CHAPTER 1
STRIP MINING (OPEN-CAST MINING) OF COAL

1.1. Introduction

The first steam shovels were introduced to strip mining in 1911, and then there has been a steady increase in the proportion of coal produced by surface mining methods. The bituminous output was produced by strip mining.

The tons per man-shift produced by strip mining methods have always been consistently higher than the tons per man-shift produced by underground methods. In 1960, the average bituminous production per man-shift was 22.9 tons in strip mining as compared with a production of 10.6 tons/man-shift by “deep mining” methods. (“Deep mining” as used in the coal mining industry refers to underground mining methods exclusive of that mined by underground augering from “high-wall” faces). During the same year the average production by augering from high-wall faces was 31.4 tons/man-shift.

The coal beds, which were under the shallowest depths of cover and most easily accessible were the first to be mined and the average depths of overburden which must be removed to expose coal seams has increased year by year. At present some operators are stripping up to 100 ft of overburden and are making plans to strip as much as 120 ft. to cope with the increased amount of rock and dirt which must be handled to expose each ton of coal, stripping machinery of ever increasing size is introduced year by year. A shovel with a dipper capacity of 115 yd$^3$ designed to move 36 million cubic yards of material yearly was put into service and a dragline equipped with a bucket of 85 yd$^3$ capacity was put into service and larger machines are in the design stages.

1.2. Planning a Stripping Operation

Assuming that a block of land has been acquired which contains sufficient coal reserves to warrant the investment in stripping equipment and cleaning plant then the next step is to accumulate information on the surface topography, and the depth
and thickness and characteristics of the coal seam as well as the nature and thickness of the overburden overlying the coal seam.

Aerial topomaps make good base maps on which to plot coal outcrops, and on which to lay out existing property lines, and to locate proposed roads, plant, spoil areas, etc. Locations for exploratory bore holes may also be spotted on these maps.

The nature and thickness of the various strata which must be stripped from the coal seam may be determined by means of diamond drill cores and the physical characteristics and composition of the coal seam may be determined by examination and analysis of diamond drill cores.

**Plant Location**

The factors which influence the location of the plant will be the same as those which affect the locations of a plant for a deep mining operation and include the influence of topography, access to rail or water transportation, sufficient area available for plant construction, availability of a site for refuse disposal, and a favorable location which will keep the haulage distance from the coal face to the plant to the practical minimum.

1.3. **Stripping Equipment Ratio**

The universal factor most used to determine the economics of strip mining is called the “ratio”. This refers to the cubic yards of overburden which must be dug to uncover 1 ton of coal. “Ratios” are as high as 20:1 at some stripping operations. At one Illinois mine as much as 80 ft of overburden is stripped to recover 28 in. of coal.

Many other factors must also be considered, such as the proportion and hardness of the rock in the overburden, the thickness and quality of the coal seam, costs of labor and materials, sale price of the coal, but basically the “ratio” is of the most importance.

**Size and Type**

Topography, coal reserves, expected selling price of the coal, type of overburden, spoil area, and tonnage of coal desired per shift are some of the major factors influencing the selection of equipment.
Once the capacity of a mine has been decided and the major stripping machine chosen, other units such as drills, power shovels for loading coal, and trucks should be selected to build a balanced production cycle.

The equipment must be in balance ratio-wise with the maximum output of the mine and at the same time capable of handling the maximum depth of overburden. In other words, maximum yardage determines size and maximum depth determines the range of the machine.

Coal is a seasonal product and it is necessary to choose equipment which meets maximum production periods; this equipment is usually larger than would be required to maintain a steady average production rate.

**“Case Study”**

**Shovels**

Shovels are available in a wide range of designs and capacities to meet most stripping conditions. *For example,*

A 3-yd$^3$ shovel with a 28-ft boom and a 20-ft dipper handle can cut to a 32-ft height. A 45-yd shovel with a 120-ft boom and a 79-ft dipper handle can cut to a height of 107 ft. The 60-yd shovel which went into service in 1956 has a 150-ft boom and can pile spoil 97 ft high. The 70-yd shovel which began stripping in 1957 has a 140-ft boom and a maximum dumping height of 96 ft.

The giant 115-yd$^3$ shovel which went into service in 1962 has a reach of more than 460 ft and is powered by fifty motors ranging from $\frac{1}{4}$ to 3000 hp. It will remove 3 million yd$^3$ of overburden per month while working in banks more than 100 ft high.

At an Ohio operation a 65-yd shovel operating around the clock exposes 2 million tons of coal per year. This shovel is equipped with a 135-ft boom and a 91-ft handle and can stack spoil 103 ft high. Its dumping reach from the outside of its crawlers is 121 ft. Overburden thickness ranges from 39 to 120 ft.

At another operation a beefed up old 33-yd shovel digs 50 ft of shale and some sandstone without the aid of explosives, averaging 10,050 bank yards per shift or 900,000 yd$^3$/month. The company uses a three man crew on this unit, two operators and a ground man on each of the three shifts. The two operators alternate running and oiling and thus operator fatigue does not become a limiting factor in production.
Draglines

Draglines are available in a wide range of sizes to meet varying conditions. A 2 ¾ yd dragline with a 110-ft boom can dig to a depth of 58 ft and pile spoil to a height of 49 ft above the bottom of the bench on which it is working. A 35-yd unit with a 220-ft boom can dig to a depth of 94 ft and pile spoil 98 ft above the tub (on which the dragline sits).

An 85-yd³ machine was introduced in 1963. It had a 275-ft boom, a 248-ft dumping radius and a 143-ft dumping height. Its working weight was about 5500 tons.

Figure 3: The largest dragline – bucket capacity 85 yd³

Typical Operation

A typical operation using bulldozers and draglines for stripping\(^3\) removes overburden consisting of from 50 to 60 ft of medium hard shale and 20-30 ft of sandstone which is drilled and shot so that about 25 ft of it is cast to the spoil area by the blast. Two heavy bulldozers then team up to remove about another 25 ft of overburden to the spoil area. A diesel dragline with a 165-ft boom and a 14 ½ yd bucket follows the bulldozers and throws material as far away from the high-wall as
possible. This dragline casts an average of about 800 yd$^3$/hr and the two bulldozers push this material to the spoil area.

Flat coal seams and steep slopes cause overburden thickness to increase rapidly as successive cuts advance into the hillside. To meet these difficult conditions the large walking dragline is most useful because of its long dumping range. The disadvantage in using the large dragline is that it must have a suitable base on which to sit and this is sometimes difficult to provide in rocky overburden. This factor must be considered in choosing between a dragline and a shovel.
1.4. Stripping Methods

The three basic stripping systems are the following:

Figure 4: Typical cross-sections of a two-seam stripping operation show how a 42-yd shovel with a 130-ft dumping radius takes the main portion of the overburden on the top seam and a 6-yd high lift machine skins off the thin layer of rock covering the lower coal.

Figure 5: Typical cross-section showing the stripping of gypsum veins (Alabaster, Michigan)
(1) A single stripping shovel traveling on the exposed coal seam digs and removes the overburden ahead of it and piles it in the cut from which coal has previously been removed.

(2) A single dragline travelling on a bench above the coal strips overburden to widen the bench for its travel way for the next cut and also removes the high-wall bench over which it has just travelled to expose the coal seam.

(3) A shovel and a dragline are used in tandem with both travelling on exposed coal. The shovel works ahead of the dragline removing the lower bench to expose the coal seam and piling the spoil in the cut from which coal has previously been removed. The dragline removes the upper portion of the overburden to form another bench and casts the spoil over and behind the spoil piled by the shovel.

Numerous combinations of shovels, draglines, bulldozers, scrapers, and wheel excavators are possible.

“Case Study”

**Back-cast Stripping**

At its Alabaster, Michigan, quarry, the U.S. Gypsum Co. mines three gypsum veins by stripping methods using two draglines, one of which strips and casts the overburden while the other re-casts a portion of the overburden.

Figure 5 is a cross-section through the stripping operation showing the 740M dragline which is equipped with an 11-yd bucket on a 200-ft boom and the 720M dragline which is equipped with a 5-yd bucket on a 135-ft boom.

The gypsum veins are loaded out onto rail cars by power shovels. The stripping ratio, including both primary overburden and shale dividers, averages 1.85 yd³/ton of gypsum rock. To produce the average annual production of a million tons of gypsum requires stripping about 1,850,000 yd³ of waste.

The percentage breakdown of costs is as follows:

<table>
<thead>
<tr>
<th></th>
<th>% of total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stripping</td>
<td>32</td>
</tr>
<tr>
<td>Drill and blast</td>
<td>15</td>
</tr>
<tr>
<td>Rock loading</td>
<td>17</td>
</tr>
<tr>
<td>Haulage</td>
<td>21</td>
</tr>
<tr>
<td>Supv. and engrg</td>
<td>7</td>
</tr>
<tr>
<td>Miscellaneous</td>
<td>8</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>100 %</strong></td>
</tr>
</tbody>
</table>
Shovel-dragline Combination

Figure 6 shows the stripping sequence in stripping and mining the 36-in No. 6 bed at Blue Crystal Mines, Inc., in Coshocton Country, Ohio.

Figure 6(a) show the method used in making the initial cut near the outcrop. The diesel dragline is equipped with a 90-ft boom and a 4-yd bucket. Using a 30° boom angle this unit has a maximum digging reach of 114 ft but is given a restricted reach of 80 ft for vertical dropping of the bucket. It can dump at any distance up to 90 ft and at a maximum height of 38 ft.

As shown in Figure 6 (a) the initial cut is made at such a location along the outcrop as to provide for natural drainage from the pit, if possible. The dragline is positioned so as to develop a 100-ft wide bench of coal, not including the coal near the outcrop which would be soft and of poor quality.

The first cut C₁ is made by down-cutting along the high-wall 80 ft from the machine and dumping the spoil S₁, 90 ft to the rear. Assuming a surface grade of 15 percent which is the average for the area, a 28-ft high-wall with a 1 in 3 slope is developed and a 35-ft wide coal bench is exposed.

The second cut, C₂, can be made from the initial position if that is desirable, opening a coal bench 65 ft wide, S₂ spoil is dumped at the same point, thereby developing a bank 39 ft high and 122 ft wide. These values are based on the loose soil, clay, and soft shale of the overburden increasing (swelling) 15 percent in volume, and on an assumed slope of 1 ½ :1.

The high sulphur materials which lie immediately above the coal bed are the last materials excavated and is buried by the S₂ spoil. The toxic materials from C₂ are dumped along the side of the spoil bank so as to permit later coverage.

A diesel shovel follows up the initial dragline work. This shovel is equipped with a 48-ft boom, a 34-ft dipper handle or “stick”, and a 3 ½ yd dipper. With a 45° boom angle the shovel has a maximum digging radius of 62 ft at a maximum height of 49 ft, a floor level digging radius of 33 ft, and a maximum dumping radius of 52 ft at a maximum height of 38 ft.

