by broken lines.

Full dip of workings in the lower seam West of the fault

$$= \frac{573 - 528}{360}, \text{ or 1 in 8.}$$

Level of seam under viaduct $= 528 - \frac{130}{8} = 513$ ft. lower seam.

" " " " " $= 513 + 90 = 603$ ft. upper seam.

" " viaduct at the surface $= 989$ ft.

∴ Allowance for draw in upper seam West of the fault

$$= \frac{989 - 603}{3} = 129 \text{ feet} \quad . \quad . \quad . \quad (1)$$

Full dip of workings in the upper seam East of the fault

$$=\frac{588\cdot4-528\cdot1}{300}, \text{ or } 1 \text{ in } 5.$$

Level of upper seam under viaduct

$$=528-\tfrac{120}{5}=504 \text{ feet.}$$

$\therefore$ Allowances for draw in upper seam East of the fault

$$=\frac{989-504}{3}=162 \text{ feet} \quad . \quad (2)$$

Width of pillar North-west of viaduct $=129+\frac{129}{8\times3}=134$ feet . (3)

,, ,, ,, South-west ,, ,, $=129-\frac{129}{8\times3}=124$ feet . (4)

,, ,, ,, North-east ,, ,, $=162+\frac{162}{5\times3}=173$ feet . (5)

,, ,, ,, South-east ,, ,, $=162-\frac{162}{5\times3}=151$ feet . (6)

The upper limb of the fault in the upper seam is shown on the plan to the West of the line *AB*. A section through the fault and seams, drawn in pencil, greatly assists in fixing the line.

Width of the fault at the point *A*

$$= 90 \text{ feet throw} \times \tan 60^\circ = 156 \text{ feet.}$$

Width of the fault at the point *B*

$$= 156 - \left[\left(\frac{360}{5}-\frac{360}{8}\right)\times\tan 60^\circ\right] = 109 \text{ feet.}$$

The fault thus widens towards the South owing to the difference in gradient of the seams at the respective sides of the fault.

The barren ground in the upper seam is indicated on the plan.

Contour Lines, their Uses, and how to Lay Them off

2. What are contour lines and what are their uses? Describe shortly how you would lay off, and subsequently show on a plan, contour lines of an area of land at intervals of 5 feet. (30)

A. A contour line is an irregular-directioned line, marked on a plan, which has the same height above or below datum throughout the whole extent of its path.

Contour lines on plans may be used for obtaining sections in any given direction to obtain details of cutting and banking in connection with pipe-lines, drains, railways, and other surface constructions. The general surface line of the area covered by the plan can easily be traced out. Geological sections showing beds, inclination, and interruptions can be constructed from geological maps containing contour lines. The thickness of cover over a coal seam can be determined from underground levels and surface contour lines on the plan.

In the process of contouring a plan, the area might be surveyed and divided into suitable squares. Readings of a staff are taken by means of a dumpy level. The reduced levels obtained, in correct relation to Ordnance datum, are known as "spot levels," and these can be used for putting the contour lines on the plan.

In highly inclined and irregular ground, where the spots are closer together, use might be made of an improved type of levelling instrument with a transit telescope, or a transit theodolite. In this way both angles and staff readings are taken, and the progress of the work is greatly facilitated.

Constructing a Scale

3. A distance of 1·4375 miles is represented on a map by a length of 4 inches. Construct a scale of chains for the map by which single chains may be measured. Show a total length of 150 chains. Show all calculations, figure the scale properly, and write on it the representative fraction. Indicate also by two small dots the points you would take on the scale to measure off a distance of 67 chains. (20)

A. $1{\cdot}4375 \times 80 = 115$ chains.

$\therefore$ 4 inches on the map represents 115 chains.

Fig. 183 (page 362) shows the method of constructing the scale of chains referred to in the question.

Take a line AB exactly 4 inches long and draw perpendicular lines at the ends of this line, as shown in the diagram. Construct a diagonal line, say CD, between the perpendiculars, exactly 11·5 parts in length, and extend it to contain 16 parts, thus terminating in E.

The scale is formed by paralleling the dots marked 1 . . . 16 upon the line forming the edge of the scale. Each division on the scale will thus represent 10 chains. The first division should be subdivided into 10 parts so as to give single chains.

The representative fraction is $\dfrac{1}{\dfrac{115}{4} \times 792}$ or $\dfrac{1}{22{,}770}$.

67 chains are indicated on the scale.

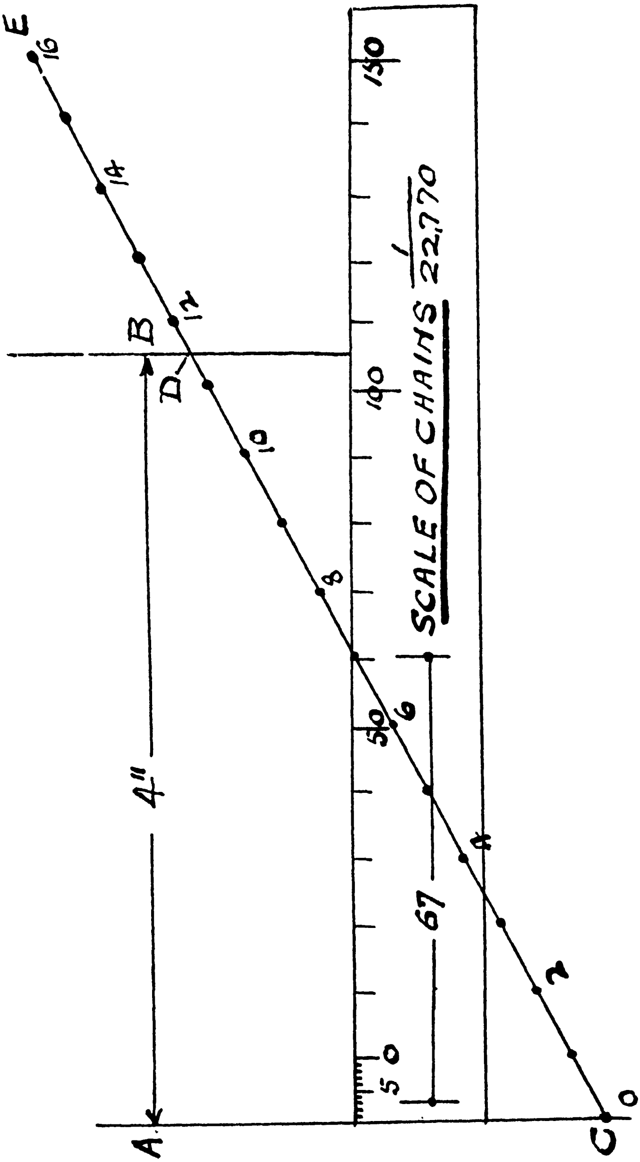


FIG. 183.—Construction of a Scale of Chains.

Laying-off a Haulage Curve

4. A main haulage roadway in a mine is about to be driven forward from its present termination on a curve of 100 feet radius, to which the existing straight road will be a tangent. State how you would proceed to lay-off the new drift which is to be driven for this purpose so as to maintain a true curve through an angle of 90 degrees. (20)

A. Length of curve $= 2\pi r \times \frac{90}{360} = \frac{44}{7} \times 100 \times \frac{1}{4} = 157$ feet.

A practical and easy method of driving an underground drift is by chords and chords produced of a selected length, and measuring small arcs, as offsets, at the end of each chord, so as to give proper direction. Such a method is illustrated in Fig. 184 (p. 364).

Assuming tangents and chords 12 feet long, there will be $\frac{157}{12} = 13$ offsets inside the curve. Starting at the point a, the first offset $b\,c = \frac{\text{tangent}^2}{2 \times \text{radius}} = \frac{12^2}{200} = 0{\cdot}72$ feet, or 8·64 inches. The remaining or intermediate offsets, $d\,e$, $f\,g$, etc., inside the curve, are between chords and are therefore $\frac{(\text{chord})^2}{\text{radius}} = \frac{12^2}{100} = 1{\cdot}44$ feet or 17·3 inches. The last offset, $h\,j$, will be outside the curve and will be between a tangent and a chord, as in the case of the first offset. This will put the road on its proper bearing, after the curve is finished, at 90 degrees to the direction at the start.

In driving the curve, $a\,b = 12$ feet, and the arc or offset $b\,c = 8{\cdot}64$ inches, to give a centre-line for the curve.

