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UNWATERING FLOODED COAL MINES IN WASHINGTON

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By S. H. Ash [2] and Thomas Murphy [3]

PURPOSE OF REPORT

The United States Bureau of Mines has published relatively little on the flooding and unwatering of coal and metal mines. Bulletin 229, Fifty-Nine Coal-Mine Fires, published in 1927, contains several instances of the flooding of mines or sections thereof to extinguish fires but does not describe their unwatering. Report of Investigations 2649, Explosion Hazards Incidental to Unwatering Coal Mines, published in 1924, warned against the possibility of gas accumulations in flooded mines and indicated the precautions to be taken. Report of Investigations 2255, An Unusual Hazard in Reopening Long-Flooded Timbered Metal Mines, published in 1921, told of methane being pocketed and the danger thereof.

The flooding of mines originates in many ways. At some mines flooding followed reopening after what was thought to be permanent abandonment; also, explosions have caused some mines to be flooded. Extinguishing fires, accidental encountering of underground streams or flooded old workings, and caves of strata admitting water from underground workings, water-bearing strata, or surface accumulations have also flooded all or parts of mines. Unfortunately, unwatering often necessitates much study and causes apprehension largely because experiences in similar situations are not generally known, as few descriptions of successful or unsuccessful methods are available. The job has sometimes been contracted for outside the local management involved and has become a matter of record for the contractor only. As the cost-plus contract system often employed in the past is now known to be expensive, careful managements elect to handle the problem themselves, in line with the general policy of cutting expense where possible. This report describes specific instances of successful unwatering by operators.

PUMPING PROBLEM IN WASHINGTON

Because large quantities of water must be handled in normal mining operations in Washington, pumping is important; any decision to unwater a mine invariably involves additional pumping equipment, not only for unwatering but for handling an increased volume of water during the life of the mine should an adjacent mine be

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drained. At some Washington coal mines the tonnage of water handled in 24 hours has exceeded the tonnage of coal hoisted, and the pumping cost has exceeded the hoisting cost per ton of coal.

MINING COSTS IN WASHINGTON

To show the relative significance of pumping costs in total expenses, condensed mining costs of some western Washington mines are submitted. They can readily be adapted to give a tentative estimate for similar situations under any wage scale or material charge.[4]

MINING COSTS IN KING AND PIERCE COUNTIES

Mining costs in King County are $2.50 to $2.96 per short ton of coal. General expenses add 52 to 83 cents per ton. Drainage is 1.2 to 5.9 cents per ton, and 11 to 27 cents for power is charged to interdepartmental operations. One mine having total costs of $3.35 per ton has relatively little pumping, whereas one removing 700 to 1,000 gallons of water per minute throughout the year has a coal-production cost of $3.36; drainage costs the latter 5.9 cents, compared with 1.2 cents for the former, with power somewhat higher.

Coal-production costs in Pierce County are $2.74 to $3.54 per ton. General expenses at one mine add 82 cents. Drainage is 1.2 to 3.5 cents per ton, and 15 to 38 cents for power is charged to interdepartmental operations. One mine with a coal-production cost of $3.54 per ton pumps 1,000 gallons of water per minute throughout the year.

In western Washington pump men receive $5.25 per day, inside laborers the same, and skilled trades $5.50 to $6.

Any undertaking that aims to eliminate or reduce pumping costs should be thoroughly studied when a comparatively large volume of water must be handled during the life of the mine.

The high cost of power for pumping and intermittent hoisting has meant virtual abandonment of local power-generating units at most Washington mines, and energy is now purchased from the large utility companies.

METHODS OF UNWATERING

Unwatering methods range from rather crude drilling to systems worked out by careful study and efficient application; others involve tunneling. Some mistakes have been made, but they have not caused loss of life.

A common mistake is failure to realize that control features, such as valves and other fittings, can not be manipulated efficiently and safely after the water has been tapped; and, oddly, the probable action of water under pressure is not always realized. Some mines where no precautions were taken as to fittings and pressure conditions were temporarily flooded when water was encountered. Some boreholes used for prospecting were not accurately located or mapped or were encountered unexpectedly. At times it has been necessary to seal off the water if control methods were not in effect when the boreholes were drilled.

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[4] Ash, S. H., Mining Operations at the New Black Diamond Mine of the Pacific Coast Coal Co.: Thesis, University of Washington, 1929.

UNWATERING NEW CASTLE MINE, KING COUNTY

The following description of successful unwatering by diamond drilling in rock formation discusses removal of more than 250,000,000 gallons.

The old New Castle mine is on the eastern shore of Lake Washington, King County, 22 miles by rail from Seattle. It was shut down on December 17, 1894, because of a fire which got beyond control, was promptly flooded, and has never been reopened. The property is owned and operated by the Pacific Coast Coal Co. of Seattle, Wash.

In 1906 the Muldoon coal bed was found workable; a slope mine was started in this measure and later worked out. For years the fine coal and reject material from the preparation plant were dumped into the slope of the old mine and were among the great difficulties that had to be overcome in unwatering the New Castle mine. At first it was feared that when the water was removed from the old mine fire would start from spontaneous combustion; but this never occurred, and it has been found that the old mine filled with black damp as the water was voided.

The old mine, stratigraphically about 400 feet above the Muldoon bed in which the active mine was being operated (see fig. 1), was unwatered to remove the hazard its presence involved in working coal beds of the active mine. The measures include four workable beds,[5] which dip an average of 40° and are separated by the usual shales and sandstones (fig. 2, *A* and