As shown in Figure 6 (b) the shovel is positioned on the pit bottom 7 ft from the toe of the high-wall left by the dragline. The shovel removes C₃ and dumps the spoil to form S₃, thereby uncovering approximately 24 ft of coal bench. This material is somewhat tougher than handled by the dragline, but swell is still 15 percent. The slope of the piled material is steeper than that of S₂, being 1 ¼ :1, so S₃ becomes a
spoil bank 30 ft high and 74 ft wide. After removing C₃ for as great a distance along the high-wall as is desirable the stripping shovel returns to the initial operation point for removal of C₄. The material along the toe of the high-wall is cast into the bottom following removal of the coal under C₃, thereby providing for coverage of the acidic materials. From position 3 the shovel removes C₄ material which is still largely unconsolidated material which does not require blasting, and dumps it to form pile S₄, uncovering the remaining 41 ft of the 65-ft wide pit. This spoil bank is 37 ft high (based on 15 percent swell and 1 ¼:1 slop) and together with S₃ gives a total bank width of 122 ft.

Figure 6: Stripping a coal seam
(a) Opening a 100-ft wide coal bench at the outcrop with a 4-yd$^3$ dragline.
(b) Uncovering a 65-ft wide coal bench with a 38-ft high-wall, using a 3 ½ yd$^3$
stripper shovel.
(c) Stripping a 110-ft wide coal bench with a 52-ft high-wall, using a 10-yd$^3$ dragline.

Removal of overburden beyond the average 35-ft high-wall left by the
stripping shovel is accomplished by an electric dragline equipped with a 140-ft boom
and a 10-yd$^3$ bucket. With a 30° boom angle this unit has a maximum reach of 152 ft
but a restricted reach of 131 ft for vertical dropping of the bucket. It can dump at any
distance up to 131 ft and at a maximum height of 65 ft.

As shown in Figure 6 (c) a 100-ft bench of coal is uncovered by the dragline in
several steps. The overburden is tough and requires blasting for fragmentation.

From position 4 the dragline digs a “keyway”, shown as C$_5$, with an average
width of 25 ft and from 25 to 35 ft deep, adjacent to the higher ground so as to
develop a practically vertical high-wall. This keyway provides two faces for attack on
the remainder of the cut face. Spoil from C$_5$ is dumped into the pit adjacent to the old
high-wall to provide a future operations footing for the dragline.

Due to the nature of the material, the swell figure is 20 percent while the spoil
bank slope is 1 ¼:1. The space occupied by the C$_5$ material is indicated as S$_5$, and
includes the cross-hatched area adjacent to the old high-wall.

The dragline then moves to any position above C$_6$ convenient for removal of
that material. Figure 6 (c) indicates a position 5 which could be used to remove all of
C$_6$, the spoil being dumped to form bank S$_6$. However, the dragline could move
toward the spoil band and work from a position on S$_5$ if necessary, casting material
beyond S$_6$. Removal of C$_6$ will involve recasting of the cross-hatched portion of S$_5$
adjacent to the old high-wall in order to remove coal adjacent to S$_5$. The spoil banks
will stand with practically vertical faces for short periods of time, but the coal must be
removed quickly to avoid trouble from slides.

Figure 6 (c) indicates a 52-ft high-wall on completion of the C$_5$-C$_6$ pit. A
maximum of 60-65 ft of overburden can be removed economically under today’s
market conditions. If the grade conditions of Figure 6 (c) remained constant, a 75-ft
wide pit could be developed to the right with a 65-ft high-wall. Frequently the grade
diminishes as the crest of a knob is approached, and another 100-ft pit might be
developed without exceeding 65 ft of overburden.
**Draglines vs. Shovels**

Some advantages of draglines over shovels for stripping applications are the following:

1. The dragline moves overland with less difficulty than the shovel and is therefore more useful where the coal reserves lie in various scattered small bodies.
2. The dragline is more effective in performing other work required in connection with the stripping operations such as building roads and ditches.
3. The dragline can clean a rough coal seam and leave it ready for loading.

The principal advantage of the shovel is that it can dig overburden which is tight or poorly broken and may make unnecessary the shooting of a portion of the overburden, or reduce the amount of shooting which is required.

**Comparative Stripping Costs – Large Shovels vs. Small Shovels**

Stewart and McDowell\(^{(8)}\) have used the actual annual output of a 70-yd shovel equipped with a 140-ft boom as compared with the annual output of a 13-yd shovel equipped with a 95-ft boom to compute typical comparative costs of moving overburden.

During the year of 1956 the 13-yd shovel moved an average of 360,000 yd\(^{3}\)/month and 600 yd\(^{3}\)/digging hour. The average overburden depth was 33 ft.

During 1957-1958, the 70-yd shovel moved an average of 1582000 yd\(^{3}\)/month or 3250 yd\(^{3}\)/hr. average overburden depth was 61 ft.

The average output of the 70-yd machine was 4.4 times the output of the smaller shovel. Based on the same three-man operating crew and the same pay scale the direct operating labor cost for the large machine was only about 24 percent of that for the small machine.

**Cost of Moving a Cubic Yard of Overburden**

The estimated operating costs have been prepared on the basis of the following (the 13-yd shovel is designated as Type 5320 and the 70-yd shovel is designated as Type 5760):

1. Labor – assume the same for both shovels as shown on the following schedule.
2. Electric power – Cost equals $0.0125 per kWh.
3. Maintenance – for the Type 5320 over a 12 to 15-year period, a cost figure of $0.03 per yd$^3$ has been used, which is a reasonable average. There are no long time averages available as yet for the Type 5760. Because of such things as modern design, welded construction and better steels, it is reasonable to assume an average maintenance cost of $0.025 per yd$^3$ for the Type 5760.

4. Amortization – It is assumed that the Type 5320 has been completely written off at this time and there would not be amortization charges applicable to this unit. The guessed present day installed cost of the Type 5760 is $3,300,000 and amortizing this over 15 years of 11 months of 720 operating hours each, the amortization charges on this unit will be $27.78 per operating hour.

The following table shows the estimated operating cost for the two shovel units covered by this analysis study:

<table>
<thead>
<tr>
<th></th>
<th>Type 5320</th>
<th>Type 5760</th>
</tr>
</thead>
<tbody>
<tr>
<td>Operator - at $4.5000</td>
<td>$4.50</td>
<td>$4.50</td>
</tr>
<tr>
<td>Oiler - at 4.0000</td>
<td>4.00</td>
<td>4.00</td>
</tr>
<tr>
<td>Pitman - at 3.7500</td>
<td>3.75</td>
<td>3.75</td>
</tr>
<tr>
<td>200 kWh - at 0.0125</td>
<td>2.50</td>
<td>-</td>
</tr>
<tr>
<td>1000 kWh - at 0.0125</td>
<td>-</td>
<td>12.50</td>
</tr>
<tr>
<td>Maintenance - at 0.0300</td>
<td>15.00</td>
<td>-</td>
</tr>
<tr>
<td>Maintenance - at 0.0250</td>
<td>-</td>
<td>56.50</td>
</tr>
<tr>
<td>Amortization -</td>
<td>-</td>
<td>27.78</td>
</tr>
<tr>
<td>Total per hour</td>
<td>$29.75</td>
<td>$109.03</td>
</tr>
<tr>
<td>yd$^3$ per hour</td>
<td>500</td>
<td>2250</td>
</tr>
<tr>
<td>per yd$^3$</td>
<td>$0.0595</td>
<td>$0.0485</td>
</tr>
</tbody>
</table>

1.5. Performance of Large Stripping Equipping Equipment

The following tables (1-3) summarize the results of a survey made by the American Mining Congress Committee on Strip Mining on shovels and draglines in operation in 1959.
### TABLE 1: Capacity per Yard of Burket or Dipper Capacity

<table>
<thead>
<tr>
<th>Shovels</th>
<th>Draglines</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bucket capacity, (yd³)</td>
<td>Capacity/hr/yard</td>
</tr>
<tr>
<td>Capacity</td>
<td>Bucket capacity, (yd³)</td>
</tr>
<tr>
<td>10 – 15</td>
<td>38</td>
</tr>
<tr>
<td>16 – 20</td>
<td>42</td>
</tr>
<tr>
<td>21 – 25</td>
<td>39</td>
</tr>
<tr>
<td>26 – 30</td>
<td>42</td>
</tr>
<tr>
<td>31 – 35</td>
<td>43</td>
</tr>
<tr>
<td>36 – 40</td>
<td>37</td>
</tr>
<tr>
<td>41 – 45</td>
<td>42</td>
</tr>
<tr>
<td>55</td>
<td>42</td>
</tr>
<tr>
<td>65</td>
<td>46</td>
</tr>
<tr>
<td>70</td>
<td>44</td>
</tr>
<tr>
<td><strong>Average</strong></td>
<td><strong>42</strong></td>
</tr>
</tbody>
</table>

### TABLE 2: Boom Length

<table>
<thead>
<tr>
<th>Shovels</th>
<th>Draglines</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bucket capacity, (yd³)</td>
<td>Length (ft)</td>
</tr>
<tr>
<td>Length (ft)</td>
<td>Bucket capacity, (yd³)</td>
</tr>
<tr>
<td>10 – 15</td>
<td>90</td>
</tr>
<tr>
<td>16 – 20</td>
<td>96</td>
</tr>
<tr>
<td>21 – 25</td>
<td>102</td>
</tr>
<tr>
<td>26 – 30</td>
<td>108</td>
</tr>
<tr>
<td>31 – 35</td>
<td>108</td>
</tr>
<tr>
<td>36 – 40</td>
<td>119</td>
</tr>
<tr>
<td>41 – 45</td>
<td>117</td>
</tr>
<tr>
<td>55</td>
<td>145</td>
</tr>
<tr>
<td>65</td>
<td>158</td>
</tr>
<tr>
<td>70</td>
<td>140</td>
</tr>
<tr>
<td><strong>Average</strong></td>
<td><strong>113</strong></td>
</tr>
</tbody>
</table>

### TABLE 3: Percent Running Time

<table>
<thead>
<tr>
<th>Shovels</th>
<th>Draglines</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bucket capacity, (yd³)</td>
<td>%</td>
</tr>
<tr>
<td>%</td>
<td>Bucket capacity, (yd³)</td>
</tr>
<tr>
<td>10 – 15</td>
<td>75</td>
</tr>
<tr>
<td>16 – 20</td>
<td>76</td>
</tr>
<tr>
<td>21 – 25</td>
<td>78</td>
</tr>
<tr>
<td>26 – 30</td>
<td>80</td>
</tr>
<tr>
<td>31 – 35</td>
<td>77</td>
</tr>
<tr>
<td>36 – 40</td>
<td>77</td>
</tr>
<tr>
<td>41 – 45</td>
<td>78</td>
</tr>
<tr>
<td>55</td>
<td>78</td>
</tr>
<tr>
<td>65</td>
<td>77</td>
</tr>
<tr>
<td>70</td>
<td>-</td>
</tr>
</tbody>
</table>

The following treatment, dealing with a computer method for determining the proper sizes of equipment to select for the most economical stripping operations, is taken by Henry Rumfelt, A. & M. College.