For the second arc or offset $d\,e$, the chord $a\,c$ is produced to d, so that $c\,d$ is 12 feet, and the arc or offset $d\,e$ is made 17·3 inches to give the centre-line of the curve. This process is repeated until the remaining 11 offsets are laid-off to finish the curve.

The position of the last offset, $h\,j$, after finishing with the curve, is clearly shown on the diagram.

Calculating the Length of Cross-measure Drifts

5. The direction and rate of full dip of two seams 50 yards vertically apart from floor to floor are N. 10° E. and 6 inches to the yard, respectively. Calculate the length of a cross-measure drift driven from the lower seam to intersect the upper, and bearing N. 20° W., (*a*) if the drift be level, (*b*) if it rise at a gradient of 3·6 inches to the yard. (30)

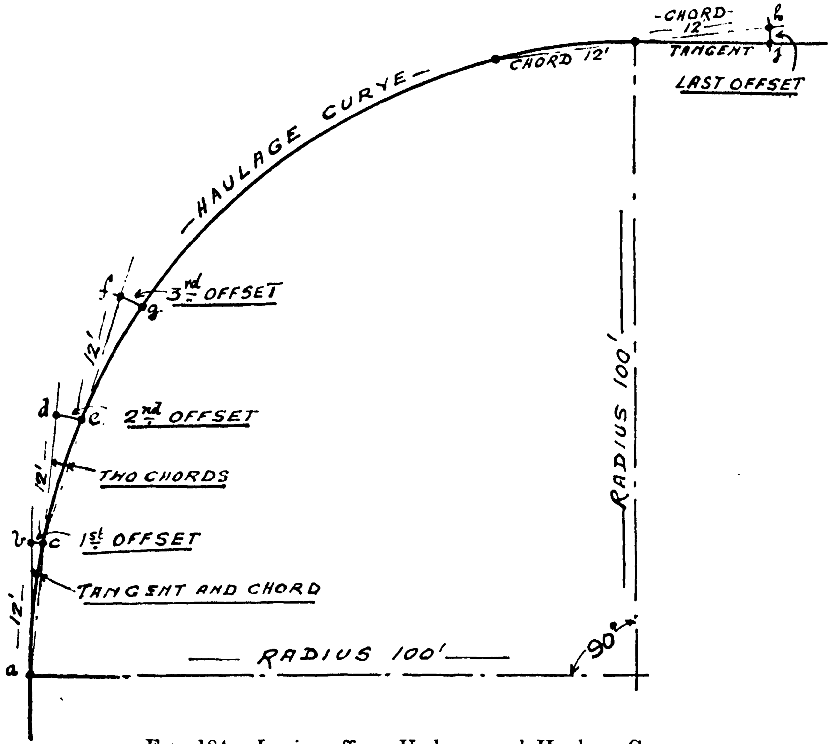


FIG. 184.—Laying-off an Underground Haulage Curve.

A. Fig. 185 is a sketch including the details of the above question. The full dip of the seams is 6 inches per yard, or 1 in 6, in direction N. 10° E. The dip of the seams in the direction of the drifts, or N. 20° W., is

$$\frac{1}{6 \times \sec 30^\circ}, \text{ or } \tfrac{1}{7}, \text{ or } 1 \text{ in } 7 \quad . \quad . \quad . \quad (1)$$

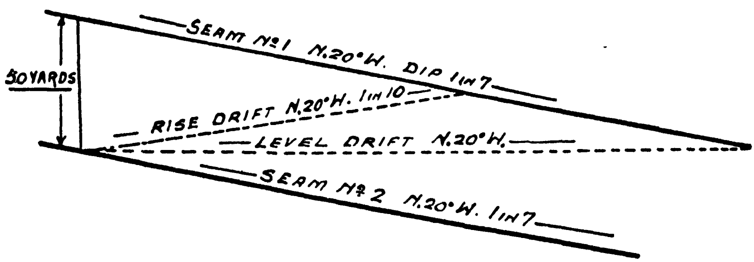


FIG. 185.—Sketch Section of Seams and Drifts.

The vertical distance between the seams is 50 yards.

$\therefore$ Length of a level drift $= 50 \times 7 = \underline{350 \text{ yards}}$. (2)

and length of drift rising 3·6 in 36, or 1 in 10

$$= 50\left(\frac{1}{\frac{1}{7} + \frac{1}{10}}\right)$$

$$= 50 \times \tfrac{70}{17}$$

$= 206$ yards plan measure . . . (3)

Actual length of rise drift at 1 in 10

$$= \frac{206 \times \sqrt{101}}{10} = \underline{207 \text{ yards}} \quad . \quad . \quad . \quad (4)$$

JULY 1940 EXAMINATION

(*Five questions only to be answered.*)

Calculation of Bearing and Distance of a New Road

1. The following are the notes of an underground traverse made with the theodolite in part of a main airway in a level seam :—

Line.	Azimuth.	Distance (ft.).
AB	271° 30′	400
BC	184° 48′	550
CD	120° 00′	490
DE	238° 30′	781

It is proposed to improve the ventilation by driving a new roadway in a direct line between the points *A* and *E*, and in order to complete the work with the maximum of speed, to drive simultaneously on four separate faces. Calculate the position of a point *F* on the line *CD* at which two faces are to be set out, and state the quadrant bearing of *FE*. (40)

A. The point *F* on the line *CD* has to be found by calculation, and for this purpose the co-ordinates of the traverse are most helpful. It is therefore desirable to calculate the co-ordinates, as given below, and from them to plot out the traverse for reference before starting with the calculation for the point *F*.

BOOKING OF TRAVERSE SURVEY AND CO-ORDINATES.

Line.	Azimuth.	Bearing.	Distance.	Latitude. N.	Latitude. S.	Departure. E.	Departure. W.	Total Latitude.	Total Departure.
AB	271° 30′	N. 88° 30′ W.	400 ft.	10·47	—	—	399·90	10·47 N.	399·90 W.
BC	184° 48′	S. 4° 48′ W.	550 ,,	—	548·10	—	46·03	537·63 S.	445·93 W.
CD	120° 00′	S. 60° 00′ E.	490 ,,	—	245·00	424·30	—	782·63 S.	21·63 W.
DE	238° 30′	S. 58° 30′ W.	781 ,,	—	408·20	—	666·10	1190·83 S.	687·73 W.
			Totals	10·47	1201·30	424·30	1112·03		

Difference: 1190·83 S. 687·73 W.

Fig. 186 shows the plan of the traverse plotted from the above total co-ordinates.

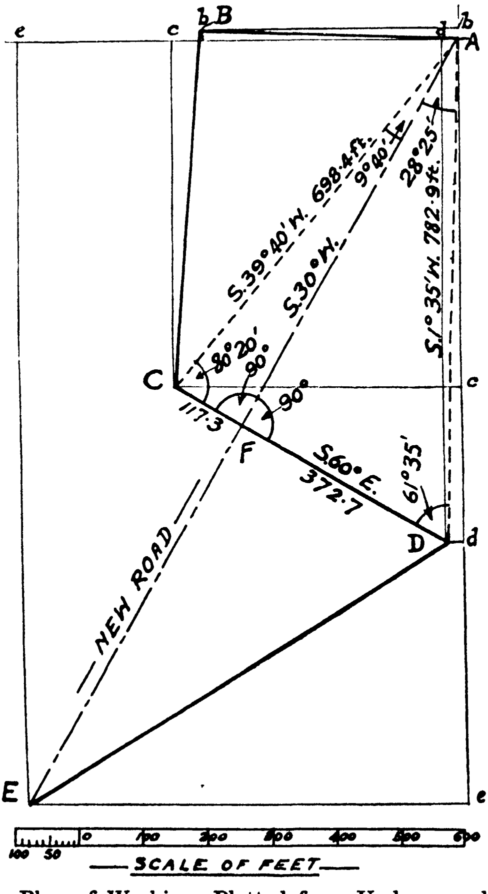


FIG. 186.—Plan of Workings Plotted from Underground Traverse.