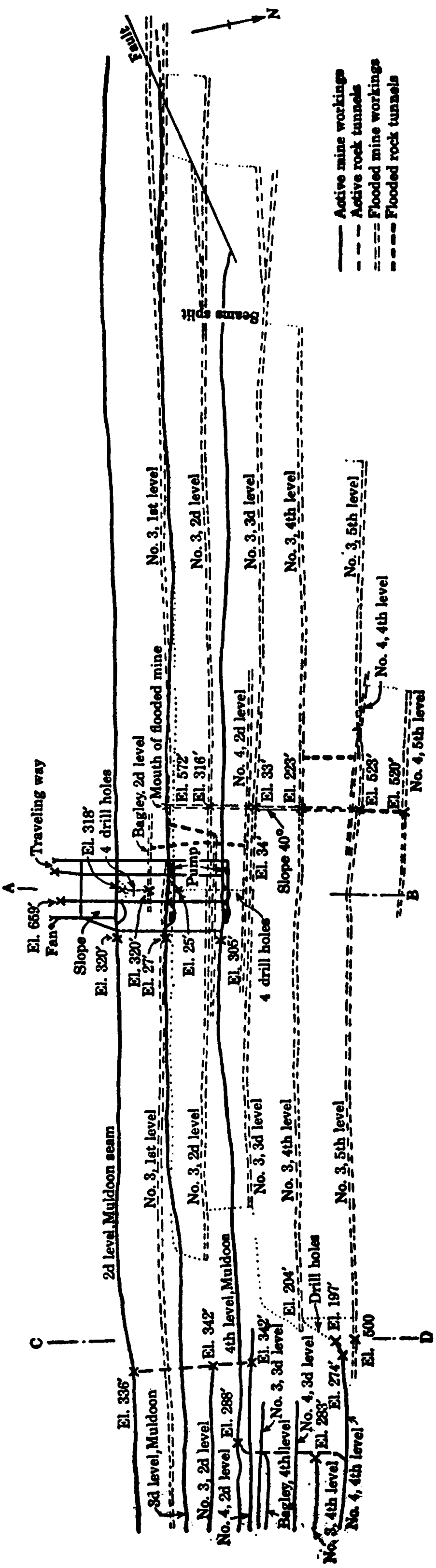

FIGURE 1.—Plan of New Castle mine, King County, Wash.

[5] Bureau of Mines, Analyses of Washington Coals: Tech. Paper 491, 1931, p. 119.

B). Before the overlying beds could be worked from the active slope it was necessary to go safely beyond the extreme limits of the old mine. As the map shows, only a comparatively small area of beds 3 and 4 could be worked without the unwatering of the old workings.

Although unwatering was contemplated for several years, it was never actually attempted until the early part of 1920.

TYPE OF DRILL USED

A type B prospecting diamond drill, driven by air supplied from the mine compressors on the surface, bored the holes. The hydraulic

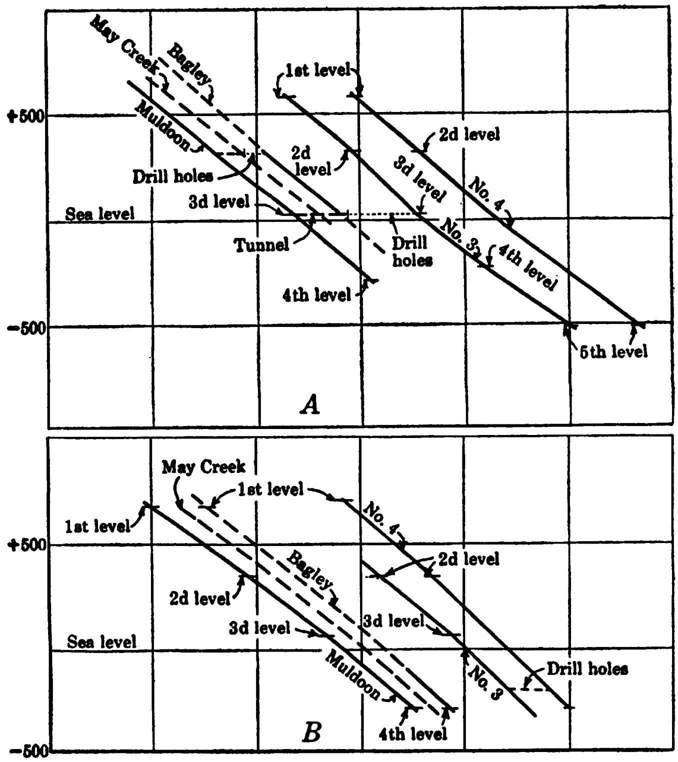

FIGURE 2.—*A*, Transverse section of strata on line A–B, Figure 1, showing relative position of coal beds, flooded area, and drill holes from underlying Muldoon bed, New Castle mine; *B*, transverse section of strata on line C–D, Figure 1, showing relative position of coal beds and drill holes from overlying bed 4, New Castle mine

cylinder of the drill was direct connected to the column-line pressure of the active mine at each level.

The Bagley bed was worked only at the second level and was not opened to any large extent. Operations in this bed were so restricted that it could be reached only by drill holes driven from a point near the slope of the active mine, which (see fig. 1) was not far from the slope of the old mine. The workings in this bed were the first tapped. This measure overlies the Muldoon bed and is connected by a rock tunnel on the second level of the old mine to the second level of bed 3.

The May Creek bed lies about midway between the Muldoon and the Bagley measures and is a dirty coal averaging about 4 feet in thickness. To avoid bringing the water into the coal bed, as well as to obviate casing the holes through it, the operators decided to drive a rock tunnel through and beyond this bed to within 50 feet stratigraphically of the flooded Bagley workings in the old mine.

Accordingly, this tunnel was driven 7 by 7 feet in dimensions to the point indicated, and boreholes were drilled thence to the Bagley bed. The first hole was driven level with a bit which made a 2-inch hole. It tapped the water successfully at a pressure of 97 pounds per square inch. In drilling through the footwall of the Bagley workings the hole caved frequently, and the clay bed which underlies the Bagley bed also gave trouble. A steel bit was used to drill through the broken ground in the footwall of the old mine.

It was decided to drill succeeding holes larger; accordingly, the machine was equipped with N-size rods and a single-tube core barrel, and 3-inch holes were drilled. This increased the pressure on the bit, and there was difficulty in getting rods out of the hole because the water forced the cuttings and loose rock between the rod and the sides of the opening. The second hole was driven at a pitch of 3½° upward, which facilitated operations, as the hole kept itself clean much more easily.

Those in charge decided to drill the next hole with B rods (about 2 inches in diameter) but still to employ an N-size single-tube core barrel (approximately 10 feet long), thus making a 3-inch hole with a small rod. The arrangement worked well; the rest of the holes on this and other levels were drilled in the same way.

Four holes were driven in the Bagley bed; the shortest was 90 feet long and the longest, where the mine had caved, 120 feet. When drilling was done with the small rods and large core barrel on a pitch of 3 to 7°, the holes cleaned themselves without difficulty and pressure was relieved so that in many instances when the rods were taken out the pressure could be controlled by the chuck of the drill alone, even when the water was tapped at 217 pounds per square inch. No gland was necessary to keep the water back, as this device maintains pressure rather than relieves it. Four holes totaling 440 feet were drilled on the second level and tapped the water at 97 pounds per square inch pressure.

SECOND SERIES OF DRILL HOLES

The second point chosen for a series of drill holes was the third Muldoon level near the main slope of the active mine. Here a 7 by 7 foot rock tunnel 200 feet long was driven slightly upgrade. It reached and passed the Bagley bed beyond the part where coal had been mined. In the old mine the Bagley bed was worked only on the second level, as it was dirty and was kept open with difficulty; consequently the bed tapped no water. The tunnel ended 319 feet horizontally from the old mine.