1.6.1. Analysis Methods

The objective is to find a method of providing a preliminary simple overcasting analysis for a stripping prospect. The approach to the problem employs indicated trends in the relationship of the weight of the machine to its ability to do stripping work. The ability to do work is established through “MUF” numbers. Also, the approach employs situation where the geometry of each cut and spoils section is assumed to take certain defined relationships for varying overburden depths. Slopes are considered to be stable which means such practical factors as the mechanics of soils are neglected for the sake of convenience.

Assumptions

Pits that follow a straight line when projected on a horizontal plane are referred to as “straightaway”. It is assumed that all sections pertain to straightaway cuts, which in turn means that areas can be treated relatively as volumes. Volumes of overburden are expressed in virgin cubic yards, when material is displaced from its virgin state to a spoil section it usually occupies a larger volume than in situ. The difference is called “swell” and it is expressed in percentages of the original volume. For example, if the original cut volume is denoted \( V \text{ yd}^3 \) and the spoil volume is 1.2 \( V \text{ yd}^3 \), then the “swell” is a positive 20 percent.

Usefulness Factor Concept (Maximum Usefulness Factor or “MUF”)

A value concept of a machine is arbitrarily established, consisting of the product of the nominal dipper size of the shovel times a functional dumping reach. A shovel’s capability to negotiate cuts in deep overburden is limited by its ability to dispose of the spoil in most instances. Thus the dumping reach, along with its respective dumping height, is significant.
The angle of repose of the spoil material must be taken into consideration. It will vary in actual practice among different mines, jobs, and materials. However, a slope frequently found in practice and used in planning of 1.25/1 is used.

“Case Study”

**Shovels**

The relationship of the shovel’s geometry in a strip pit is shown diagrammatically in Figure 7. To gain maximum advantage in constructing spoil piles the shovel is placed so that its tracks which are adjacent the spoil are as close as possible to the deposit rib. A vertical plane passed through the line of the rib of the deposit would contain the point designated as circle T. The dumping reach r is thereby established for each shovel analyzed. The MUFs for any one shovel is defined as the product of the dipper in cubic yards and the dumping reach in feet, shown as r. In other words the MUFs are equal to the load moment about point circle T in terms of cubic yards times feet. Letter designations are assigned the machines studies, and the results of the computations together with machine gross weights are tabulated in Table 8.

![Figure 7: Shovel reach diagram.](image1)

![Figure 8: Dragline reach diagram.](image2)
### Table 8: Summary Tabulation Shovel Data

<table>
<thead>
<tr>
<th>Shovel designation</th>
<th>Working weight (lb)</th>
<th>MUFs (yd³ ft)</th>
</tr>
</thead>
<tbody>
<tr>
<td>a</td>
<td>1,785,000</td>
<td>2156</td>
</tr>
<tr>
<td>b</td>
<td>2,030,000</td>
<td>1740</td>
</tr>
<tr>
<td>c</td>
<td>3,070,000</td>
<td>3938</td>
</tr>
<tr>
<td>d</td>
<td>3,345,000</td>
<td>4350</td>
</tr>
<tr>
<td>e</td>
<td>4,950,000</td>
<td>6734</td>
</tr>
<tr>
<td>f</td>
<td>5,790,000</td>
<td>7630</td>
</tr>
<tr>
<td>g</td>
<td>6,000,000</td>
<td>8264</td>
</tr>
<tr>
<td>h</td>
<td>6,558,000</td>
<td>8580</td>
</tr>
<tr>
<td>i</td>
<td>7,700,000</td>
<td>9900</td>
</tr>
<tr>
<td>j</td>
<td>7,875,000</td>
<td>11,720</td>
</tr>
<tr>
<td>k</td>
<td>8,585,000</td>
<td>-</td>
</tr>
<tr>
<td>l</td>
<td>11,600,000</td>
<td>15,419</td>
</tr>
<tr>
<td>m</td>
<td>13,900,000</td>
<td>20,102</td>
</tr>
</tbody>
</table>

Figure 9: Relationships of gross machine and “MUF” numbers. Graph for shovels.

Figure 10: Relationships of gross machine weight and “MUF” numbers. Graph for...
MUF vs. Gross Shovel Weight

The graph shown on the rectangular coordinate line chart, Figure 9, depicts the trend where the ordinate represents the MUFs and the abscissa represents the gross machine weight (shovel). The result is significant in that the curve appears to follow generally a straight line regardless of manufacture or size range of machines. According to the curve, each MUFs unit requires 745 lb. gross weight in the shovel. In determining the curve, the data were first plotted on a large scale work graph. It was then observed that the points representing shovels actually working in the field were more faithful to the trend line (i.e. they had less “scatter”) than shovels existing only on paper. Consequently, all points, both those representing existing and “paper” machines, are not weighed equally. The points for existing machines are given more “weight” than those for “paper” machines. For this reason the curve was not determined by strict use of the method of least squares as is the practice in certain statistical plotting.

Table 9: Summary Tabulation Dragline Data

<table>
<thead>
<tr>
<th>Dragline designation</th>
<th>Working weight (lb.)</th>
<th>MUFd (yd³ .ft)</th>
</tr>
</thead>
<tbody>
<tr>
<td>a</td>
<td>375,000</td>
<td>587</td>
</tr>
<tr>
<td>b</td>
<td>450,000</td>
<td>760</td>
</tr>
<tr>
<td>c</td>
<td>550,000</td>
<td>835</td>
</tr>
<tr>
<td>d</td>
<td>640,000</td>
<td>1240</td>
</tr>
<tr>
<td>e</td>
<td>695,000</td>
<td>1080</td>
</tr>
<tr>
<td>f</td>
<td>840,000</td>
<td>1370</td>
</tr>
<tr>
<td>g</td>
<td>1,274,000</td>
<td>2060</td>
</tr>
<tr>
<td>h</td>
<td>1,299,000</td>
<td>1970</td>
</tr>
<tr>
<td>i</td>
<td>1,460,000</td>
<td>3680</td>
</tr>
<tr>
<td>j</td>
<td>1,467,000</td>
<td>2560</td>
</tr>
<tr>
<td>k</td>
<td>1,600,000</td>
<td>2610</td>
</tr>
<tr>
<td>l</td>
<td>1,750,000</td>
<td>4330</td>
</tr>
<tr>
<td>m</td>
<td>1,900,000</td>
<td>3100</td>
</tr>
<tr>
<td>n</td>
<td>1,965,000</td>
<td>3440</td>
</tr>
<tr>
<td>o</td>
<td>2,460,000</td>
<td>4410</td>
</tr>
<tr>
<td>p</td>
<td>2,650,000</td>
<td>4130</td>
</tr>
<tr>
<td>q</td>
<td>2,930,000</td>
<td>5120</td>
</tr>
<tr>
<td>r</td>
<td>3,050,000</td>
<td>6240</td>
</tr>
<tr>
<td>s</td>
<td>3,175,000</td>
<td>6320</td>
</tr>
<tr>
<td>t</td>
<td>3,200,000</td>
<td>7110</td>
</tr>
<tr>
<td>u</td>
<td>3,335,000</td>
<td>6500</td>
</tr>
<tr>
<td>v</td>
<td>3,730,000</td>
<td>7640</td>
</tr>
</tbody>
</table>
Draglines

As with the shovel, the dragline’s ability to negotiate cuts in deep overburden is limited by its ability to dispose of the spoil in most circumstances. The dragline normally works from the surface of the cut or a bench floor slightly below the surface. The machine’s dumping height in this situation is important but it is not an influencing factor. For the type cut section and operation visualized, the dumping reach $r$, Figure 8 is the controlling factor.

Walking draglines are mounted on circular bases called “tubs”. Different models within any one manufacturer’s line and variations among the manufacturers result in different ground to base bearing pressures. In order to have a standardized basis for comparison, the tub diameters are varied from actual to a hypothetical whereby bearing pressure are uniformly maintained at 10 psi.

With the hypothetical diameter and allowing 5 ft safety distance between the top edge of the old high wall and the near point of the tub a moment center (circle T) is established. Measuring from circle T to the dumping point gives a moment arm $r$ for each dragline considered.

A figure of 4750 lb/nominal dragline bucket size is taken to represent the unit weight of bucket plus load contents. The specified suspended load of the machine is divided by 4750 to give the nominal capacity in cubic yards. This resulting from bucket capacity times the arm $r$ (distance in feet) gives the maximum usefulness factor for each stripping dragline. Thus, the MUFd, maximum usefulness factor for a dragline, is defined as the product of the nominal bucket size in cubic yards and the dumping reach $r$. It is equal to the load moment about circle T in terms of cubic yards times feet.

MUF vs. Gross Dragline Weight

Letter designations are also assigned the draglines studied. Tabulations of MUFd figures for the analyzed draglines are shown in Table 8. Only electric powered draglines are considered.

The graphs shown on the rectangular coordinate line chart, Figure 10, depict the trends where the ordinate represents the MUFd figures and the abscissa represents the Gross Machine Weight (dragline). There are two curves shown, each of which follows generally a straight line.
The solid line, curve A, represents the trend resulting from the specifications data of the dominant manufacturers’ models which are in operation. As with the operating shovel’s data, the data of existing dragline models when plotted are quite faithful to the line. The dash line, curve B, represents a more advantageous trend which could possibly become the significant one for newer and especially larger draglines rather than curve A. Curve B is not as well defined as curve A and is established more or less arbitrarily by taking into account recent specifications of newly announced larger and some uprated machines.

The slope of curve A figures 1/575 and the slope of curve B figures 1/467 which means that 575 lb. of machine weight are required for each MUFd in the former case and 467 lb. would be required similarly in the latter.

**Pit Section – MUF Relationships**

The foregoing demonstrates unexpectedly simple but at the same time logical relationships between the usefulness numbers (MUF) and the gross weights of the machines analyzed. To accomplish the stated objective it is necessary to next demonstrate a relationship between the pit section geometry and the required MUF numbers for varying depths of overburden. The demonstrations take into account both shovel and dragline type operations. The numbers would be related, in turn, to projected machine gross weights which, in effect, establish relationships between stripping machinery mass and overburden volumes (or depths). Therefore, those mentioned in this section are the required numbers for the hypothetical sections being studied, whereas the ones determined in the preceding section are taken from current or recent machines for the purposes of defining trends.

**Assumptions**

In order to determine a relationship between the MUF numbers of the required machines for different depths of overburden hypothetical situations with a number of assumptions are established. From Figure 11 “Shovel section”, a generalized formula to give dimension \( r \) in terms of \( H \), \( t \), and \( W \) is derived. First, assume the deposit is bituminous coal, of thickness \( t \) feet and the cut width of \( W \) feet. The spoil angle of repose is 38.5° (1.25 in 1 slope). The berm width \( b \) does not enter into the computation because it is assumed to be sufficiently small that the digging
effectiveness of the shovel from its indicated position is not affected. A similar viewpoint is taken in assuming the high wall slope of 1 in 3.