Calculations for the point F :—

$$\text{Natural Tangent of Bearing } AE = \frac{\text{Diff. in departure}}{\text{Diff. in latitude}} = \frac{687\cdot73}{1190\cdot83} = 0\cdot5775.$$

$\therefore$ AE has a bearing of S. 30° W. . . . . (1)

$$\text{Natural Tangent of Bearing } AC = \frac{445\cdot93}{537\cdot63} = 0\cdot8294.$$

$$\left.\begin{aligned} &\therefore\ AC \text{ has a bearing of S. } 39°\ 40' \text{ W.} \\ &\text{and } AC = \text{Latitude} \times \text{secant of } 39°\ 40' \\ &\qquad = 537\cdot63 \times \frac{1}{0\cdot769} = 698\cdot4 \text{ ft.} \end{aligned}\right\} \quad . \quad . \quad (2)$$

$$\text{Natural Tangent of Bearing } AD = \frac{21\cdot63}{782\cdot63} = 0\cdot0276.$$

$$\left.\begin{aligned} &\therefore\ AD \text{ has a bearing of S. } 1°\ 35' \text{ W.} \\ &\text{and } AD = 782\cdot63 \times \text{secant of } 1°\ 35' \\ &\qquad = 782\cdot63 \times \frac{1}{0\cdot9996} = 782\cdot9 \text{ ft.} \end{aligned}\right\} \quad . \quad . \quad (3)$$

In the triangle AFC, $F\hat{A}C = 9°\,40'$; $F\hat{C}A = 80°\,20'$; and $A\hat{F}C = 90°$.
In ,, ,, AFD, $D\hat{A}F = 28°\,25'$; $A\hat{D}F = 61°\,35'$; and $A\hat{F}D = 90°$.

$$\begin{aligned} \therefore\quad CF &= 698\cdot4 \times \sin 9°\ 40' \\ &= 698\cdot4 \times 0\cdot1680, \text{ or } 117\cdot3 \text{ ft.} \\ \text{and } FD &= 782\cdot9 \times \sin 28°\ 25' \\ &= 782\cdot9 \times 0\cdot4759, \text{ or } 372\cdot7 \text{ ft.} \end{aligned} \quad . \quad . \quad (4)$$

The quadrant bearing of the line FE is S. 30° W. . (5)

Drawing a Section of a Cross-Measure Drift from a Plan and Levels

2. The accompanying plan shows an area of workings which have reached a large fault with displacement up to West. It also shows a cross-measure drift AB which has been driven to form a landing. It is proposed to continue the latter drift in the direction shown on the plan, but at such a gradient from B as will enable it to intersect the seam on the West side of the fault near the point C. Draw in your answer book, from the data shown on the plan, a section of the proposed cross-measure drift ABC and a roadway driven in continuation thereof in the seam to about the point D. Assume the angle of the fault-plane to be 33 degrees from the vertical, and that the inclination of the seam to North is as indicated by the levels shown on plan. (30)

A. Fig. 187 shows the plan of workings referred to in the above question and also the section of the proposed cross-measure drift required.

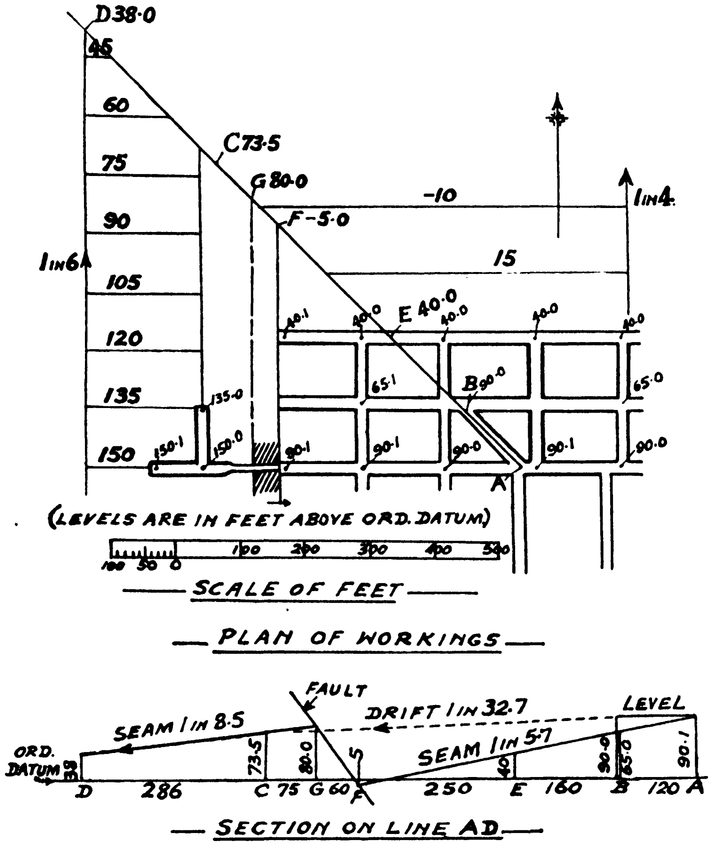


FIG. 187.—Plan of Workings and Section of Proposed Cross-Measure Drift.

From the plan details, the full dip of the seam is 50 feet in 200 feet, or 1 in 4 due N., at the East side of the fault. The dip in the line of the proposed drift at a bearing of N. 45° W. is 1 in $\sqrt{16 + 16}$, or 1 in 5·7. The full dip, assumed to be due N. at the West side of the fault, is 15 feet in 90 feet, or 1 in 6. The dip of the seam in the line of the proposed drift at N. 45° W. is therefore 1 in $\sqrt{36 + 36}$, or 1 in 8·5.

The following levels and distances can be picked off the given plan for the plotting of the required section (the strike lines and their heights in bold type have been added to the plan):—

Levels at *A* are	Seam 90·1,	Drift 90·1		
,, ,, *B* ,,	,, 65·0	,, 90·0,	Distance *AB*	120 ft.
,, ,, *E* ,,	,, 40·0	,, —	,, *BE*	160 ,,
,, ,, *F* ,,	,, − 5·0	,, —	,, *EF*	250 ,,
,, ,, *G* ,,	,, 80·0	,, —	,, *FG*	60 ,,
,, ,, *C* ,,	,, 73·5	,, 73·5	,, *GC*	75 ,,
,, ,, *D* ,,	,, 38	,, —	,, *CD*	286 ,,

The drift from the point *B* to the point *C* dips at 1 in 32·7, and the seam forming the continuation of the drift dips at 1 in 8·5.

Surveying of Workings nearing the Leasehold Boundary

3. An exploring heading in a certain seam is believed to have almost reached the leasehold boundary, and it is desired to determine accurately the position of the face of the heading in relation thereto. The seam is being worked by a drift 600 feet in length, dipping into the measures from the surface at a gradient of 12 inches to the yard, and the heading face is about half a mile distant from the mouth of the drift. Not more than six settings of an instrument will be necessary in making a traverse between the mouth of the drift and the heading face. State fully how you would carry out the necessary work so as to obtain the maximum accuracy, and illustrate your answer by a sketch. (30)

A. For the conditions stated in the above question it would be advisable to use a theodolite for both the surface and underground surveys, as accuracy is important and can be guaranteed by this method if a qualified mine surveyor is employed.

Starting at a convenient point on the boundary at the surface, to include as much of it as possible, a theodolite survey should be begun and continued down the drift, dipping at 1 in 3 to the face of the heading underground which is approaching the boundary, as shown in the sketch plan, Fig. 188.

The instruments required would be a theodolite reading to 20 seconds, with an accurate centring device; three sets of tripods for the instrument, with levelling device; a chain 100 links long (tested); two plumb-lines; chalk; and safety-lamps or other device to fit the top of the tripod. Set up the instrument at the point *B*, and take the magnetic bearing of the line *AB*. With instrument at B, take the horizontal angle ABC in clockwise direction, and measure the lines AB, BC. With the instrument at the point C and with BC as base, take the horizontal angle BCD in clockwise direction to the top of the drift ; measure the line CD.

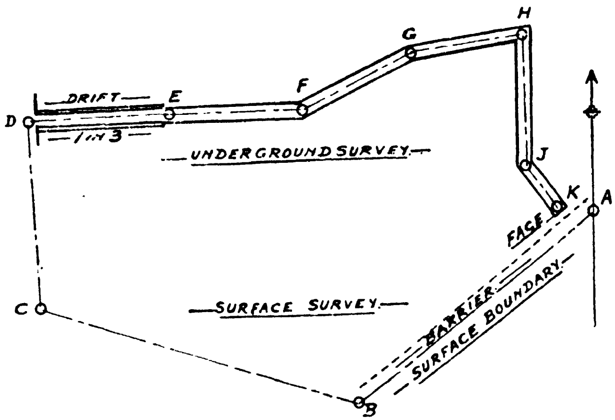


FIG. 188.—Sketch Plan showing Surface Boundary and Underground Workings.