The rock was found to be solid, and four 3-inch holes were driven with the N core barrel and bit, using B rods. Caved ground was found at the third level in the footwall of the old mine. The holes were driven with a rise of 3½° to reach the old workings about 10 feet above the old gangway. They all holed through at about 319 feet, and the pressure was found to be 217 pounds per square inch.

The last 20 feet of these holes were drilled with a 3-inch steel bit. The ground underlying the footwall of bed 3 in the old mine is chiefly coarse-grained sandstone, thoroughly saturated with water; consequently these holes made considerable water when within 30 feet of their collars. In all, the four holes were driven by the diamond drills 1,276 feet from the end of this rock tunnel.

When these holes were completed the drill was moved to the fourth level of bed 4, which had been reached by a rock tunnel driven beyond and east of the old workings. The location of this level can be seen to the left of Figure 1 near the lower end of section line C–D. Here five 3-inch holes aggregating 1,300 feet and averaging 260 feet were drilled like those on the third level. The first hole tapped the water at 210 pounds per square inch.

To obtain a solid face the rock tunnel was driven 21 feet into the footwall of bed 4. It was decided to make the boreholes strike about 100 feet from the face of the fourth level of bed 3 in the old mine. To do this by tunnel and drill holes driven at right angles to the direction of the level would have involved extension of the fourth level of bed 4 in the new mine. As this would have delayed operations it was decided to drive the rock tunnel and boreholes on a line which struck the level at somewhat less than a right angle, requiring greater length of borehole but considerably less time to complete the work.

As stated, the old mine was first allowed to fill with water; later culm was dumped into the slope for years. As difficulties had been experienced from coal blocking the holes on the second level, it was assured that no trouble would be had with those drilled on the fourth level of bed 4, or placed farthest from the slope of the old mine and passing into that mine through the hanging wall and not the foot wall. The first 60 feet of the second hole passed through sandstone saturated with water, making the workings wet. The water soon drained off, however; the remaining 180 feet of the ground between beds 3 and 4 was fine-grained sandy shale through which water would not pass; the hole therefore was dry.

Although existing maps were known to be in error, it was decided to accept the elevations as correct. The first drill hole, driven to enter the old mine about 20 feet above the probable floor of the gangway, indicated an error of about 50 feet in the horizontal projection. The hole was dry 2 feet from the old mine, notwithstanding a pressure of 210 pounds per square inch on the rock.

No water was noticeable until the hole was within 1 foot of the old workings, where the drill was stopped for one-half hour. Even then the water was not troublesome; but when it was shut off from the rod, a small amount continued to flow. The drill broke through after being driven 1 foot farther. The rods had been pulled 4 feet from the old mine and the core barrel cleaned so that there would be no probability of the hole becoming blocked. The entire hole was drilled with an N-size diamond bit.

Experience gained at this mine showed clearly the benefit of drilling into the hanging wall instead of the foot wall, as there is no broken ground or dirt to be passed. Although the fine coal came through the hole, the drill was driven a short distance into the measure after the boring proper was completed. This project showed clearly that in driving toward an old mine danger can be avoided only by keeping drill holes ahead.

When the rods were put into the hole under pressure or moved to the hole, the drill drum with rope and brake was employed. The cable was run from the drum to the face or rear of the machine around a sheave and attached to the rods by clamps. The rods were taken out or put in place in 10 or 20 foot lengths, depending upon conditions.

The face is prepared for drilling as follows (see also fig. 3, depicting the pipe arrangement): The control valve is attached to a casing which must be put in the hole a short distance. For this purpose a hole was driven 10 to 20 feet with a 3-inch bit. At this mine the casing holes in the upper levels were driven 10 feet and in the fourth level 20 feet; the additional length was allowed because of softness of the sandstone and proximity of the coal bed. The

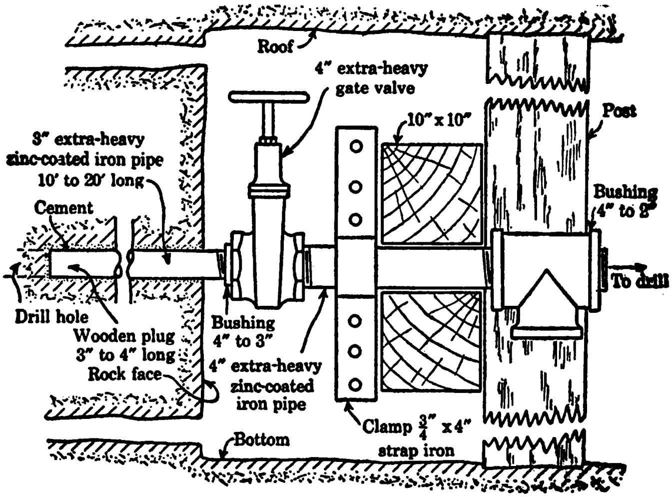

FIGURE 3.—Valve and pipe arrangements for drilling New Castle mine

casing hole was then reamed out with a diamond bit throughout its entire depth to allow a 3-inch pipe to be concreted into place.

Zinc-coated pipe was employed as it was thought that the zinc would protect the pipe from the acid mine water. Extra-heavy pipe and valves were used for greater pressures than that on the second level. When the hole was reamed out a 3 or 4 inch wooden block was inserted in the end of the pipe, to be drilled out when the hole was driven past the casing; the hole was then filled with cement mixed with enough water to make handling easy.

As much of this grout as possible was forced into the hole. Here about one sack of cement was needed for a 20-foot casing hole. Next, the hydraulic cylinder forced the 3-inch pipe with the wooden plug into the hole, driving the cement backward around the outside of the pipe. A valve was then put upon its outer end and the whole allowed to stand until the cement had set (4 to 7 days).

Usually, pipes were installed simultaneously for each of two drill holes before the first was driven through; next, the remainder of each hole was drilled. This practice aimed to forestall leakage of water from complete to incomplete holes through fractures in the rock that would prevent placing the remaining casing pipes. A clamp was put on the pipe. A 4-inch valve was used to allow the core barrel to be withdrawn from the 3-inch hole without injuring the valve seat. A bushing to fit the rod and steady it was employed in the outer-run opening of the 4-inch T placed outside the valve. After the rods were removed the valve was closed until it was time to let the water run. The T also was moved to the next hole to be drilled. The valve was fastened directly to the casing pipe to have as few connections as possible. On the fourth level the soft rock in the face kept spalling off, and after the first hole was drilled the face was concreted behind the bracing timbers, the valve being placed outside (fig. 3).

The ground drilled was mostly soft sandstone and sandy shale and the average rate of drilling 48 feet per 8-hour day. Drilling was conducted by the management, with one drill man and two or three assistants. The cost of drilling, not including diamonds, was 53.3 cents per foot. Here only one set of diamonds, eight stones in all, was used. The cost per foot for diamonds was 13.2 cents.