If \( r = \text{reach (as shown in Figure 11)} \) in feet,
\[ S = \text{swell in percentage}, \]
\[ H = \text{depth of overburden in feet} \]
\[ W = \text{cut width in feet} \]
\[ t = \text{deposit thickness in feet}, \]
the resulting formula is as follows:

\[
r = [1.25] \times \left[ \left( 1 + \frac{S}{100} \right) H - t + \frac{W}{5} \right]
\]

(A)

Figure 11: Shovel cross-section

In order to deal with varying dipper sizes “Estimated Shovel Output Table” is constructed. To facilitate computations, selected factors are assumed constants. The factors selected are those which can usually be closely approximated for a given prospect once the type machine is selected and a general operating approach is determined. For this example, it is assumed that regardless of shovel size the average cycle time will be maintained at 56 sec, the dipper factor will always be 80 percent and the monthly operating factor will be kept at 85 percent. The construction in the table shows that in line with these assumptions the expected monthly output will be 31,200 D yd³, were D is the nominal dipper size in cubic yards.
ESTIMATED SHOVEL OUTPUT TABLE

<table>
<thead>
<tr>
<th></th>
<th>D</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dipper size (yd³)</td>
<td></td>
</tr>
<tr>
<td>Dipper factor</td>
<td>80 %</td>
</tr>
<tr>
<td>Dipper load, (yd³), bank measure</td>
<td>0.8D</td>
</tr>
<tr>
<td>Cyclic time (sec)</td>
<td>56</td>
</tr>
<tr>
<td>Passes per hour</td>
<td>64</td>
</tr>
<tr>
<td>Theoretical hourly output (yd³) bank measure</td>
<td>51D</td>
</tr>
<tr>
<td>Scheduled monthly hours of operation</td>
<td>720</td>
</tr>
<tr>
<td>Theoretical monthly output (yd³), bank measure</td>
<td>36,800D</td>
</tr>
<tr>
<td>Monthly expected operating factor</td>
<td>85%</td>
</tr>
<tr>
<td>Expected actual output per month (yd³) bank measure</td>
<td>31,200D</td>
</tr>
</tbody>
</table>

Where, \( Q_c \) = the required net tons of cleaned coal required per month,
\( H \) = the overburden depth in feet, and
\( L \) = the yield in net tons of cleaned coal per acre,
then
\[
D = \frac{(Q_c)(1613)(H)}{L}(31,200) \quad (B)
\]

The required MUFs for the shovel at any overburden depth, \( H \), is given by the product of equations (A) and (B).

\[
MUFs = rD = \frac{[1.25 \times [(1+S/100)(H) - t + W/5]]}{((Q_c)(1613)(H)/(L)(31,200))} \quad (C)
\]

From Figure 12 “Dragline Cross-section” a generalized formula to give dimension \( r \) in terms of \( H \), \( t \), and \( W \) is derived on p.427. First, it is assumed the high-wall slope will remain at a fixed figure of 1 in 3 and that 10 ft of surface will be removed to provide an operating bench. Then, the following are taken as variables:
the deposit is bituminous coal of thickness \( t \) ft, and the cut width is of \( W \) ft. Angle \( \theta \) of 38.5° represents the angle of spoil repose.

If \( r = \text{reach (as shown in Figure 12) in feet}, \)
\( S = \text{swell in parentage}, \)
\( H = \text{depth of overburden in feet}, \)
\( W = \text{cut width in feet}, \)
\( t = \text{deposit thickness in feet}, \)

the resulting formula is as follows:

\[
r = [0.33H – 3.3] + [1.25 \times (1 + S/100)(H) – t + W/5] \quad (D)
\]

In order to deal with varying bucket sizes, “Estimated Dragline Output Table” is constructed in which a number of factors are also made constant. For example, it is assumed that regardless of size of dragline, the cycle time will be maintained at 58 sec, the bucket factor will always be 80 percent, and the monthly operating factor will be kept at 85 percent. In line with the method of computing and the assumptions, the expected monthly output will be 30,400 B yd³ where \( B \) is the nominal bucket size in cubic yards.

<table>
<thead>
<tr>
<th>Bucket size (yd³)</th>
<th>B</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bucket factor</td>
<td>80%</td>
</tr>
<tr>
<td>Bucket load (yd³), bank measure</td>
<td>0.8B</td>
</tr>
<tr>
<td>Cyclic time (sec)</td>
<td>58</td>
</tr>
<tr>
<td>Passes per hour</td>
<td>62</td>
</tr>
<tr>
<td>Theoretical hourly output (yd³)</td>
<td>49.5B</td>
</tr>
<tr>
<td>Scheduled monthly hours of work</td>
<td>720</td>
</tr>
<tr>
<td>Theoretical monthly output (yd³)</td>
<td>35,800B</td>
</tr>
<tr>
<td>Monthly expected operating factor</td>
<td>85%</td>
</tr>
<tr>
<td>Expected actual monthly output (yd³)</td>
<td>30,400B</td>
</tr>
</tbody>
</table>

Where \( Q_c = \) the net tons of cleaned coal per month,

\( H = \) the overburden depth in feet, and

\( L = \) the yield per acre in net tons of cleaned coal,

then

\[
B = (Q_c)(1613)(H)(L)(30,400) \quad (E)
\]

For the derivation of equation (E) see p. 428.

The required MUFd for the dragline at any overburden depth, \( H \), is given by the product of equations (D) and (E).
\[ MUFd = r \cdot B \]
\[ = \left\{ [0.33H - 3.33] + [1.25] \times [(1 + S/100)(H) - t + W/5] \right\} \times \{(Qc)(1613)(H)/(L)(30,400)\} \] 

1.6.2. Illustrative Example

The employing of the developed information may be demonstrated in part by an example. The example selected is well within the range of the prepared data.

Take an imaginary coal strip mining prospect compatible with the output tables and the cut sections already constructed. Assuming information is known about the property which permits establishing as constants certain of the variables in the equations which have been derived. Further assume the natural dispositions of the coal and overburden are such as to appear at the outset to provide approximately equal advantage to shovel and dragline. However, it is tentatively decided to first treat with the shovel type operations.

The 5 ft (t) coal seam is expected to yield 7500 (L) net tons of clean coal per acre. The spoil angle of repose remains 38.5° from horizontal. If mined with a dragline the cut widths (W) would be 80 ft and if with a shovel they would be 60 ft wide. The average production rate being considered is 83,500 (Qc) net tons of cleaned coal per month. The overburden depths (H) vary from 60 ft to 120 ft.

Since there are several other involved aspects of the problem, it is desired to know prior to more detailed study the approximate specifications of the shovels that would be required for different overburden depths. For comparison purposes it is desired to know the relative MUF requirements of the shovels and of the shovels and of the draglines for the same overburden depths. Furthermore, the swell is expected to be 20 percent but it is desired to know what effect would result should the swell turn out to be something greater than the 20 percent – say 25, 30, or even 35 percent.

Appropriate conversions from variables to constants were made in the formulae and the digital computer was employed through the following programming:

Program A. Equation (A) which gives the reach requirement (r) for the shovel section was solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four separate groups:

- First group, holds the swell (S) to 20 percent.
- Second group, holds the swell (S) to 25 percent.
Third group, holds the swell (S) to 30 percent.
Fourth group, holds the swell (S) to 35 percent.

*Program B.* Equation (B) which gives the dipper size requirement (D) for the shovel section is solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft.

*Program C.* Equation (C) which gives the MUFs numbers [a product of equation (A) and equation (B)] for the shovel section is solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four separate group:
- First group, holds the swell (S) to 20 percent.
- Second group, holds the swell (S) to 25 percent.
- Third group, holds the swell (S) to 30 percent.
- Fourth group, holds the swell (S) to 35 percent.

*Program D.* Equation (D) which gives the reach requirement, (r), for the dragline section for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four groups.
- First group, holds the swell (S) to 20 percent.
- Second group, holds the swell (S) to 25 percent.
- Third group, holds the swell (S) to 30 percent.
- Fourth group, holds the swell (S) to 35 percent.

*Program E.* Equation (E) which gives the bucket requirement (B), for the dragline section is solved for different overburden depths from 60 to 120 ft in increments of 2 ft.

*Program F.* Equation (F) which gives the MUFd numbers [a product of equations (D) and (E)] for the dragline section is solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four separate group:
- First group, holds the swell (S) to 20 percent.
- Second group, holds the swell (S) to 25 percent.
- Third group, holds the swell (S) to 30 percent.
- Fourth group, holds the swell (S) to 35 percent.

**Summary**

The results of the computer work are not given because of their bulk. However, they are discussed and some are shown graphically.
Effect of Swell

The results indicate the percentage variation in required operating reach figures for different swell percentages are substantially uniform at all overburden depths for each machine. For example, at all intervals between 60 ft and 120 ft overburden the 25 percent swell group shows 4 percent more reach requirement for the shovel, the 30 percent swell group shows 7.5 percent more reach required, and the 35 percent group shows 11 percent more reach required than the 20 percent group. The dragline results show 3.5 percent more reach required for the 25 percent group, 6.5 percent more for the 30 percent group, and 9.5 percent more for the 35 percent group than for the 20 percent group.

Figure 13: Graph showing relative “MUF” requirements for shovel type and dragline type simple overcast excavation.
Shovels vs. Draglines

Another significant finding is the uniform relationship between the MUFs curve and the MUFd curve shown in Figure 13. The dragline usefulness numbers (MUFd) are approximately 1.25 times the shovel usefulness numbers (MUFs) at corresponding overburden depths when each series is analyzed on the basis of 20 percent swell. On the basis of weight comparisons alone, the dragline would have a slight advantage over the shovel. However, at this point in the analysis the dragline is eliminated from further consideration due to insufficient advantage indicated over the shovel and to other factors beyond the scope of the study.

Shovel Requirements as Affected by Overburden Depth

A set of curves designated H, r-D curves, Figure 14, are next considered. They consist of three curves, each related to the overburden depth H, which is represented on the abscissa. The three ordinates represent separately the figures obtained from digital results for the indicated reach r, the indicated dipper size D, and the indicated gross machine weight. The latter are obtained by applying the 745 multiplier to the
MUFd digital results. At a glance the three requirements for the shovel can be found for any one overburden depth. For example at H equals 90 ft a 52 yd\(^3\) dipper size is indicated. Also indicated are the operating reach \(r\) of 144 ft and the gross machine weight of 5,540,000 lb. At a unit price per pound for the machine of, say, 65 cents the price of the shovel would be approximately $3,600,000. At 120 ft overburden depth the indicated dipper size \(D\) is 69 yd\(^3\), the operating reach \(r\) is 189 ft and the gross machine weight is 9,700,000 lb. The approximate price at 65 percent pound would be $6,300,000. From these data, together with other pertinent aspects, an optimum overburden depth would be tentatively determined and the conventional type detailed stripping analysis including cost estimates undertaken.

1.6.3. Discussion and Conclusions

The true value of the demonstrated approach to preliminary stripping analysis cannot be determined at this time. Only experience and trial with real problems can give the answer. It must be emphasized that aim here is to demonstrate an approach for attacking such problems. The examples and the assumptions employed are to be considered only as the media through which the approach and procedure are demonstrated. They are deliberately selected slightly on the unrealistic side so as to focus attention on the approach and procedure.