Remove the instrument to the point D, and with the line CD as base, take the horizontal angle CDE and angle of inclination of the drift, and measure DE on the slope. Continue in this way until the point K is reached. When the survey is finished, the co-ordinates should be calculated and plotted to the scale of the plan, thus giving the relative position of the boundary and the face of the heading K. As this is not a tied survey, great care must be taken to have all the angles and distances carefully checked when doing the work.

The following table shows the method of booking the survey, using imaginary figures for a part of the traverse only :—

Line.	Exterior Angle.	Interior Angle.	Azimuth.	Bearing.	Plotting Distance.	Remarks.
					Chains.	
AB	—	—	230° 00′ 00″	S. 50° 00′ 00″ W.	20·63	Clear bearing.
BC	245° 15′ 20″	114° 44′ 40″	295° 15′ 20″	N. 64° 44′ 40″ W.	21·72	
CD	260° 35′ 40″	99° 24′ 20″	15° 51′ 00″	N. 15° 51′ 00″ E.	12·67	*D* is at top of drift.
DE	272° 12′ 20″	87° 47′ 40″	108° 3′ 20″	S. 71° 56′ 40″ E.	8·60	9·10 chains at 19°; *E* is at foot of drift.
EF	179° 0′ 40″	180° 59′ 20″	107° 4′ 00″	S. 72° 56′ 00″ E.	0·25	Underground road.

Calculation of Thickness of Seam and Breadth of Outcrop from Boring Details

4. The floor of a seam of limestone is cut by a vertical bore at a depth of 75 yards from the surface. The core of limestone taken from the bore is 20 feet long, and shows the strata to dip at 33 degrees. If the surface rises uniformly at 1 in 12 in the same direction as the strata, calculate (*a*) the breadth of the seam at the outcrop, measured on the slope of the rockhead, assuming that a thickness of 10 feet of alluvium overlies the rockhead ; and (*b*) the thickness of the limestone seam, measured at right-angles to the plane of stratification. (20)

A. Fig. 189 (page 372) is a section showing the details as given in the above question. The enlarged diagram shows the following dimensions :—

In triangle ACD, the angle ACD is 33°, and as the angle A is 90°, the angle CDA is 57°. Because AD measures 20 feet, then AC is 20 tan 57°, or 30·8 feet . . . . . . (1)

In triangle ABC, the tangent of the angle CAB is 1/12 ; therefore $\angle CAB$ is 4° 45′, $\angle ACB$ is 147°, and $\angle ABC$ is 28° 15′.

By the sine rule, AB is $\dfrac{30\cdot 8 \times \sin 147°}{\sin 28° \ 15'}$ or 35·45 feet.

The width of the limestone seam at the rockhead outcrop is therefore 35·45 feet . . . . (2)

The thickness of the seam (t) at right-angles to the plane of stratification is 20 feet × cos 33°, or 16·77 feet . . (3)

Surveying of a Road which has to be Reconditioned

5. A slightly undulating and moderately straight underground roadway, about 8 feet in width and 400 yards in length, in which a main-and-tail haulage operates, is about to be reconditioned by the erection of steel arches, and to be widened to give a finished width of 12 feet prior to the installation of an endless haulage. The roadway is to be made straight throughout its entire length, and to be re-graded so as to give a uniform inclination from end to end. Describe fully the various steps which you would take to give effect to the requirements above stated. (20)

A. The procedure in dealing with the above conditions depends very much on the straightness of the road. If it is possible to get

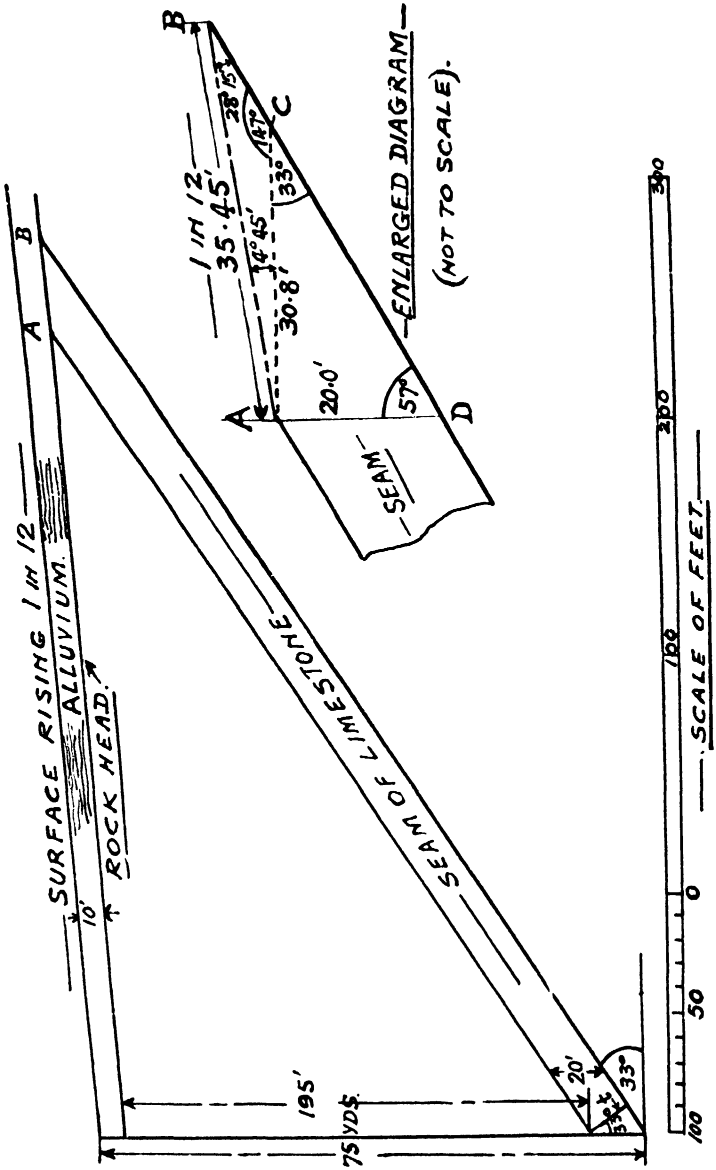


FIG. 189.—Section illustrating Question 4.

a straight line through the entire length, the survey might be carried out by using the miners' compass and plumb-lines, or possibly the theodolite and plumb-lines. In the event of the above methods not being possible, the miners' compass might be used for making an accurate survey of the road, offsets to the road sides being taken regularly throughout the survey.

When these details are complete, a large-scale plan of the road can be made, so that a design of the centre-line can be prepared. It might be possible to take most of the cutting from one side of the road. After the sides of the road are complete, the dumpy level can be used to obtain a section along the centre-line of the new road. When this section is plotted to scale, a design of gradient can be made and the adjustment of gradient carried out.

Correcting the Error of a Vernier Survey

6. You are asked to carry out a short traverse in a level but crooked roadway, using a dial with vernier attachment. You have also had definite instructions to start and finish your traverse with a free bearing. If, on completing the traverse, you found that your final free bearing was 35 minutes in excess of the vernier azimuth reading, and that a check survey could not conveniently be made, how would you amend your vernier bearings before plotting them ? Illustrate your answer by a sketch, and assume that five settings of the instrument were necessary in the traverse between the points at which the first and final magnetic bearings were obtained. (20)

A. The free bearing at the beginning of a vernier survey is very important, and great care should be exercised when taking it to have the compass clamps in good working order, and also to make sure that the needle is not attracted in any way. If the bearing is not correct within reasonable limits, the error throughout the survey is progressive, thus giving incorrect details. If it is at all possible to check this bearing before starting with the adjustment to the survey, this should be carried out.

Fig. 190 shows a sketch plan of a traverse survey with five bearings as stated above. The magnetic azimuths of the various lines are marked on the sketch together with the distances, as follows :—Instrument set at A, needle free and pointing to Magnetic North, head clamp and vernier clamped at zero. With N-sight leading and vernier unclamped the magnetic azimuth AB is 87° 25′ as a free bearing. The instrument is removed to B and the

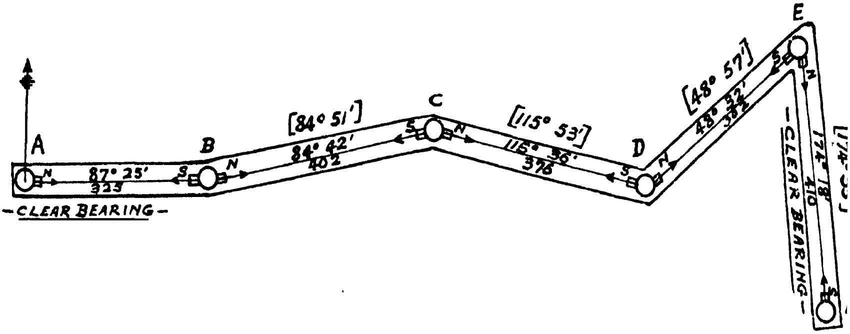


FIG. 190.—Sketch Plan of Traverse Survey with Miners' Compass.

vernier clamped at 87° 25′ with head clamp free. With the eye at N-sight take a back sight to A, and clamp the head of the dial. With eye at S-sight and vernier unclamped, take a fore sight to the point C. The magnetic azimuth of BC is 84° 42′. Follow on in this way until the point F is reached, when a second free bearing is obtained and found to be 35 minutes in excess of the vernier reading or magnetic azimuth.