TAPPING AND DRAINING FLOODED MINES IN THE ROSLYN FIELD FROM A LOWER LEVEL AT RELATIVELY HIGH PRESSURES

The following description of a successful unwatering method involves the use of steel bits; all drilling was done in the coal bed, and 419,000,000 gallons of water were voided.

WORKING CONDITIONS

The workings of the mines of the Northwestern Improvement Co. in the Roslyn field, eastern Washington, lie generally along the north flank of a pronounced syncline, and the bed dips 6 to 52°. The coal is generally 4½ feet thick.

The first mine in the field, No. 1, was opened in 1886 at an elevation of 2,347 feet, where the coal outcropped in a gulch. A straight slope was driven down the pitch, and entries were developed east and west on the strike. In 1897 a shaft was sunk from the surface in the valley at an elevation of 2,210 feet down to and intersecting Mine 1 on the eighth east level at an elevation of 1,581 feet. From this point down to the fourteenth west level the slope mine was called No. 4. The bottom of the slope at fourteenth west level is at an elevation of 1,165 feet, and almost in the syncline.

In October, 1909, a disastrous explosion in Mine 4 wrecked the shaft and tipple and did great damage underground. The shaft tower was rebuilt shortly thereafter and the shaft reopened; but the slope and entries were not cleaned, and the mine was allowed to fill with water to within approximately 300 feet of the shaft collar. The water level had to be held at an elevation of 1,934 feet, or 276 feet below the shaft collar, as here it would run through an opening in the barrier pillar into the workings of Mine 5, which had been developed to the east.

Before 1923 the water in shaft 4 had been held down by bailing with two 1,200-gallon steel tanks, running in the regular cage guides and operated by the steam hoist, but in that year this system was replaced by an air lift of 42,000 gallons per hour capacity. The mine "made" about 90,000,000 gallons of water per year.

The first, second, and third west levels in Mine 5 had been driven through to the upper workings of Mine 4 at the time of the explosion there. After the explosion a 300-foot barrier was left between the two mines below the third west level in Mine 5.

Mine 5 was subsequently developed to the ninth west level, and about the time this level was driven to its stopping point at the barrier small drippings of water began to appear in roof crevices. Ordinarily this would cause no alarm, as such crevices are common in this particular field. However, it had long been observed that ordinary ground water running into the mine has no odor, whereas water that has remained even a few months in an abandoned mine is likely to be strongly sulphurous. The water in the ninth west level had this

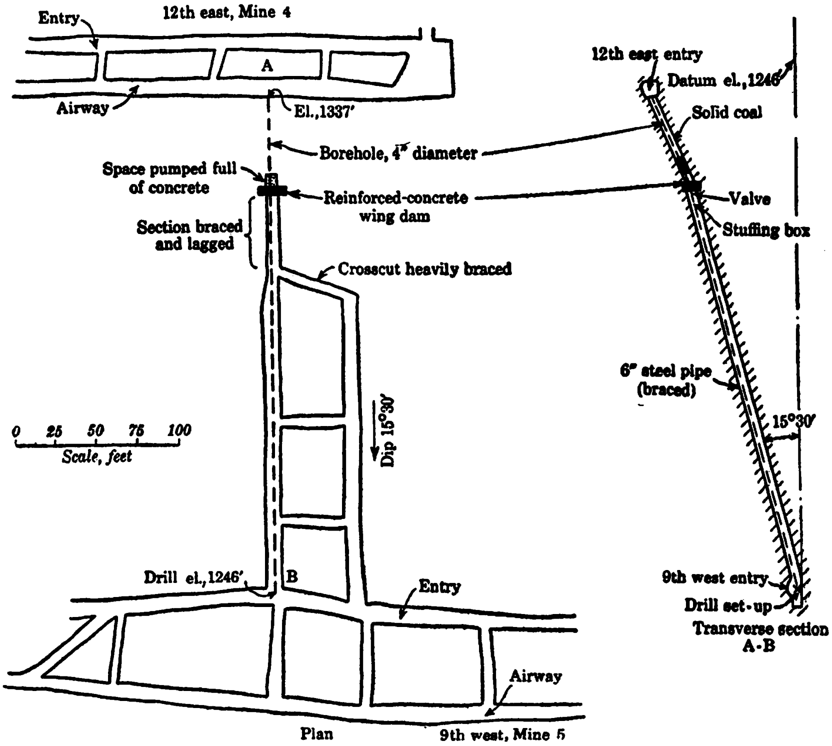

FIGURE 4.—Drilling scheme for unwatering Mine 4 from Mine 5, Roslyn field

odor; it was decided that it was undoubtedly coming through crevices over the 300-foot barrier from Mine 4 and that as lower levels in Mine 5 were driven to the barrier the condition would grow worse, so that a flow of water might occur that could not be handled and the mine would be flooded.

METHOD ADOPTED

The operators thought, therefore, that in the interests of both safety and economy Mine 4 should be unwatered, but to do this in the ordinary way—that is by cleaning up and ventilating the shaft and slope and lowering a pump by stages—would be slow and expensive, so it was decided to tap and void the water by drilling from Mine 5.

Figure 4 shows that the twelfth east level of Mine 4 had been driven nearly 3,000 feet and the thirteenth east level only 900 feet. The ninth west level of Mine 5 had been driven 3,300 feet to the barrier line of Mine 4 and if continued on its regular course would be 313 feet south of and below the twelfth east air course in Mine 4. It was therefore decided to drive the open west entry of Mine 5 300 feet ahead; next, to drive an incline up within 50 feet of the twelfth east air course in Mine 4, if conditions permitted; and then to drill through and tap the water.

Two inclines, 14 feet in width and connected at the faces with a crosscut, were turned off the ninth west level on 50-foot centers. They were driven west 200 feet straight up pitch 5, which was 15° at this point. The outside incline was then stopped, and the inside one narrowed to 6 feet and driven on up with the pick; no explosive was used from this point.