It should be emphasized that this study does not take into account the many variables which are encountered in real stripping problems. The formulae derivations and the illustrative example require hypothetical situations. Actually the entire study is based upon situations and trends which are not necessarily fixed. While the trends seem quite clear there is not proof that they are absolute and they could conceivably change. Therefore, the study should be accepted with the understanding of the existence of this possible limitation.

**Derivation of Formula**

Equation (A) – Derivation

Refer to Figure 11.

Let \(t\) = thickness of deposit (ft)

\(W\) = width of cut (ft)

\(h\) = height of spoil above datum (ft)

\(r\) = reach as shown schematically if Figure 11 measured from circle T (ft)
\[ H = \text{cut height (ft)} \]
\[ A_s = \text{spoil area (ft}^2) \]
\[ A_c = \text{cut area (ft}^2) \]
\[ S = \text{swell (\%)} \]
\[ A_s = (t) (W) + (h - W/25) (W) + (W/25) (W/2) \]
\[ = (t) (W) + hW - W^2/5 \] (1)
\[ A_c = (W) (H) \] (2)
\[ A_s = (1 + S/100) (A_c) \]
\[ = (1 + S/100) (W) (H) \] (3)

Combining (1), (2), and (3)
\[ (1 + S/100) (W) (H) = (t) (W) + (h) (W) - W^2/5 \]
\[ (1 + S/100) (H) = (t) + (h) - W/5 \]
\[ (h) = (1 + S/100) (H) - (t) + (W/5) \]

but \( r = 1.25 \) h. Therefore,
\[ r = [1.25] \times [(1 + S/100) (H) - (t) + (W/5)] \] (A)

Equation (B) – Derivation
Refer to Estimated Shovel Output Table, p. 420
Let \( Q_c = \) the required net tons of cleaned objective (coal) required per month
\( L = \) the yield in net tons of cleaned objective (coal) per acre
\( H = \) cut height in feet
\[ 4840 = \text{yd}^2/\text{acre} \]
\[ (H/3) (4840) = 1613 H = \text{yd}^3/\text{acre} \]
\( (Qc/L) (1613 H) = \text{yd}^3/\text{month} \) required stripping
but 31,200 \( D = \text{yd}^3/\text{month} \) stripping
\[ 31,200 D = (Qc/L) (1613) (H) \]
\[ D = (Qc/L) (1613) (H)/(L) (31,200) \] (B)

Equation (C) – Derivation
\( \text{MUFS} = r \cdot D \) by definition
\[ \text{MUFS} = \{[1.25] \times [(1 + S/100) (H) - (t) + (W/5)]\} \times \{(Qc) (1613) (H)/(L) (31,200)\} \] (C)

Equation (D) – Derivation
Let \( t = \) thickness of deposit (coal) (ft)
\( W = \) width of cut (ft)
h = height of spoil above datum (ft)

r = reach as shown schematically in Figure 12
(measured from circle T) (ft)

H = cut height (ft)

As = spoil area (ft²)

Ac = cut area (ft²)

S = swell, percentage

\[
As = (W) (t) + (W) (h – W/2.5) + (W) (W/2.5)/2
= Wt + Wh – W²/5
\quad (4)
\]

\[
Ac = (W) (H)
\quad (5)
\]

\[
As = (1 + S/100) (Ac)
\quad (6)
\]

Combining (4), (5), and (6)

\[
(1 + S/100) (WH) = Wt + Wh – W²/5
\]

\[
(1 + S/100) (H) = t + h – W/5
\]

\[
h = (1 + S/100) (H) – t + W/5
\]

\[
r = (H – 10)/3 + (1.25) (h)
= \{0.33H – 3.3\} + \{(1.25) (1 + S/100) (H) – t + W/5\}
= \{0.33H – 3.3\} + \{(1.25) (1 + S/100) (H) – 1.25t + 1.25 W/5\}
\quad (D)
\]

Equation (E) – Derivation

Refer to Estimated Dragline Output Table, p. 421.

Let Qc = the required net tons of cleaned objective (coal) required per month

\[
L = \text{the yield in net tons of cleaned objective (coal) per acre}
\]

\[
H = \text{cut height (ft)}
\]

\[
4840 = \text{yd}²/\text{acre}
\]

\[
(H/3) (4840) = 1613 H = \text{yd}³/\text{acre}
\]

\[
(Qc/L) (1613H) = \text{yd}³/\text{month required stripping}
\]

but \[
30,400B = \text{yd}³/\text{month stripping}
\]

\[
30,400B = (Qc/L) (1613) (H)
\]

\[
B = (Qc) (1613) (H)/(L) (30,400)
\]

Equation (F) – Derivation

\[
\text{MUFd} = r \cdot B \text{ by definition}
\]

\[
\text{MUFd} = \{[0.33H – 3.3] + [(1.25) (1 + S/100) (H) – 1.25t + 1.25 W/5]\}
\times \{(Qc) (1613) (H)/(L) (30,400)\}
1.7. Wheel Excavators

“Case Study”

There are two types of wheel excavators of interest to the strip miner. One is the large type developed for the brown coal industry in Germany; and the other is the type developed mostly in the coal strip mines of Fulton Country, usually referred to as the Kolbe wheel excavator.

(i) German Machines

The German-type wheel excavator is usually crawler mounted with the mountings arranged to give three-point suspension. These machines are designed so that there are large crawler areas in contact with the ground so that ground bearing pressures are usually kept to between 16 and 19 psi.

This enables the machines to operate on bench surfaces in the overburden where the supporting soil strength is low without the necessity of laying mats. It is reported that travel gear may account for as much as 30 percent of the total weight of one of these machines.

To give a general idea of the spread in sizes of machines available it maybe stated that one of the smaller machines has a service weight of about 55 tons and a theoretical output of about 260 yd³/hr, and the digging boom has a range from 20 ft above to 2.5 ft below track level; one of the larger machines has a working weight of 5500 tons, a theoretical output of 13,000 yd³/hr, and a digging range from 165 ft above to 65 ft below track level.

Operation

Generally the overburden to be stripped from the brown coals of Germany is a uniform, easily-dug material such as sand or other friable material which can be dug easily and which will flow easily in the conveyor system.

The German strip mine usually involves removing thick overburden from a thick seam, or seams, of brown coal and the wheel excavator may be required to dig the overburden, or the coal.

In one method of operation the wheel excavator progresses along the bank removing material from ground surface to base in a wide strip. Crowding action may
be provided by a boom equipped with an in and out thrust, or it may be accomplished through the traveling action.

The spoil area may be a nearby worked-out pit or it may be on the opposite side of the same pit being worked if conditions allow it.

Transportation of the spoil may be by highly organized rail haulage, or by a system of conveyor belts, or by a combination of the two systems. Conveyors are generally favored for the shorter hauls and rail haulage for the longer hauls.

**Example of Installation**

Figure 15 and 16 show a German wheel excavator installed in a lignite mine at Arjuzanx, France. This installation will supply about 1 million tons of lignite a year to a 120,000 kW steam power station which was constructed to utilize the lignite in this deposit.

Lignite beds are from 5 ft to 20 ft thick and are situated under 50-100 ft of cover. It is estimated that 7,200,000 m³ of overburden will have to be moved for every 1 million tons of coal mined.

The digging unit illustrated can deal with a maximum overburden thickness of 100 ft and can dig about 12 ft below the operating surface on which it sits. Maximum theoretical capacity is 2530 m³/hr. The digging unit weighs 1400 tons and its power requirements are 1237 kW.

Figure 15: Scheme of operation – bucket and wheel excavators at lignite mine
(Arjuzanx, France)
The spoil disposal machine illustrated also weighs 1400 tons and requires 1226 kW. It is equipped with a boom 110 m in length. The total distance from the centerline of the digging wheel to the end of the spoil disposal boom is about 750 ft.

After the lignite is exposed it is excavated and loaded onto conveyor belts by two bucket chain excavators, each of which has a capacity of 400 tons/hr.

(ii) U.S. Wheel Excavators

The type of machine developed in the United States also has a wheel digging device and an internal conveyor system.

All existing U.S. wheels are mounted on stripping-shovel bases with four-corner support. A hydraulic jack at each corner allows the machine to be leveled. Average bearing pressures exerted by the crawlers on the ground surface are about 45 psi since these machines operate from the coal surface.

The digging wheel and the stacker boom cannot be independent in these machines and as the digging wheel swings through the arc of its cut the stacker boom also swing through this same arc.

Digging wheels have been developed to handle the variety of overburden material which is found in Fulton Country, Ill. This material grades from sticky clays containing some floating boulders, to shale of varying degrees of hardness and varying composition. In addition the wheels must also be able to dig several inches of frost which forms during winter operations.
Domestic wheel units are designed to be employed in tandem stripping operations. At present each one is working with stripping shovel. The pit development and cut widths follow the orthodox pattern of prairie type shovel stripping operations. To allow the shovel and wheel excavator to pass at will pit widths are about 110 ft.

**Operation**

The wheel excavator precedes the shovel and takes the top portion of the cut, leaving the lower for the shovel. The shovel digs the lower, which is usually material which has been blasted, and builds a spoil pile immediately adjacent to the coal rib. When taking the top of the succeeding cut the wheel places its spoil well back in heaps which resemble in plan a series of overlapping crescents. It should be noted that although this is a tandem operation each machine handles a portion of the spoil and that no spoil is handled more than once.

At one operation is a wheel strips and spoils 10,000 yd$^3$ per shift, moving the top 25-30 ft and leaving the bottom 30-35 ft for a 33-yd shovel. Cuts taken by both machines are 45 ft wide, but the wheel excavator cut is offset from 10 to 15 ft from the shovel cut to provide a bench that prevents spillage of unconsolidated top material onto the uncovered coal.

Figure 17 shows a typical shovel and wheel excavator stripping operation.
1.8. Miscellaneous Stripping Equipment

“Case Study”

**Bulldozer-shovel Stripping**

A possible combination for stripping up to 35 ft of softer material is the small shovel and the bulldozer. With this type of setup the bulldozer works across the outcrop and takes off 10-12 ft of loose material – sometimes up to 20 ft. The shovel is used to remove the more solid material down to the top of the coal.

A bulldozer plays an important role at an Ohio mine recovering 30-in. coal under 60 ft of shale and sandstone. The unit cuts down 12 ft of shale and makes a level bench for a 5-yd shovel which removes the remaining overburden. After stripping is completed to a 60-ft high-wall, the bulldozer levels spoil for seeding.

The shovel-bulldozer setup is not designed for high output but can be used effectively where cover is relatively soft and a large capital expenditure is not feasible.

**Bulldozer Stripping**

Where stripping is done by bulldozers alone a minimum of two should work together. For efficient materials handling an average of not more than 35 ft of cover should be moved and the terrain should be gently rolling or hilly to permit easier movement of overburden.