To correct the error, leaving out of account the free bearing at the start :—

$$\text{Correction for a certain line} = \frac{\text{Error} \times \text{length of line}}{\text{Length of all the lines}}.$$

$$\text{Correction for line } BC = \frac{35 \text{ mins.} \times 402}{(402 + 376 + 352 + 410)}$$

$$= \frac{35 \times 402}{1540} = 9 \text{ mins.}$$

$$\text{,, ,, ,, } CD = \frac{35 \times 376}{1540} = 8 \text{ mins.} + 9 \text{ mins., or } 17 \text{ mins.}$$

$$\text{,, ,, ,, } DE = \frac{35 \times 352}{1540} = 8 \text{ mins.} + 17 \text{ mins., or } 25 \text{ mins.}$$

$$\text{,, ,, ,, } EF = \frac{35 \times 410}{1540} = 10 \text{ mins.} + 25 \text{ mins, or } 35 \text{ mins.}$$

The corrected magnetic azimuths are shown in brackets on the sketch plan, and they are as follows :—

AB 87° 25′ (clear bearing)
BC 84° 42′ + 9′ = 84° 51′
CD 115° 36′ + 17′ = 115° 53
DE 48° 32′ + 25′ = 48° 57′
EF 174° 18′ + 35′ = 174° 53′ (clear bearing).

JULY 1941 EXAMINATION

(*Five questions only to be answered.*)

Calculation of "Displacement" and "Want" of a Fault from Boreholes and Details of Workings

1. A colliery leasehold contains two seams of coal which lie 50 yards vertically apart in the strata. The accompanying plan shows certain workings in the upper seam which have reached the line of a large fault with hade 30 degrees from the vertical. It also shows the sites of three bores which have been put down from the surface to prove the position of the lower seam in the area to the North of the fault. From the data shown on the plan, calculate approximately the vertical displacement, in feet, of the fault at the point marked X. Thereafter show on the plan the extent of the barren area in the lower seam, due to the fault, and indicate the direction of displacement. (40)

A. Fig. 191 shows the plan referred to above. The width of

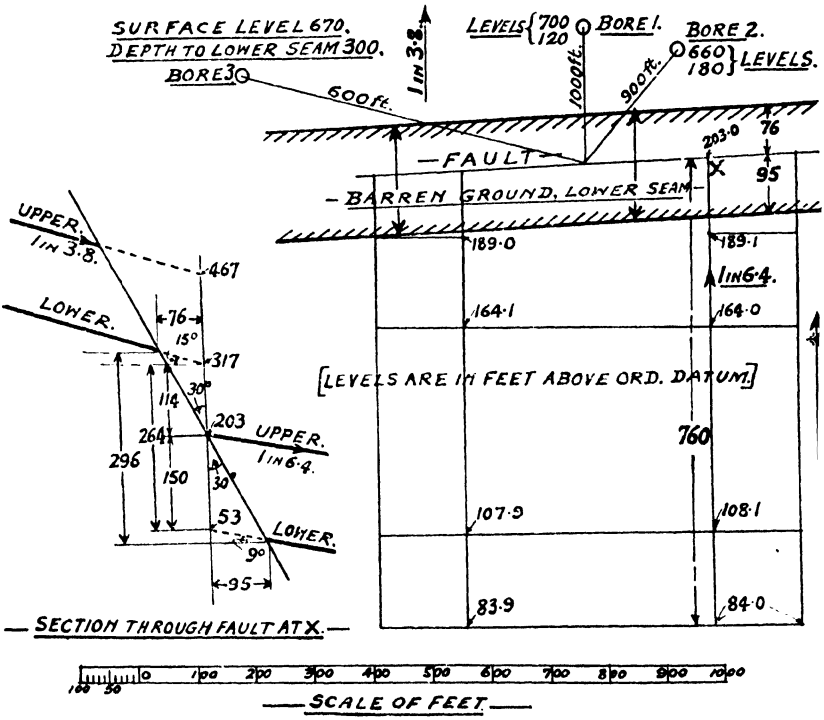


FIG. 191.—Plan referred to in Question 1.

the barren ground has been added to it and is distinguished by shading, while added figures are shown in bolder type. From the workings in the upper seam the following details are available :—

(i) The top seam is rising due North towards the fault at $\frac{203 - 84}{760}$, or 1 in 6·4 ; (ii) the level of the lower seam at the fault is 203 − 150, or 53 feet.

By plotting out the position of the boreholes to scale on the plan, the full dip of the strata beyond the fault is seen to be due South, and it amounts to 1 in 3·8. The following details are now at hand for getting the required results, which can be checked up from the section included in Fig. 191 :—

(i) The level of the lower seam at bore No. 1 is 700 − 120, or 580 feet.

(ii) The level of the lower seam at the point X is $580 - \frac{1000}{3 \cdot 8}$, or 317 feet.

(iii) The angle of dip of the seams South of the fault is $\frac{57}{6 \cdot 4}$, or 9 degrees.

(iv) The angle of dip of the seams North of the fault is $\frac{57}{3 \cdot 8}$, or 15 degrees.

(v) The displacement of the seams vertically in the strata is 317 − 203, or 114 feet between upper seam and lower seam through the fault; and 203 − 53, or 150 feet between upper and lower seams at the fault, in all 264 feet (see sectional drawing in Fig. 191).

The displacement of the seams at the fault is greater than the 264 ft. given above, and amounts to (114 + 114 tan 30° tan 15°) + (150 + 150 tan 30° tan 9°) or 132 + 164, thus making a total of 296 feet approximately, say 300 feet. Thus the fault is a rise one to the North of approximately 300 feet.

The barren ground in the lower seam is 132 tan 30° + 164 tan 30°, or 76 ft. + 95 ft., thus making a total of 171 feet, as shown on the plan.

Plotting Sections of Railway Cutting from Notes of Field Book

2. The following notes are taken from a field book showing the result of levelling operations along the line of a proposed branch railway :—

Draw cross-sections at peg 10 and peg 11, which are 100 feet apart, and thereafter calculate the volume of excavation between these pegs in forming the railway. The width at formation level is 14 feet, the sides of the cutting slope at 1 vertical to $1\frac{1}{2}$ horizontal, and the railway is to rise at 1 in 50 towards peg 12. The formation level at peg 11 is 30·5. Plot the sections to a scale of 1 in. = 10 ft. (30)

A. The following are the given details of levelling operations from field book. The figures given in italics are calculated results.

FIELD-BOOK NOTES OF LEVELLING OPERATIONS.

Back Sight.	Inter Sight.	Fore Sight.	Rise.	Fall.	Reduced Level.	Remarks.
6·50					45·85	At peg 10.
	8·25			*1·75*	*44·10*	30 ft. to right at peg 10.
	4·60		*1·90*		*47·75*	30 ,, ,, left ,, ,, 10.
	6·00		*0·50*		*46·35*	At peg 11.
	9·50			*3·00*	*42·85*	30 ft. to right at peg 11.
	4·00		*2·50*		*48·35*	30 ,, ,, left ,, ,, 11.
		5·50	*1·00*		*46·85*	At peg 12.