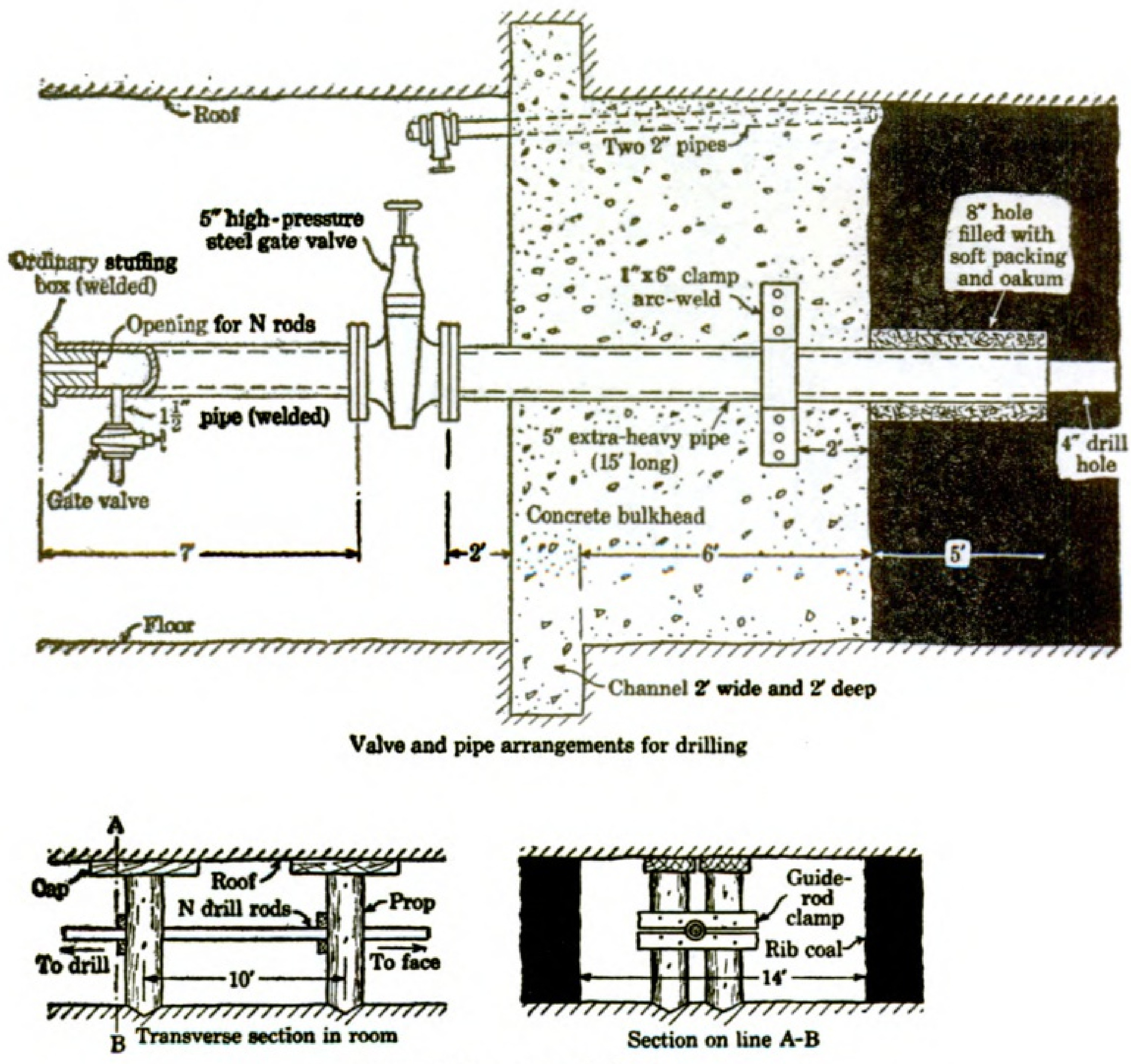

FIGURE 5.—Valve, pipe, and drill-rod arrangements for unwatering Mine 4 from Mine 5, Roslyn field

Three pilot holes, one straight and one on each flank about 20° from the center line, were kept 30 feet ahead at all times. No water was encountered, and the place was kept timbered close to the face with 4-piece square-sets of 10 by 10 inch fir, lagged solidly to prevent the ribs from sloughing. When the place had advanced to about 60 feet of the flooded entry above there was a natural depression in the floor, and it was decided to anchor a concrete bulkhead there. The incline was driven up only 8 feet farther, which left about 52 feet to be drilled. Figure 5 shows the preparation of the face for drilling. A 2½-inch hole was drilled 5 feet straight into the face about 2 feet down from the roof and subsequently enlarged to 8 inches by a special reamer of 2-inch extra-heavy pipe with cutting wings arc-welded to it.

A piece of 5-inch extra-heavy pipe 15 feet long was then inserted and centered in the 8-inch hole. Twelve rings of soft packing were pushed to the back of the hole, and the remainder was filled with oakum around the pipe and tamped solidly. A clamp of 1 by 6 inch soft steel was bolted and arc-welded to the pipe 2 feet from the face of the coal, so that it would be imbedded in the concrete bulkhead.

Eight feet back from the face a channel 2 feet wide and 2 feet deep was cut in both roof and bottom with a pneumatic pick. Each coal rib was channeled 3 feet wide and 3 feet deep. Forms were then built across the incline on the downhill edge of the channels. Two 2-inch pipes were placed in the forms so that the inside ends were almost against the coal face near the roof and the outside ends projected 2 feet outside. The space between the forms and the face was filled with concrete; the material for that at the roof was pumped in through the 2-inch pipes as grouting.

The bulkhead was allowed to set for three weeks; then a mixture of neat cement and water was pumped in through the two 2-inch pipes under a pressure of 300 pounds. This cement appeared around the edges of the bulkhead and even in small crevices in the roof 30 feet down the incline. Three days later water under a pressure of 350 pounds per square inch was let into the 5-inch drill pipe, and no leaks showed.

A 5-inch high-pressure steel gate valve was flanged to the end of the pipe running through the bulkhead to the coal. Another piece of 5-inch pipe 7 feet long was flanged to the gate valve, and an ordinary stuffing box with packing gland was welded to the outside end. A 1½-inch pipe connection was welded to the 5-inch pipe near the stuffing box and fitted with a gate valve.

DRILLING

The management operated a large steam-driven diamond drill in this field; the steam drive was taken off, and a 15-horsepower series-wound 550-volt locomotive motor was installed with suitable gearing and a water rheostat for speed control. It would have been rather awkward to set the drill on the pitch near the bulkhead, so a place was shot out in the bottom just off the ninth west entry and the drill set up and braced securely. (See fig. 4.)

It had been decided to drill a 4-inch hole through to the standing water. For this purpose a drill was made by cutting four spiral slots about 2 feet long equidistant around the circumference of a piece of 2½-inch extra-heavy pipe; tool-steel spirals terminating in cutting ends were welded in these slots, and a short section of miner's fishtail drill was welded to the end as a pilot. The shank of this drill bit was fitted with a threaded connection to be attached direct to the N rods of the diamond drill.

The line of drill rods from the entry to the bulkhead was almost 250 feet long and carried on notched wooden brackets nailed to posts set in pairs every 10 feet. Another notched bracket was nailed over the rods on each set of posts to prevent the rods from buckling under thrust pressure. (See fig. 4.)