Figure 18: At this anthracite operation overburden is removed and coal is recovered in seven planned steps. The completion of Step 7 is at a depth of 750 ft below the original surface.
After the initial cut is made the material should be pushed at right angles away from the outcrop, and the bulldozers should work together, one following the other, and slightly overlapping the path of the leading unit to pick up side spillage. After the pit is filled sufficiently the bulldozers should start pushing to the main spoil area away from the high-wall.

**Stripping Anthracite**

In the anthracite districts of Pennsylvania coal seams which are accessible from the surface usually pitch at fairly steep angles. This means that stripping pits are usually relatively narrow and deep, and that overburden cannot be gotten rid of merely by casting operations. Consequently overburden must be hauled and dumped outside the pit area.

Figure (18) shows the plan being followed for the stripping and mining thick anthracite veins. Overburden is removed in 40-ft benches; excavation is 6 ½ yd$^3$ shovels and haulage by 22-ton trucks. The pit will eventually reach a depth of 750 ft.

**Loading Coal from the Seam**

After the coal seam has been stripped and cleaned the coal is loaded onto truck or onto tractor – trailer units for transportation to the cleaning plant. Shovels dipper capacities of 1-7 yd$^3$ are commonly used for digging the coal from the seam and loading it onto the haulage units.

If the coal seam is hard and unbroken it may be necessary to loosen it by ripping with a bulldozer so that it can be loaded.

Special equipment may be needed to clean the final layer of shale or clay from the top of the coal seam before it is loaded. Scoop loaders, backhoes, or power sweepers may aid in the final cleaning of the coal seam; the first two pieces of equipment have been used in recovering thin layers of coal which may stick to the bottom after the rest of the seam has been loaded.

**Scraper Stripping**

Rubber-tired tractor – scraper units are used as auxiliary equipment when it is necessary to move a portion of the overburden for a distance of several hundred feet. They find application in moving the loose soil and the relatively soft upper portion of the overburden.
Scrapers, push-loaded by big bulldozers, slice off overburden in 25-yd³ bites at an eastern Ohio operation. Six units remove 1800-2000 yd³/hr while working banks 50-60 ft high. Ripper equipped tractors precede the scrapers to break the friable sandstone for the scrapers.

At another operation five tractor-scrappers aided by a rooter have removed 30-35 ft of cover working to a 75-ft high-wall and producing 1700 tons/day of coal. Shovels remove the lower portion of the overburden.

1.9. Haulage Equipment

Sizes of Haulage Units

The size of haulage units used in strip mining ranges from that of a small 2-ton standard dump truck up to the 110-ton tractor trailer units used in some of the large strip mines. A unit in common use in larger operations is the diesel tractor pulling a bottom-dump semi-trailer of 40-60 tons capacity.

The size of a haulage unit should be matched to the capacity of the coal loading shovel. For example a 5 to 7-yd shovel works well with a 40-ton truck and a 3 to 4-yd shovel teams well with a 25-ton hauler. A good rule of thumb is to use trucks with four to five times the dipper capacity of the loading shovel.

In general a 20-ton haulage unit may be equipped with an engine of about 200 h.p., a 40-ton unit with an engine of 300 h.p., a 59-ton unit with 400 h.p., and a 70-ton unit with about 450 h.p. Larger units with capacities in the 90 to 100-ton range may be equipped with engines of up to 700 h.p.

When engine sizes exceed 600-700 h.p. mechanical problems in transmitting this power to the drive wheels increase rapidly with engine size. For applications where haulage grades are steep, as in some open pit metal mining operations, haulage units have been built with an individual electric motor drive on each wheel. This arrangement eliminates the mechanical problems with transmission and differential.

Economics of Haulage Units

Figure (20) shows in graphical form how the cost per ton-mile decreased from a peak of 5.5 cents per ton-mile in 1953 to 4.7 cents per ton-mile in 1960 as the size of coal haulers and the quantity of coal hauled increased.
Another coal company has in service tractor – trailer haulage units of 40, 50, and 90 tons capacity. The 90-ton units are powered by 700 h.p. engines which operate at speeds of 50-55 mph empty and 45-47 mph loaded on level main road sections. The 90-ton units average 1.36 mpg (miles per gallon) of fuel as compared with 1.77 mpg for the 50-ton units and 1.78 mpg for the 40-ton units.

Overall costs per ton-mile, including road maintenance, tractor repair and maintenance, trailer repair and maintenance, lubrication, fuel and tire costs, but excluding depreciation were 47 percent less for the 90-ton units as compared with the 40 and 50-ton units.

![Figure 20: Sizes of haulage units ton-miles hauled vs. cost per ton-mile](image)

**Roads**

It is apparent that the roads constructed for trucks hauling from 60 to 100 tons of coal at speeds from 30 to 40 miles per hour must be very well constructed and maintained in order to avoid disintegration of the road surface and excessive wear on haulage units.
Portions of such roads are relocated at frequent intervals as the pit face advances. The following features are common to well constructed haulage roads.

1. Sub-base is well prepared by removing top soil on original ground and compacting all filled areas.
2. Roads are made extra wide to prevent single-plane rutting.
3. Side ditches are made extra deep to remove all surface water and underground seepage.
4. All adverse grades are broken at minimum distances of 1000 ft with level stretches 500-1000 ft in length being provided.
5. All curves are laid out as long radius curves.
6. Base material is made from 1 to 3 ft thick, depending upon weight of haulage units.
7. Large size base material is used, usually broken slate, limestone, or sandstone, as available.
8. The material for the road surface should be graded material containing fines to permit packing and to prevent water absorption.
9. The surface cross-section should be arched to permit immediate surface water runoff.

1.10. Coal Augers

Augering has grown to the point where more than 8 million tons of coal a year is produced by this means. Auger-mining of coal from the surface can be divided into two types of operation.

1. Augering of high-walls prepared, or left, by stripping operations. These augering operations do not usually require any extensive pit cleanup and serviceable haulage roads are already in existence. In addition the pit is usually wide enough to accommodate any of the standard augering units available.
2. When teamed with stripping equipment augers permit higher banks to be stripped because the combined cost of auger coal and strip coal can be made to equal or better the cost when strip mining alone is done under thinner cover.
3. Augering of outcrop coal usually requires the preparation of working benches and roads. These preparations for augering will probably add 10 to 15 cents a ton to the overall mining cost, depending upon the seam thickness. Width of
benches required will vary from 20 to 100 ft depending on the augering equipment and the haulage methods used.

**Characteristics of Coal Seam**

To be susceptible to profitable augering a coal seam should lie relatively flat or should have a constant pitch and be free from rolls and faults. The seam should not contain wide or excessively hard horizontal bands of sandstone or other abrasive material which will cause excessive wear and breakage of bits, pilots, and heads.

**Preparing for Augering**

When a coal seam is to be augered in conjunction with a stripping operation care should be taken in blasting so that the high-wall will be left in the best possible condition, and loose material should be cleaned down so that crews will not be endangered by slides or falling rock.

The pit should be left clean for a width sufficient to accommodate the augering equipment. Augering operations should be carried out as soon as possible after stripping is completed before the high-wall begins to weather and to slough. If augering is to be carried out on an old high-wall then a bulldozer and/or a shovel may be needed to clean up the site for augering.

**Augering Operations**

The facing up operation and bench preparation should be done in such a manner that the augering machine will have solid footing and the bench area will have natural drainage.

The site for each auger should be selected so that as much as possible of the seam height can be removed from one hole. However, if the seam thickness is such that one auger cannot recover most of the coal, a second hole, above the first, can be drilled.

Pillars to 4-12 in. should be left between holes. The pillar thickness should never be less than 4 in. and a rule of thumb is to leave 1 ½ in. of pillar for every 10 in. of auger diameter. It may be necessary to leave a 3 or 4 ft wide pillar every fifty holes to prevent a general subsidence of the roof and resultant squeezing action on the augers.
Crew Requirements and Productivity

Crew requirements vary, depending on the type of equipment. McCarthy auger units which are self-moving are usually operated with a two-man crew for the single auger units and three-man crew for the dual auger units. Compton auger units require a three-man crew for the single auger units and a four-man crew for the multiple auger units. These units require a bulldozer to move the auger from hole to hole. Some Compton units have been equipped with skids. Cardox auger units require two or three men depending on the model.

Figure (22) is a chart from which the amount of coal which an auger will produce in a shift may be estimated.

In addition to the auger crew a bulldozer operator or shovel operator may be required for preparing the face and the site for augering operations. In addition trucks and drivers will be required to transport the mined coal to the cleaning plant.

Figure 22: Auger production vs. auger diameter

Examples of Auger Installations

An Ohio strip mine operator supplements his operations with high-wall augering. A 30-in auger unit operated by a four-man crew adds 250-300 tons/day to the mine output. The augers penetrate to a depth of 200-220 ft.
A crew of four men operates a 30-in. self-moving auger in an Alabama pit to produce 140-150 tons of coal per shift. The auger penetrates to a depth of 125 ft. One of the four-man crew drives loaded trucks to the preparation plant; one truck serves as a surge bin while the driver makes a round trip to the preparation plant with the other.

A self-moving 47-in. unit works in a mountain-top pit only 26 ft wide. Three augers, two full times and one spare, are used at this property to recover coal that could not be stripped because of the hazard of spoil rolling or sliding downhill and damaging mine installations.

In Ohio a 30-in. self-moving auger produces 400-500 tons/day in a three-shift operation. Working in areas previously deep mined, the unit has produced up to 275 tons in a single shift.
2.1. Introduction

The first remotely controlled mining machine was developed by the Union Carbide Corp. in 1946 in connection with experiments with underground coal gasification. Two machines were built to make 36-in. diameter air passages in the seam, the second machine being remotely controlled. Although this company’s experiments with underground gasification were discontinued the remotely controlled tunneling machine worked so well that it was decided to build a larger model.

The major problems to be solved were in the design of a device to indicate to the operator the position of the machine with respect to the top and bottom of the seam and devices to keep the machine level crosswise, to avoid spiraling, and to keep the holes straight, parallel and at the proper spacing.

![Figure 2.1: Patterns for auger holes on turns](image)

Blocking on outside turns allows for maximum extraction

Sets of parallel holes are used on inside turns

Figure 2.1: Patterns for auger holes on turns

The new machine went into service in October, 1949. It bored a roughly oval hole, 38 in. high and 116 in. wide. A second pass, below the first, recovered the coal in seams of greater height than 38 in. The cutting heads consisted of four overlapping
cutting heads driven at 60 rpm through a speed reducer by two 60 h.p. motors in parallel. The cusps between the holes were removed by fixed blades at the top and the bottom to produce an even floor and roof.

The machine was propelled on crawler tracks powered by a 7 ½ h.p. variable speed DC motor. Horizontal steering was accomplished by moving a pair of guide shoes which bore against the sides of the hole.

A train of portable belt conveyors, each 30 ft long, and mounted on two rubber-tired wheels, was used to transport the coal from the miner to the surface. As the machine advanced it pulled the train of conveyors after it.