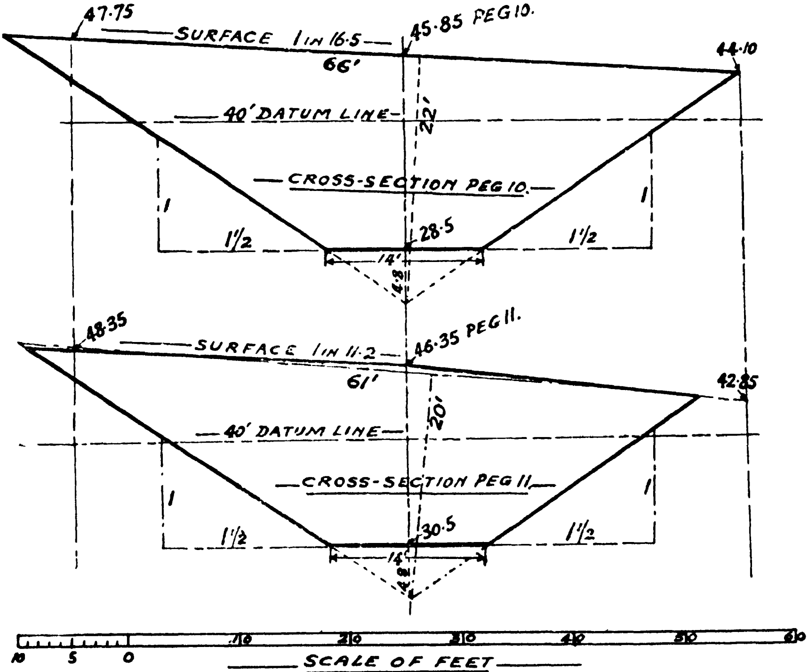


FIG. 192.—Cross-sections of Railway Cutting at Pegs 10 and 11.

Fig. 192 shows cross-sections to required scale at pegs 10 and 11. The formation level at peg 11 is 30·5, and as the railway rises 1 in 50 towards peg 12, the formation level at peg 10 will be 30·5 – 2·0, or 28·5 feet.

The area of cross-section at peg 10 is $\frac{66 \times 22}{2} - \frac{14 \times 4{\cdot}8}{2}$, or 692 sq. ft.

Similarly ,, ,, ,, ,, ,, ,, 11 is $\frac{61 \times 20}{2} - \frac{14 \times 4{\cdot}8}{2}$, or 576 sq. ft.

The volume of excavation between pegs 10 and 11 is $\left(\frac{692 + 576}{2}\right) \times 100$ ft., or 63,400 cu. ft., say 2,348 cu. yds.

Equalising an Irregular Boundary Line on a Mine Plan

3. The undernoted bearings and measurements define an irregular boundary line on a mine plan between two points *A* and *B*, the latter being a point on a straight line *XY*, bearing from South to North. Plot the bearings and measurements to a scale of ½ inch = 100 feet, and thereafter lay down a straight line from *A* to a point on *XBY*, so that the areas to the North and South respectively of that line will be equal. Check the answer by calculation of the respective areas.

From *A*, N. 63° 30′ W. 185 ft.
S. 45° 00′ W. 245 ,,
S. 80° 45′ W. 175 ,,
N. 55° 15′ W. 250 ,,
S. 60° 30′ W. 300 ,, to *B*. (30)

A. Fig. 193 shows the boundary line plotted from the given

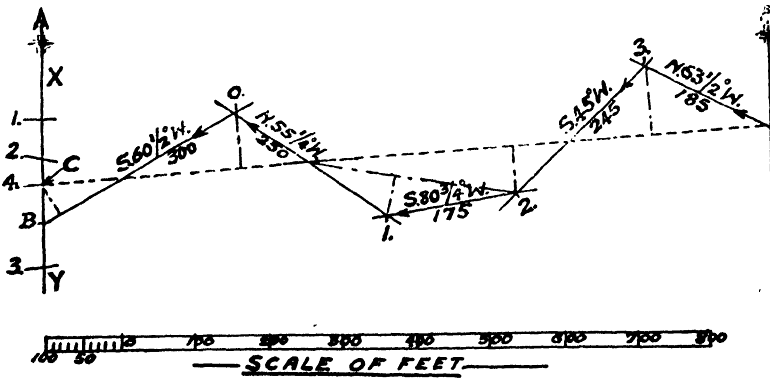


FIG. 193.—Plan showing Traverse Survey and Straight Boundary Line.

bearings and distances, also the straight boundary line AC to replace it. The figures on the plan indicate the method of paralleling applied to get the latter line.

Check.—Area North of line $= \frac{270 \times 95}{2} + \frac{250 \times 75}{2}$

$= 22{,}200$ sq. ft.

,, South ,, ,, $= \frac{130 \times 80}{2} + \frac{360 \times 64}{2} + \frac{290 \times 38}{2}$

$= 22{,}230$ sq. ft.

The above calculations are therefore a check of the position of the constructed boundary line.

Plotting by Co-ordinates and Calculation of Tie Line

4. State what you understand by the expression "plotting by co-ordinates," and the advantages of such method in comparison with plotting by a protractor. The co-ordinates, in feet, of two points A and B, in relation to a point of origin O, are as follows :— A, 1279·87 North, 582·47 West ; B, 755·18 South, 1191·85 East. Calculate the length and azimuth of the line AB, and roughly check your answer by plotting to a scale of $\frac{1}{4}$ inch = 100 feet. (20)

A. Plotting a survey by co-ordinates means that the various bearings of the survey are plotted by measurements made in North and South, and East and West directions, termed "Latitudes" and "Departures," instead of marking off such bearings by means of a protractor and measuring the distances by a scale. The latitudes and departures are termed "Co-ordinates" and are calculated as follows :—

Latitude = Distance × Cosine of Bearing, and are either N. or S. according to the bearing.

Departure = Distance × Sine of Bearing, and are either E. or W. according to the bearing.

Advantages of Plotting by Co-ordinates.—The advantages of plotting surveys by co-ordinates over plotting by protractor are :—

(i) Each point of the survey can be plotted by scale from a common origin, and more accurate plotting is ensured in this way as no progressive errors are introduced. Total co-ordinates are necessary to allow of this, and they can be deduced easily from the bearing co-ordinates.

(ii) In a closed traverse, the accuracy of the survey can be determined from the co-ordinates without plotting. In a traverse which does not close, a tie line can be calculated from the co-ordinates without plotting, and greater accuracy results. Areas enclosed by surveys can be calculated from the co-ordinates without plotting the survey.

(iii) Surveys can be plotted on plans without the damaging effects of pin-pricks, parallel rulers, and rubbers.

(iv) Correct bearings can be calculated easily from the figures taken from the plan by scaling.

(v) Plans can be divided into squares by lines parallel with true North and South, and East and West for plotting purposes; and shrinkage of any part of a plan can be detected and allowed for.

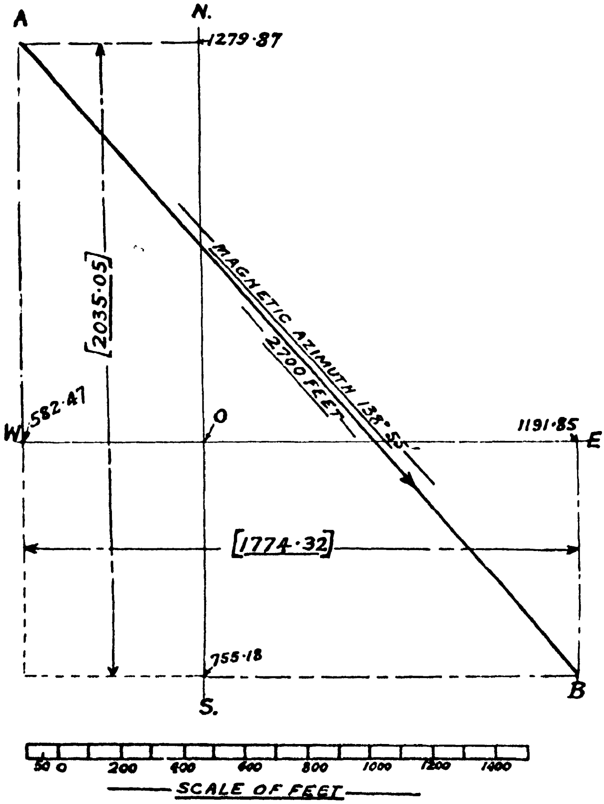


FIG. 194.—Plotting by Co-ordinates.