The water in Mine 4 was pumped as low as the air line in the shaft would take it (an elevation of 1,837 feet). The twelfth east air course, where the water was to be tapped, was at an elevation of 1,341 feet, indicating a pressure of 217 pounds per square inch. The drill rods passing through the packing gland had a cross-sectional area of 5.21 square inches, so that the thrust on the rods from the pressure of the water was 1,133 pounds. Valve, pipe, and drill-rod arrangements are shown in Figure 5.

A source of water supply from slope 5 had been provided for washing the drill cuttings out of the hole. The pressure on this water could be run up to 350 pounds per square inch and was maintained at about that point, as it was desired that when the drill broke through to the flooded mine the flow of water should be in that direction for a while to wash away trash or small rock.

A man was stationed at the 1½-inch valve to control the return water and cuttings. After about 20 feet the cuttings showed that the drill had run out of the coal into the shale cap rock, which was not much harder to drill than the coal; the hole was put through in 95 minutes.

Rope blocks had been rigged up to ease the drill out of the hole, but it was found that on account of the friction of the long line of drill rods on the supporting brackets a few men could lower them out of the hole and disconnect them. The drill bit was backed out through the gate valve into the stuffing box, the valve was closed, and the job was done.

PUMPING

A 6-inch pipe line had been laid from the bulkhead at the drill hole to a centrifugal pump of 600 gallons per minute capacity under a 665-foot head working at a station within an elevation of 508 feet. As the level of the water in Mine 4 was at an elevation of 1,847 feet the head on the Mine 5 pump was only 326 feet. The pump had been equipped with a variable-speed motor for this service, and the water was lowered in Mine 4 until it would no longer run to Mine 5 pump in sufficient quantity; then another pump was installed in the line in the low point of the ninth west entry, and the water was boosted up to the first pump. Fourteen months of steady pumping lowered the water in Mine 4 to the twelfth east entry, and in the early part of 1929 another tap was made at the bottom of slope 4 from the tenth west entry in Mine 5, which drained the remainder of the water from the mine. This tap was made practically the same way as that described and was against a pressure of 91.5 pounds per square inch.

A similar method was followed subsequently in unwatering the Big Bed mine workings of the same company, which overlie the active workings of Mine 3. Figure 6 shows details.

FLOODING OF ACTIVE MINE BY OLD MINE

In unwatering, if mining operations are in progress along with drilling, certain precautionary measures should be taken, especially when the water is reached and the workmen are employed below the drilling site.

Miners were at work several thousand feet below the site chosen for unwatering an abandoned adjacent mine in an overlying bed that contained enough water to fill the lower active level and its workings. The intervening strata comprised approximately 75 feet of soft, coarse-grained sandstone under a hydrostatic pressure of approximately 80 pounds per square inch. The management believed that a 2-inch hole could be drilled, the pipe and valve connections inserted when the hole reached the water, and the pipe tightened in place by proper caulking methods. The pipe and

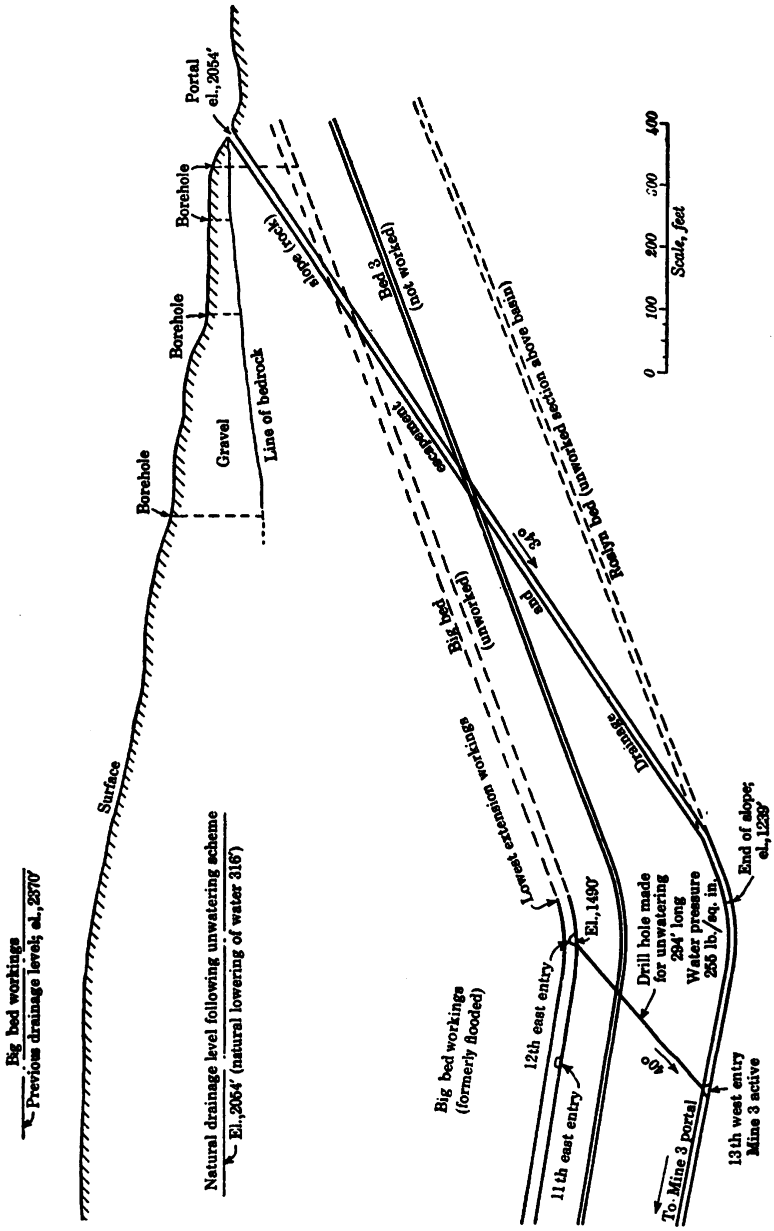

FIGURE 6.—Method of unwatering Big Bed workings above Mine 3, Roslyn field

attachments were ready, and the drilling was successfully done with a steel bit and rods of 1½-inch pipe in 10-foot lengths. However, when the water was reached a movable section of loose material collected around the bit, bringing the total hydrostatic pressure to play on the rod section; in being withdrawn by hand the rod got away from the drillers, was forced out of the hole, and jammed in such a manner that the valve section of pipe could not be inserted immediately. Because of loose material the hole was being enlarged rapidly, and the miners in the lower level had to be withdrawn hastily. A large portion of that level was flooded, and the following rather unique method was employed to close the hole and control the water flow.