Several devices for obtaining the information needed by the operator in order to steer the machine were provided. A “stratascope”, or device for measuring the pressure on a cutter bit was mounted on the two outer cutting heads. The pressure changed as the bit cut through the various layers in the seam. An electrical signal from each stratascope was shown on an oscilloscope screen, in the form of an irregular circular trace. The operator soon learned from the shape and location of these irregularities whether the seam was rising or dipping, and steered the machine to closely follow the top.

A pair of pendulums sent out electrical signals to show the inclination of the machine both fore and after, and crosswise.

An electric drill at the side of the machine was used to measure rib thickness. Each time the machine was stopped to add a conveyor this drill was actuated. It drilled through the rib in about 40 sec and indicated to the operator the distance at which it broke through into the adjacent hole.

Maximum depth of hole bored with this miner was 690 ft, limited by the number of portable conveyors available. It was normally operated at a speed of 2 ft/min, corresponding to a production of about 2 ½ tons/min.

The normal operating crew was three men, consisting of a machine operator, portable crane operator, and helper.

This experimental machine, while not a commercial success, pointed the way to the development of a practical system for mining by remote control.

A new machine was built and went into operation in 1953. This machine bores the same size hole but is heavier and more powerful than the previous model. The cutter heads are driven by a single 200-h.p. motor at a speed of 60 rpm. A rotary drum type cutter is used to remove the roof cusps.
A major improvement was made in the method of conveying coal from the machine to the surface. Instead of stopping to add sections of belt conveyors as needed the conveyors are assembled into a continuous train, 800 ft long, which follows the machine without stopping. The train extends along the high-wall when not in the bore hole. Another improvement was made in the conveyors themselves. Instead of 24-in. wide belt conveyors these are 60-in. wide chain and flight conveyors. “Minor” roof falls up to 6 ft wide, 8-10 in. thick, and 100 ft long are caught by these wide conveyors and carried to the surface without incident. This eliminates the need for men to go underground to clean up such falls.

Instrumentation on this machine was refined and made more rugged and reliable than that on the older model. Automatic controllers maintain the machine on a predetermined course at the desired angle to the horizontal, correct automatically for “spiral” and steer the machine to bore an absolutely straight hole.

The machine operator sits at a desk watching the oscilloscopes which tell him where the machine is with respect to the top and bottom of the seam. In addition another operator is required to observe the action of the conveyor train. This two-man crew can mine and load into trucks an average of more than 400 tons of coal per shift, working in a 44-48 in. seam.

An improved model of this machine, known as the Pushbutton Miner, and built by the Joy Manufacturing Co., was placed in trial operation in 1961.

The boring unit and conveyors are essentially as in the previous model but the penetration depth has been increased to 1000 ft and as the conveyors retract they are stored in a circular ramp structure or Heli-Track. This is a mobile crawler-mounted structure 45 ft high, 77 ft long, and 48 ft wide which weighs more than 600 tons which, in addition to storing the conveyors and borer, houses the control room from which it is operated, and a repair shop.

The boring unit removes the coal for a width for 9 ft 9 in. Pillars between adjacent holes are between 2 and 4 ft wide, depending upon strata conditions.

2.2. Land Reclamation

Planning Reclamation (Case Study)

The two principal problems involved in the reclamation of an area which has been strip mined are:
(1) Restoring the surface to usefulness after mining is completed.
(2) Preventing pollution of streams by acid runoff waters.

The type of restoration work depends upon such factors as contour of the land, type of overburden, and the State laws regulating restoration.

In some cases the spoil will not support plant life until it has been decomposed by weathering for several years. In such case the only immediate steps which can be taken to restore the land are to backfill and level it. In any event the material in spoil banks should be analyzed to see whether it will support plant life before any planting is done.

Sourness of soil acidity is probably the biggest problem in land reclamation. One method of combating an acid soil condition is to work only the top layer of the spoil in preparing a seedbed.

Where grass or legumes are to be planted, the area should be prepared by harrowing, rather than by plowing. This technique avoids bringing the acid soil to the surface. The hard, compact, and dry soil structure left after leveling with a bulldozer is a further handicap.

For most spoil areas tree planting is recommended. Where the spoil is too steep for planting tree seedlings, it may be seeded, preferably with Black Locust and Sericea lespedeza.

Among the seedlings, Locust, Shortleaf Pine, Scotch Pine, and White Pine have shown the most promise, with Black Locust making the most rapid growth.

Autumn olive appears to be a most promising shrub, coupling fast growth with heavy production of fruit for wildlife.

Contour furrowing for tree planting has been practiced in some areas of West Virginia. Such furrowing results in the collection and retention of moisture which stimulates faster growth; but if the soil is too loose it can result in excessive build-up around the seedlings. As a consequence, they may wind up too deep in the ground.

Crown vetch has proved to be especially valuable on slopes for erosion control and is also an excellent forage crop for cattle.

**Strip Mining Laws**

A new strip mining law was enacted by the Pennsylvania State legislature and became effective on January 1, 1962. The law contained the following provisions pertaining to the bituminous coal fields:
On “productive land” and in built-up areas, strip pits must be backfilled at a slant of 45° from the bottom of the excavation to the high-wall of the cut. “Productive land” is defined as that which has produced farm crops within the five previous years.

When pit extend to highways, public buildings, churches, or community buildings, backfills must be restored to the original contour within 100 ft. For any remaining distance up to 750 ft, there must be a 45° backfill.

In non-productive areas – such as woodlands and wilderness – operators must backfill at an angle of 70° to the top of the highest cliff and the bottom of the pit must be covered with 5 ft of earth.

2.3. Drainage

A strip mine is a natural man-made reservoir and a constant struggle is required to keep water out of the pit. Every effort should be made to keep water from entering the pit by ditching above the high-wall and by diverting natural streams to new channels, if necessary.

When water does enter the pit it should be removed as much as possible by gravity flow. It is desirable, when possible then culverts or drain pipes may be installed at intervals.

Since the high-wall advances constantly, portable pumps are most commonly used when pumping is required. Sumps are excavated at intervals along the pit for installation of these pumps.

Pumps may be mounted on skids or on wheels and in the larger sumps pumps may be float mounted.

Figure 27: Truck mounted drill
Pumps may be driven by electric motors or by diesels. Air cooled diesels are becoming increasingly popular because of their portability and low maintenance requirements. Plastic pipe and aluminium pipe is coming increasingly into use as discharge lines since they are light and easily portable.

2.4. Drilling Blast Holes

The two types of drills in most common use in strip mines are: (1) the rotary, and (2) the auger.

**Rotary Drills**

Rotary drills are of the dry type, using compressed air to remove the cuttings from the holes, and using tri-cone bits for cutting. A large variety of rotary drills is on the market, including sizes of bore holes from 5 up to 15 in. in diameter. Some are mounted on crawlers while others are truck mounted.

In operations where the overburden is thick operators generally favor the heavy crawler mounted machines which can drill holes up to 12 in. in diameter.

These large diameter holes enable operators to break more ground with fewer holes and also make it possible to use low cost explosives such as ammonium nitrate.

**Augers**

Augers are also available in a variety of sizes and capacities. These machines are usually truck mounted. One heavy duty auger can bore an 8-in. hole up to 150 ft deep or a 12-in. hole to a depth of 60-80 ft. a variety of bits are available for various types of material to be drilled, and bits are available which will cut limestone and coarse grained sandstone.

**Drill Performance**

To provide sufficient pit width for a 65-yd shovel one company uses a rotary drill to drill hole on three levels in hilly country. Roadways are on 27-ft centers, with holes spaced 27 ft apart on each level. Abrasive action of sandstone wears out a bit after 7000-9000 ft of drilling. In an average shift, two men sink 535 ft of 9-in. hole.

At another operation an average of 110 ft of 9-inch hole is drilled in 8 hr while drilling with a rotary drill in 50 ft of medium hard shale covered by 12 to 40 ft of sandstone. Bit life averages 21,000 ft of hole.

At another stripping operation 625 ft of 10 ½ in. hole is drilled per shift with a rotary drill in hard sandy shale that ranges from 45 to 70 ft thick. Bit life is 13,700 ft.
Two men handle the drilling and shooting assignments at a Pennsylvania mine recovering two seams. They employ a truck mounted rotary drill to sink 6 ½ in. holes at the corners of 15 × 15 ft squares in 48 ft of clay and sandstone. In an average shift they drill and shoot thirty holes.

At one Ohio mine two crews use auger type machines to sink an average of 600 ft of hole each per shift in sandstone overburden.

At another Ohio mine two drill 400 ft of 8-in. hole per shift and also help charge the holes with AN-oil mixture at the end of the shift.

**Vertical vs. Horizontal Drilling**

Although most blast holes are drilled vertically there are special situations in which it may be more economical to use blast holes drilled horizontally into the high-wall. Some factors which favor horizontal drilling are:

1. Overburden is of such nature the drill roads are difficult to build, as in rough terrain or there is soft ground in which drills may become mired. The horizontal drill operates from the floor of the pit and eliminates the necessity for building drill roads at the top of the high-wall.

2. The cover is thin and the rock is hard. In this case long horizontal holes will give a better breaking action than short vertical holes.

3. There is a layer of tough rock close to the coal seam. Horizontal holes allow the explosive to be concentrated in or adjacent to this hard rock for better breaking effect.

Both augers and rotary drills are available mounted for horizontal drilling. Heavy augers are available which will bore 12-in. diameter holes horizontally to depths of up to 150 ft. these augers may be truck mounted or may be mounted on a self propelled rubber-tired chassis.

At an Indiana mine a rotary drill equipped with a single mast bores horizontal holes 9 in. in diameter and 48 ft deep without adding drill sections. Penetration rate is 50 in./min. in the best drill shift 816 ft of holes were drilled by one man.

This drill operates one full shift and one part shift 6 days a week and drills enough holes to prepare overburden for operation of a 40 yd³ shovel 24 hr a day and seven days a week.
**Inclined Drilling**

Some of the newer drills are so designed that their masts can be tilted to drill holes at angels up to $30^\circ$ from the vertical. Advantages claimed for inclined drilling are the following:

1. Toe and back breakage can be eliminated.
2. Fragmentation is better because of better use of explosive energy as well as reduced resistance at the bottom of the hole.
3. Less footage of hole and less explosives are required per ton of rock broken.
4. Smaller diameter holes can be used.
5. The overburden will be thrown a greater distance.

The disadvantage of drilling inclined holes is that they are more difficult to load as the material being charged may tend to hang up in an inclined hole whereas it would drop to the bottom of the hole in a vertical hole.

**Percussion Drills**

![Figure 28: Typical bits for rotary drilling](image)

Percussion and rotary-percussion drills are not extensively used in bituminous coal stripping operations because the overburden to be drilled usually consists of the
softer shales and sandstones. Drills of this type are used extensively in quarrying operations in the igneous and metamorphic rocks and to some extent in the metamorphic rocks encountered in anthracite stripping operations.

Down-the-hole (DTHD) drills have largely supplanted the hammer and drill steel type for large diameter and deep blast holes. With the down-the-hole drill the drill unit with the rock bit attached goes down the hole so that all of the striking energy of the pneumatic hammer is transmitted to the rock bit through a short rod instead of the pneumatic portion of the drill remaining on the surface and the hammer blows being transmitted to the bit through a long drill steel.