Fig. 194 shows the plotting required by the above question. To find the azimuth of the line *AB*, the origin is moved to the point A, and the co-ordinates of the point B with respect to A are obtained :—

Latitude of B = 1279·87 + 755·18, or 2035·05 South . (1)

Departure of B = 582·47 + 1191·85, or 1774·32 East . (2)

The natural tangent of the bearing AB can be found from the above co-ordinates :—

$$\text{Tan of bearing } AB = \frac{\text{Departure}}{\text{Latitude}} = \frac{1774{\cdot}32}{2035{\cdot}05} = 0{\cdot}8729.$$

∴ Bearing of line AB is S. 41° 05′ E., and the magnetic azimuth of AB is therefore 180° − 41° 05′, or 138° 55′ . . . (3)

$$\text{Length of line } AB = \sqrt{(\text{Lat.})^2 + \text{Dep.}^2} \text{ or Latitude} \times \frac{1}{\text{Cosine } 41^\circ\ 05'}$$

$$= \frac{2035{\cdot}05}{0{\cdot}7538} = \underline{2700 \text{ feet}} \quad . \quad . \quad . \quad . \quad (4)$$

Plotting New Workings on an old Plan made at Abandonment of Seam

5. A small colliery leasehold contains a shallow seam of coal which was partly worked before being abandoned about 40 years ago. It has been re-opened by a cross-measure drift dipping in from the surface. A well-preserved plan of the old workings, with surface lines, is available, but no meridian line is shown on it. You are asked to make the first survey of a small area of the new workings, and thereafter to lay them down on the plan in correct relation to the surface lines. State clearly how you would do so, using an ordinary mining dial for the survey work. (20)

A. As the old plan is well preserved, it is presumably to be used as a working plan of the seam referred to above. Two important points must be attended to before this can be carried into effect : (*a*) The magnetic meridian must be put on the old plan and dated for future correction, and (*b*) the old plan must be checked and corrected for shrinkage.

To carry out this work successfully, a surface survey must be made by the compass available. Such a survey should be made with the vernier on the meridian base-line method and it should include all the surface boundaries, and the position of the mouth of the cross-measure drift. When this survey is plotted and traced on the old plan, comparisons can be made and corrections made for shrinkage. At the same time the magnetic meridian line can be traced through to the old plan.

The survey of the underground workings should start from the mark of the previous survey at the outcrop of the drift, as a continuation of the previous survey. In plotting the new workings on the old plan, care should be taken to allow for shrinkage by using a specially-constructed scale for this purpose.

Gunter's Chain—its Construction and Use

6. What is the length of Gunter's chain? Why was that length adopted and subdivided into links? What are the practical advantages and disadvantages of such a chain compared with a steel band, (*a*) when used in a surface traverse, and (*b*) when used in an underground traverse? (20)

A. Gunter's chain has an overall length of 792 inches, or 66 feet, or 22 yards, and it is subdivided into 100 links, each link being 7·92 inches long. This length has been adopted to allow of the rapid computation of areas, while the subdivision on the decimal system greatly facilitates this work. Scales subdivided in the same way as the chain are used for plotting surveys. A square chain is 22 × 22 yards, or 484 sq. yards, and as there are 4,840 sq. yards in an acre, a square chain is therefore one-tenth of an acre. Similarly 10 sq. chains or 100,000 sq. links are equivalent to an acre, and sq. links can therefore be expressed in acres by simply dividing by 100,000.

For surface surveys, steel bands are often used in preference to chains. They are easily manipulated, more accurate, easily read in good light, and easily negotiated over rough ground. There are no small links to catch up on projections. They are also easily cleaned and kept free from rust and dirt. If not carefully used, however, they are liable to kink and eventually break.

For underground traverse surveys, chains are mostly used in preference to steel bands. They are more flexible for handling as the survey progresses, more easily read in bad light, and they can withstand rough usage better. Steel bands are liable to be damaged or broken when used for underground traverse work, while collections of grease and dirt make reading more difficult; consequently, mistakes in reading might be a more or less frequent occurrence.

APPENDIX I

COAL MINES ACT, 1911

BOARD FOR MINING EXAMINATIONS

Notes for the guidance of Students who intend in the future to take the Examinations for a Certificate of Competency.

Standard of General Education

Future applicants who were *under* 17 *years of age* on 21st June, 1935, will not be admitted to the Examinations for Manager's or Under-Manager's Certificates unless they have first satisfied the Board for Mining Examinations that their general education has reached a standard prescribed by the Board's Rules. General education for this purpose means, not knowledge of mining itself, but knowledge in certain basic subjects—namely, English, Mathematics, Drawing and Science—which a student must possess before he can acquire a satisfactory and intelligent knowledge of the mining subjects of the Board's Examinations.

It is most important that young mining students, and boys who intend to study mining after they leave school, should realise the importance of this new requirement if they have any idea that they will want, when they grow up, to try to obtain a Manager's or an Under-Manager's Certificate. They should make quite sure that they are properly qualified in respect of their general education *before* they proceed very far with their studies in mining subjects. Otherwise, they may find in the years to come that they have got to suspend their mining studies in order to improve their general education—or even, when they make application to take one of the Board's Examinations, that they are not qualified to sit.

How is a boy to make sure that he is qualified in respect of his general education ? The answer is that as a general rule he must obtain and submit to the Board for Mining Examinations either (*a*) one of the certificates* indicated in the first list at the end of these notes, or (*b*) an approved certificate* issued by one of the examining bodies named in the second list.

*A boy who does not possess one of these certificates but holds some other certificate covering the four subjects, English, Mathematics, Drawing and Science, and at least of equal standing, can submit full particulars to the Board, and the Board may accept such alternative certificate if they think fit ; but this will only be done in very exceptional cases such as that of an applicant who has been educated outside Great Britain.

When a student has obtained his Certificate of General Education he should proceed to have it registered by the Board for Mining Examinations at once. For this purpose he should send the Certificate to

Board for Mining Examinations,
Mines Department,
Dean Stanley Street,
Millbank,
London, S.W.1.

asking that it may be accepted as evidence that he has attained the standard of general education prescribed by the Board's Rules. If the certificate is accepted by the Board, it will be registered free of charge and will be returned without delay to the student, who will then be able to proceed with his studies of mining subjects with the certainty that, as regards his general education, he is eligible to sit for the Board's technical examinations in due time. If the certificate is not in order the student will be so informed.

If a boy is in any doubt or would like further information about his particular case, he should communicate with one of the Technical Institutes or Mining Classes in his district, or with one of the Examining Bodies shown in the second list.

First List

(*School Certificates which will be accepted as evidence that the holder has attained the prescribed standard of general education.*)

The School Certificate of the Oxford and Cambridge Schools Examination Board.

The School Certificate of the Oxford Delegacy for Local Examinations.

The School Certificate of the Cambridge Local Examination Syndicate.

The Certificate of the ordinary Matriculation Examination of any University in Great Britain.

The Certificate of the General School Examination of the University of London.

The School Certificate of the University of Durham.

The School Certificate of the Northern Universities Joint Matriculation Board.

The School Certificate of the University of Bristol.

The School Certificate of the Central Welsh Board.

The Day School Certificate (Higher) of the Scottish Education Department (or the Leaving Certificate of that Department).

Second List

(*Information as to which of the Certificates issued by these bodies will be accepted can be obtained from Local Technical Institutes or Mining Classes or from the bodies concerned.*)

Northern Counties Technical Examinations Council,
1, Claremont Place, Newcastle-on-Tyne, 2.

Yorkshire Preliminary Mining Examinations Board,
County Hall, Wakefield.
Union of Lancashire and Cheshire Institutes,
33, Blackfriars Street, Manchester, 3.
East Midland Educational Union,
14, Shakespeare Street, Nottingham.
Union of Educational Institutions,
174, Corporation Street, Birmingham, 4.
Kent County Examinations Board,
Springfield, Maidstone, Kent.
Examining Board of the Joint Mining Education Advisory Committee, Monmouthshire and South Wales Coalfield,
County Hall, Newport, Mon.
East of Scotland Joint Committee for Mining Preliminary Examinations,
Heriot-Watt College, Edinburgh, 1.
West of Scotland Joint Committee on the Organisation of Classes in Science and Technology,
Royal Technical College, Glasgow, C.1.

APPENDIX II

STATUTORY RULES AND ORDERS 1933, No. 1166*

COAL MINES

Application for Manager's and Under-Manager's Certificates.

1. An applicant for a First-Class Certificate of Competency (hereinafter called a First-Class Certificate) must be at least 23 years of age, and an applicant for a Second-Class Certificate of Competency (hereinafter called a Second-Class Certificate) at least 21 years of age, on the first day of Examination. A certificate of competency will not be issued to a successful candidate until he reaches the age of 23. In every case an applicant must furnish to the Board for Mining Examinations (hereinafter called the Examining Board) the following testimonials and certificates of his preliminary qualifications :—

(i) Satisfactory testimonials, on forms provided for the purpose, of his sobriety and general good conduct ;

(ii) A certificate of proficiency in First-Aid from a Society or Body approved by the Board of Trade ; and

(iii) A Fireman's Certificate, which for the purposes of this Rule means a Certificate from a Mining School or other institution or authority approved by the Board of Trade, as to his ability to make accurate tests (so far as practicable with a safety-lamp) for inflammable gas and to measure the quantity of air in an air-current, and that his hearing is such as to enable him to carry out his duties efficiently.