A small tree approximately 6 inches in diameter was tapered to a point in a length of approximately 40 feet. Six pieces of 2-inch and 4-inch lumber were nailed at about 4-foot intervals to be used as handholds. Twelve men, six on each side, volunteered to handle the plug and after some effort inserted the point in the collar of the hole; it was forced in and held tightly until a chain block could be attached and the plug tightened, bringing the water under control. A second hole was drilled, but a pipe and valve were properly inserted before the water was tapped. The hole mentioned was almost 5 inches in diameter at the collar before the water was stopped.

PUMP INSTALLATIONS

An unusual amount of pumping is required in Washington coal mines because of the number of beds worked, outcrops in water-soaked gravel, and heavy rainfall. Early mining frequently extracted the coal near the outcrops; and as chain pillars left between levels, unless very thick, eventually run out, the coal must now be left intact for at least one level.

Drainage in western Washington coal mines costs 0.5 to 1.25 per cent of the total cost of a ton of coal, not including power, a large item.

Except for "sinking pumps" most of those used in Washington mines are of centrifugal type, ranging from one to six stages, usually driven direct by alternating current at 220 to 2,300 volts. At some mines pumping is done through several stages, the maximum lifts being approximately 700 feet vertically. Owing to the long pipe lines and their size, considerable detrimental friction head is apparent in some instances. The following sections describe some typical installations.

CENTRIFUGAL PUMPS

At one large open-light mine there are several pump installations in the slope workings. All of these pumps use alternating current, and all motors are nonpermissible.

1. A single-stage centrifugal pump in the return airway of one entry is direct connected to a 220-volt, alternating-current, 15-horsepower induction motor. The wiring is electrically efficient, and closed-type switches are used; but the air return is on a split of air such that in case of fire the smoke could pass over some workmen in another entry. The quarters used are not fireproof or equipped with fire doors. A carbon-tetrachloride fire extinguisher is provided at this installation. A pressure gauge indicates 52 pounds per square inch.

2. A 2-stage centrifugal pump of 65 pounds pressure, driven by a direct-connected, 25-horsepower, 440-volt, alternating-current motor, is installed at the lowest point in the mine. The installation

is electrically efficient, and the feed cable is hung on messenger wire. The pump room is on the return airway, is not fireproofed, and has no fire doors. In all pump sites in this mine smoke from an electrical fire could pass down the manway as the returns are used as manways. Two carbon-tetrachloride fire extinguishers are placed in the pump room.

LACK OF FIRE AND ELECTRICAL PROTECTION

A large closed-light mine employs several pumps. Certain points of installation do not follow practice recommended by the Bureau of Mines, and even in the newer installations the pumping stations should be selected to afford more protection in case of fire. This applies to underground electrical equipment as a whole.

1. The main pumping unit comprises a centrifugal pump which runs about 8 hours each day, delivers 600 gallons per minute against a 665-foot head, and is driven by a direct-connected, 2,200-volt alternating-current, 200-horsepower induction motor. The installation is good and electrically efficient. The pump room is fireproof and is provided with iron fire doors set in concrete and on a separate split of air. The fire protection comprises three soda-acid-type fire extinguishers. Almost no extinguishers serviceable for electrical fires and of the carbon-tetrachloride type were in use. Because of the high electrical conductivity of the soda-acid type they are likely to prove hazardous, and because of the dangerous gases evolved in the use of carbon tetrachloride other nonconductive types of fire extinguishers are more suitable for fighting fires underground near live wires.

2. At one place there is a pump between an entry and the air course into which considerable gas from feeders is bleeding. This pump is driven by a direct-current 500-volt motor, with open-type switches and rheostat. This pump is not in a fireproof chamber; and besides the danger from fire, installations of this type offer serious risk in case of a shutdown of the fan if the gas in the back entry should leak through the stoppings into the entry.

PUMP ROOM AND MINE WORKINGS

At one mine a new pumping unit is a 3-stage centrifugal pump rated at 800 gallons per minute against a 550-foot head, direct connected and driven by a 2,300-volt, 200-horsepower induction motor. This installation has many commendable features compared with the usual type; the pump room is fireproofed and placed between an intake air manway and the main return; a small pipe in the concrete stopping allows intake air to pass over the pump and into the return but is too small to afford much protection in case of fire. The motor is only about 8 feet from the manway, which is too close; this condition could be improved if fire doors were placed on the intake side of the pump room and a small iron regulator door installed in the stopping in addition to the pipe. In case of fire the regulator could readily be opened and the fire doors closed. Fire extinguishers are in place.

PLUNGER PUMPS

At one large gassy mine there are several pump installations in the slope workings, of which three use alternating current and the others 500-volt direct current.

1. A triplex plunger pump is installed relatively near the top of the mine and is chain-driven by a 500-volt, direct-current motor. The switches are of the open type, and the pump is on a separate split; it is housed in a room which is fireproof except for the coal rib on the high side. There are no fire doors, and a soda-acid-type fire extinguisher affords some fire protection.

2. Another pump of 350 gallons per minute capacity is driven by a 65-horsepower, 500-volt, direct-current motor. The line switch is of the open type and on an iron frame. The pump room is directly off the manway and is fireproofed but has no fire doors. A door in a stopping can be opened and leads to the slope return airway. Plenty of water is available in a sump directly under the pump. Smoke from an electrical fire could pass down the intake-air manway. A soda-acid fire extinguisher and a box of dry sand are at hand and could be used in case of fire.

HIGH-CAPACITY HIGH-HEAD CENTRIFUGAL PUMPS

1. The main pumping unit at one mine is a centrifugal pump driven by a 2,200-volt, alternating-current induction motor, with a capacity of 600 gallons per minute against an 840-foot head; it was actually pumping about 800 gallons per minute against a 750-foot head, and 13 hours of operation each day were said to be required at that particular time. The pump is not placed in a fireproof room but is outside the old pump station, which is fireproofed and well protected. A soda-acid fire extinguisher and a box of sand constitute the fire protection.

2. At one large gassy mine the main pumping unit, near the slope bottom, runs about 8 hours daily and consists of a 3-stage centrifugal pump operating at 600 gallons per minute against a 500-foot head and driven by a direct-connected, 2,200-volt, alternating-current, 250-horsepower induction motor. The installation is good and electrically efficient. The pump room is fireproof but has no iron doors. It is on a separate split of air; but the air returns on the main haulageway, which is also the main return airway. Carbon-tetrachloride fire extinguishers afford some fire protection.