2. (*a*) Save as is hereinafter expressly provided, an applicant must satisfy the Examining Board that he has had practical experience in mining at the working face and other parts of the underground workings for a period of not less than 5 years (or not less than 3 years if he holds an approved Degree or an approved Diploma in scientific and mining training granted after a course of study of at least 3 years) ; provided :

(i) That such experience shall have been obtained for a period of at least 2 years (or at least 1 year and 6 calendar months if the applicant holds such approved Degree or approved Diploma as before mentioned), in the performance or

*There are some war-period alterations to these Statutory Rules and Orders which are not included in the text.

responsible control of stonework, timbering, repairing and of either the getting of minerals (by hand or machinery) or any work directly connected with such getting at the face of the mine; and

(ii) That practical experience for a period not exceeding 6 months obtained either underground or on the surface in engineering workshops which are definitely associated with mining machinery may, at the discretion of the Examining Board, be accepted as part of the practical experience of not less than 5 years as aforesaid.

(*b*) The practical experience required by this Rule, saving the experience in engineering workshops under Clause (*a*) (ii), shall have been obtained either in Great Britain in a mine or mines under the Coal Mines Act, 1911, or partly therein and partly elsewhere in such coal mine or coal mines as shall, in the judgment of the Examining Board, have provided equivalent experience.

3. Subject to the provisions of Rules 6 (*d*), 7 (*d*) and 8 respectively, an applicant for a Certificate of Competency shall be required (A) to qualify at a written examination, (B) to qualify at an oral examination held in conjunction with such written examination, and (C) to obtain at least 60 per cent. of the maximum marks for the whole examination (written and oral).

Written Examinations

4. An applicant for a FIRST-CLASS Certificate shall be required to possess such a knowledge of mathematics, physics (including electricity), chemistry, geology and engineering science as will enable him to qualify at a written examination in relation to mines under the Coal Mines Act, 1911, on the subjects in the following syllabus:

(i) *Winning and Working.*—The geology of coal and other stratified deposits. Boring and Sinking. Systems of laying out and working under varying conditions. The application of machinery to mining. Methods of supporting roof and sides. Blasting and general knowledge of explosives and other means of getting minerals.

(ii) *Theory and Practice of Ventilation.*—The properties, identification and practical estimation of gases met with in mines. Sources, effects and control of heat in mines. Natural ventilation, fans and other ventilators. The distribution and control of the air underground. Stoppings and air-crossings. Construction, use and testing of safety-lamps.

(iii) *Explosions, Underground Fires and Inundations, their Causes and Prevention.*—Gas. Coal-dust. Spontaneous heating. Rescue operations, apparatus and organisation. Recovery of mines after explosions, fires and inundations. Precautions against water and gas in approaching disused workings and when mining under waterlogged strata.

(iv) *Machinery.*—For winding, hauling, pumping, mechanical coal-cutting and conveying, etc. Generation and transmission of power; mechanical, steam, compressed air, hydraulic, electrical. Strength of materials.

(v) *Surveying, Levelling and Drawing.*—Magnetic declination. Loose and fast needle dialling. Calculation of areas and volumes. Contour lines and levelling. Traversing with the theodolite underground and on the surface. Connecting of surface and underground surveys. Triangulation. Mine plans and sections. The use, care and testing of instruments.

Each applicant must produce a plan of a mine survey and a section prepared from an underground levelling made and drawn by himself with the original plottings and the notes from which the plottings have been made, and the work must be certified by him as having been carried out by himself. The plan and section must have been made and drawn not more than 2 years before the date of the oral examination hereinafter mentioned.

(vi) *General Management and Mining Legislation.*—Layout and organisation of surface arrangements under varying conditions. First-Aid and Ambulance Work. Legislation relating to safety, health and hours of employment. General Regulations and Orders. Writing of Reports.

5. An applicant for a SECOND-CLASS Certificate shall be required to qualify at a written examination on the undermentioned subjects, in relation to mines under the Coal Mines Act, 1911, in which the questions set will be of a nature suitable for practical working miners :—

(i) *Methods of Working.*—Systems of laying out and working, under varying conditions, of coal and other stratified deposits. The application of machinery to mining. Methods of supporting roof and sides. Shot-firing.

(ii) *Ventilation.*—The properties, identification and practical estimation of gases met with in mines. Natural ventilation, fans and other ventilators. The distribution and control of the air underground. Stoppings and air-crossings. Measurement of air-currents. Construction, use and testing of safety-lamps.

(iii) *Explosions, Underground Fires and Inundations.*—Causes and prevention. Gas. Coal-dust. Spontaneous heating. Rescue operations, apparatus and organisation. Precautions in approaching disused workings and mining under waterlogged strata. Recovery of mines after explosions, fires and inundations.

(iv) *Machinery.*—Machinery and plant in common use at a colliery, including the use of electricity, and with special reference to safety.

(v) *Surveying and Levelling.*—Elementary surveying and levelling. Arithmetic (calculation of areas and the volumes of simple solids).

(vi) *Mining Legislation.*—Legislation relating to safety, health and hours of employment. First-Aid and Ambulance work. General Regulations and Orders. Writing of Reports.

6. (*a*) The maximum marks at a written examination for a First-Class Certificate shall be as follows :—

Subject 1.—Winning and working	250
,, 2.—Theory and practice of ventilation . .	200
,, 3.—Explosions, underground fires and inundations .	130
,, 4.—Machinery	150
,, 5.—Surveying, levelling and drawing . . .	140
,, 6.—General management and mining legislation .	130
Total	1,000

(*b*) The maximum marks at a written examination for a Second-Class Certificate shall be as follows :—

Subject 1.—Methods of working	250
,, 2.—Ventilation	250
,, 3.—Explosions, underground fires and inundations.	200
,, 4.—Machinery	100
,, 5.—Surveying and levelling.	100
,, 6.—Mining legislation	100
Total	1,000

(*c*) An applicant shall be deemed to have qualified at a written examination if he obtains 40 per cent. or more of the maximum marks for each subject, and 55 per cent. or more of the maximum marks upon the six subjects collectively.

(*d*) An applicant who fails to obtain 40 per cent. of the maximum marks in one subject or in two subjects, but who nevertheless obtains 60 per cent. or more of the maximum marks upon the six subjects collectively shall be eligible to be re-examined at the next succeeding written examination in the respective subject or subjects in which he so failed, and if he then qualifies in such subject or subjects he shall be deemed to have qualified at a written examination.

Oral Examinations

7. (*a*) An applicant who qualifies at a written examination shall be required to qualify at an oral examination held in conjunction therewith and based upon the Syllabus for a First-Class Certificate or a Second-Class Certificate as the case may be and designed particularly to test his ability in the practical application of his technical knowledge.

(*b*) The maximum marks at an oral examination shall be 300 for a First-Class Certificate, and 400 for a Second-Class Certificate.

(*c*) An applicant shall be deemed to have qualified at an oral examination if he obtains 50 per cent. or more of the maximum marks.

(*d*) An applicant who obtains at least 60 per cent. of the maximum marks at a written examination and also at least 50 per cent. of the maximum marks at the oral examination held in conjunction therewith but fails to obtain at least 60 per cent. of the maximum marks for the whole examination (written and oral) shall be eligible to be re-examined at the next succeeding oral examination.

General

8. An applicant who is eligible, but by reason of illness is unable to attend to be re-examined under the provisions of Rule 6 (*d*) or to attend an oral examination under the provisions of Rule 7 (*a*) or Rule 7 (*d*) may (if the Examining Board in their discretion so decide) for the purposes of such provisions attend the next respective succeeding examination.

9. The Rules made by the Board for Mining Examinations on the 23rd day of December, 1926, are hereby revoked, provided that any applicant who was qualified to sit and duly sat at any examination for a First-Class Certificate or a Second-Class Certificate held prior to the date of these Rules coming into force shall be deemed to have satisfied the requirements of Rule 2 of these Rules.

These Rules may be cited as the Mining Examinations (Certificates of Competency) Rules, 1933, and shall come into operation on the first day of March, 1934.

INDEX

Winning and Working

Theory and Practice of Ventilation

Explosions, Fires, and Inundations

Machinery: Electrical

Machinery: Steam and General

Surveying, Levelling, and Drawing