PUMP PLACEMENT

1. At a large, gassy mine a timber pump room on the main intake near the slope bottom contains four 6-stage centrifugal pumps direct-connected to four 2,300-volt induction motors. Formerly some of the wiring for lights was hung on nails. This pump room presented considerable fire risk because of its construction and place; all pump rooms should be connected with the return airway and, if feasible, should be on separate splits. A carbon-tetrachloride extinguisher was available in the pump room. Although the high-voltage wiring was in good condition, a hydrant with hose attached placed near the

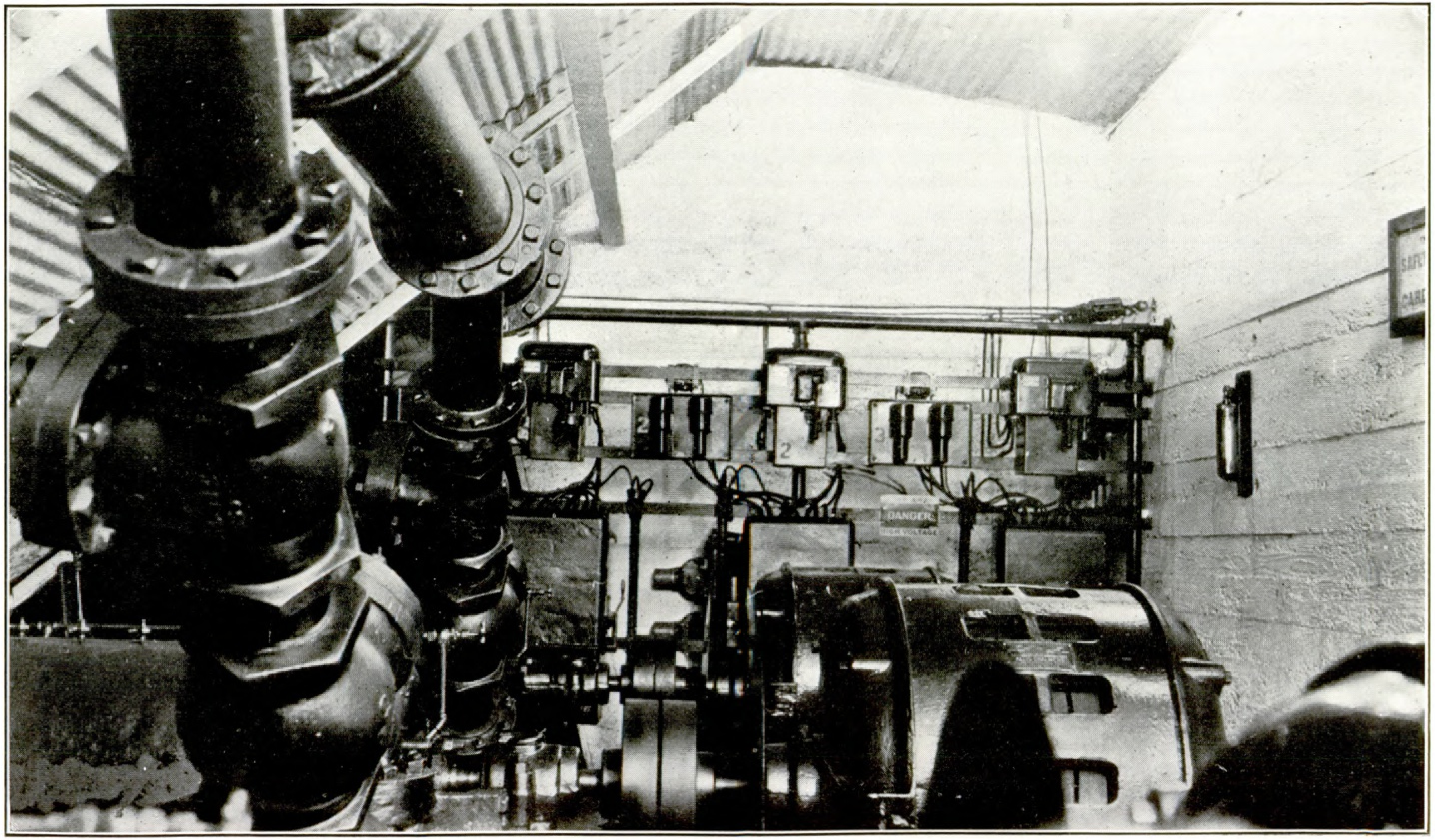

FIGURE 7.—Fireproofed pump installation in Washington coal mine

pump room would decrease the probability that a fire might make headway on the slope.

2. At another gassy mine the main pumping unit is in a chamber cut into the slope pillar and practically on the slope. It is a 3-stage centrifugal pump operating about 8 hours each day at 350 gallons per minute against a 250-foot head and driven by a direct-connected, 440-volt, alternating-current, 50-horsepower induction motor. The pump room, on a separate split of air, is not of fireproof construction and has no iron fire doors. In fact, there is no fire protection of any kind. The open-type electrical switches are on a wooden frame. As the pump room is virtually open on three sides, it could only be fireproofed at considerable expense; and as it is on a separate split, smoke could pass directly into the main-slope return in case of a fire.

FIRE HAZARDS IN PUMP ROOMS

Because of the amount of timber used in many pump stations a serious fire hazard exists. This fact, although not recognized to the extent that it should be, was provided for when the Washington State mining laws were revised, as follows:

Sec. 92 * * *.

All permanent underground pump rooms must be thoroughly lined with fireproof material, unless the same are excavated in solid rock.

Fire protection must be provided according to the State mining laws, Section 85.

There are and have been some excellent pump installations, of which one is shown in Figure 7. This pump room was concreted and fireproofed throughout, was on a separate split of air, and was equipped with fire doors.

It is noteworthy that a year ago the largest coal-producing company in Washington completed the concreting and fireproofing of all pumping stations and quarters for auxiliary electrical equipment installed underground at its mines and provided them with automatically closing iron fire doors. This program has entailed great expense; but in addition to the economic saving that it insures in case of fire, a precedent has been set in the interest of safety that can and should be followed at all mines in the State. Pumping is a serious problem at many mines of the company, and because of the numerous installations careful record is kept of all such operations. This record is entered as a written report signed by the pump man and examining fire boss.

SUMMARY

Unwatering flooded coal mines may be a simple job or a difficult and dangerous one; it may involve only the lowering of a pump as the water recedes or may necessitate tapping water under pressure in workings high above present operations. The regular drainage and pumping of water from mines should not be confused with the unwatering of flooded workings, although drainage pumps may help in such unwatering; therefore this report describes pump installations, including types, rooms, capacity, power required, electrical systems, and fire protection. A few drainage costs are given. Unwatering flooded mines is, at best, costly, difficult, and hazardous, requiring considerable knowledge of the formations and skill in surveying, drilling, handling piping and valves, pumping, and protecting against inrushes of water or gas or of both; in some instances explosive gases have been encountered even in mines that had not been known to liberate such gas, this being true in metal as well as in coal mines.