**Drill Bits**

Rotary drills usually employ tricone bits of the type which was originally developed for the oil industry. A tricone bit consists of three cones free to rotate about their axes which are face studded with cutting teeth. For soft formations these teeth are rather long and are spaced relatively far apart. For extremely hard rock the cones are studded with rounded tungsten carbide knobs which serve as the breaking or crushing teeth.

To obtain optimum operating results with rotary bits it is necessary to apply heavy pressure to them. For 9-in. bits or larger in hard formations it is customary to apply approximately 6000-7000 lb./in. of bit diameter, and to rotate these bits at approximately 40-50 rpm.

**Bit Life**

Experimental drilling is desirable to determine the optimum combination of bit size, speed of rotation, and bit weight (amount of push on the bit).

A quarry in medium-hard limestone employed a drill which had the ability to apply approximately 45,000 lb. weight to the bit and was equipped with an air compressor with enough capacity to clean cuttings from a 7 ¾ in hole. This machine had been drooling with 6 ¼ in. bits and bit life had been approximately 700 ft per bit with an average penetration rate of about 10 in./min. By changing to 6 ¾ in. bits and increasing the bit weight from 30,000 to 38,000 lb., the penetration rate increased to 12 in./min. Previously the 6 ¼ in. bits had always failed due to worn bearings, but the bearings in the 6 ¾ in. bits were larger and even at the increased bit weight provided a bit life of 1600 ft. Further experimentation revealed that it was possible to increase the bit life to 3000 ft per bit with an average penetration rate of 8 in./min by lowering the
revolutions per minute and reducing the bit weight. Figure 29 shows graphically the results of tests on 7 \( \frac{3}{8} \) in. bits.

Figure 30: Typical bits for auger

In some instances where soft but abrasive materials are to be drilled it may be more economical to use less expensive bits which can be discarded when they become dull.
At one stripping operation where blast holes were drilled in very abrasive sandstone it was found that tricone bits would wear out after only 90-110 ft of drilling. When a “finger type” auger bit head with replaceable bits was substituted for the tricone bits, bit costs were reduced by two-thirds, to less than 4 cents per foot of hole drilled.

**Auger Drill Heads**

Bit heads for augers generally are of the types which have two or more projecting “fingers” which may be faced with tungsten carbide and which act as cutting tools to groove and rip the rock at the bottom of the hole. Figure 30 shows some of the drill heads available for augers.

**Factors Affecting Costs**

**Bit Costs:**
The general features of larger bits are more rugged and the size of the bearings increases with the increase in bit diameter. This permits application of higher bit weight per inch of bit diameter and results in higher rates of penetration and longer bit life. Although the larger bits are more expensive, the longer bit life often accounts for a lower bit cost per foot.

**Labor:**
The drilling crew remains the same regardless of the size of hole drilled, therefore the labor cost per foot of hole drilled varies inversely with the penetration rate.

In the overall picture it is the cost per ton of rock broken which determines the economy of the drilling process. Thus the number of holes required to break a given tonnage is also a factor to be considered. Since larger holes can hold more explosive, fewer holes are required to break a given tonnage of rock.

Thus the relative costs of drilling various sizes of holes vary with various types of drills and bits. Thus the overall cost of overburden preparation depends upon a number of interrelated factors which can only be evaluated after testing a variety of drill types, sizes, and hole spacing.

2.5. Blasting

**Explosives**

Ammonium nitrate (with carbonaceous material added) is the most commonly used blasting agent in strip mining operations. Although fertilizer grade ammonium
Nitrate had long been known to be explosive under certain conditions, a mixture of ammonium nitrate and carbon black, which was packed in polyethylene bags, is used for blasting.

A mixture of granular ammonium nitrate and fuel oil is presently in extensive use in strip mining operations. These ingredients are usually mixed in the proportion of 100 lb. of ammonium nitrate to 5 or 6 lb. of fuel oil. The resulting “AN-FO” mixture may be bagged for transportation to the drill holes or it may be transported in bulk and poured or blown into the holes by means of compressed air.

Ammonium nitrate is a relatively slow velocity explosive and explosion must be initiated with high explosive primers. Ammonium nitrate also detonates better when it is loaded in large diameter drill holes and difficulties are sometimes experienced in getting complete detonation in drill holes which are of small diameter and relatively long.

Many of the explosive manufacturers are producing variations of the “AN” mixtures. These may contain technical grade ammonium nitrate, mixed with a carbonaceous dust and a sensitizer, such as nitromethane.

In addition the conventional high explosives, such as dynamite, may be used in special situations when extremely hard or tough strata must be broken.

**Mixes**

Most companies making their own AN-FO mixes use either a “prilled” or grained ammonium nitrate with No. 2 fuel oil. Approximately 4 quarts of oil are added to each 100lb. of ammonium nitrate. (Note: “prills” are pallets.)

Many of the larger companies have their own mixing plants where the ingredients can be metered as they flow to a mechanical mixer. Some companies use the product immediately after it is packaged while others prefer to let the mixture season before using it. The seasoning period varies from several hours to several days.

Other producers prefer to mix the nitrate and oil at the hole site. Oil is poured over the opened bags at each hole then left to percolate down through the ammonium nitrate for a short period before holes are charged.

The most recent development is a slurry type blasting agent. New slurry consists of a mixture of ammonium nitrate, sodium nitrate, high-explosive sensitizer, and water. It has a density of about 1.5 and a rate of detonation of about 17,000 ft/sec.
Figure 31: (a) Popular design for a permanent-type plant for blending nitrate and fuel oil provides for thorough mixing
(b) Typical design of portable mixing equipment includes nitrate hopper, oil supply, mixer, and air-placement unit.

**Primers**

Prilled ammonium mixtures react differently with different priming systems. Dynamite used as a primer causes the AN-FO to release a larger volume of gas which is desirable for heave and throw of the bank while a detonating cord causes a sharp cracking impact which is desirable when a hard rock must be shattered.
One operator uses about 1 lb. of primer for each 150 lb. of AN-FO mixture in dry holes and about 1 lb. of primer for each 50 lb. of AN-FO in wet holes.

Figure 32: Characteristics of ammonium-nitrate explosives
(a) The detonation velocity is increased as the size of prill is decreased.
(b) Water has a marked effect on velocity; failure occurring at 8 percent moisture.
(c) Velocity falls faster with fuel-lean than with fuel-rich mixtures.
(d) Above 4 percent diatomaceous earth, the detonation velocity falls rapidly.
Figure 33: Comparison of the costs of priming with dynamite and with detonating fuse
Loading

At smaller operations the AN-FO mixtures are usually bagged and the bags are charged into the blast holes. At larger operations, where consumption of nitrate is several hundred tons or more annually, nitrate maybe transported in bulk in enclosed hopper trucks and mixed with fuel oil as it is discharged into the holes.

Horizontal holes may be loaded with bagged nitrate using mechanical tampers to push the bags into the hole. A more efficient system for charging horizontal holes is that which employs a truck-mounted unit to blow the bulk mixture into the holes through a plastic tube.

Bagged drill cuttings are commonly used for stemming material for horizontal holes. Split plugs and wedges have also been used to stem horizontal holes because they are easier to place than the bagged cuttings.

Drill cuttings are commonly used for stemming in vertical holes.

Where blast holes pass through strata of varying degrees of hardness an effort is usually made to obtain the greatest concentration of explosive in the toughest strata. A denser mixture may be formulated for placement in such rock, with increased density being obtained by proper gradation of the grain size of the mixture of ammonium nitrate granules and prills.

Blast Hole Patterns and Spacing

The blast-hole pattern used and the spacing of holes depend upon a number of factors including thickness of overburden, type of rock in overburden, and size of the blast holes.

Spacing between the rows of holes which parallel the high-wall may be anywhere from 15 to 25 ft and spacing between individual holes in a row may be from above 20 to 30 ft.

The accompanying Figures show typical drill-hole patterns and spacing in use at several operating mines.

A formula based on the size of hole, strength of rock, and type of explosive has been developed. From this formula the maximum allowable hole spacing may be computed. Table 10 has been derived from the formula. Hole spacing maybe computed if the diameter of the hole and the type and condition of the rock is known.
Figure 34: Blast-hole patterns and spacing

(a) Deck loading concentrates part of the charge in the upper portion of thick overburden for better fragmentation.

(b) Deck loading is also advantageous in overburden consisting of rocks with varying hardnesses. Millisecond delays increase the effectiveness of explosive.

(c) Buffer shooting possible a uniform drilling pattern, eliminates large chunks which are sometimes produced when shooting against an open face.
The values in the table are based upon the use of an oxygen-balanced, prilled ammonium-nitrate fuel oil mixture.

**Table 10: Typical borehole spacing**

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Tensile strength (psi)</th>
<th>Spacing (ft/in.) of borehole diam</th>
</tr>
</thead>
<tbody>
<tr>
<td>Strong granite</td>
<td>1298</td>
<td>1.92</td>
</tr>
<tr>
<td>Strong anhydrite</td>
<td>1220</td>
<td>1.97</td>
</tr>
<tr>
<td>Strong limestone</td>
<td>890</td>
<td>2.30</td>
</tr>
<tr>
<td>Average granite</td>
<td>888</td>
<td>2.32</td>
</tr>
<tr>
<td>Marble</td>
<td>860</td>
<td>2.37</td>
</tr>
<tr>
<td>Weak anhydrite</td>
<td>800</td>
<td>2.45</td>
</tr>
<tr>
<td>Graywacke</td>
<td>700</td>
<td>2.62</td>
</tr>
<tr>
<td>Strong sandstone</td>
<td>583</td>
<td>2.85</td>
</tr>
<tr>
<td>Average limestone</td>
<td>480</td>
<td>3.15</td>
</tr>
<tr>
<td>Marlstone</td>
<td>480</td>
<td>3.15</td>
</tr>
<tr>
<td>Weak granite</td>
<td>422</td>
<td>3.37</td>
</tr>
<tr>
<td>Average sandstone</td>
<td>412</td>
<td>3.40</td>
</tr>
<tr>
<td>Salt (potash-halite)</td>
<td>400</td>
<td>3.46</td>
</tr>
<tr>
<td>Greenstone</td>
<td>380</td>
<td>3.55</td>
</tr>
<tr>
<td>Weak limestone</td>
<td>280</td>
<td>4.12</td>
</tr>
<tr>
<td>Weak sandstone</td>
<td>280</td>
<td>4.12</td>
</tr>
</tbody>
</table>

(a)
Figure 35: Blast-hole patterns and spacing
Casting Overburden with Explosives

In explosives casting large amounts of low-cost ammonium nitrate mixtures are loaded into medium sized drill holes in a usual ratio of more than 1 lb. of powder per cubic yard of overburden. The explosive charges are detonated through millisecond delay electric blasting caps. When the shot is fired, a large part of the overburden is blasted into the pit away from the high-wall and up on the spoil pile where it attains a favorable angle of repose.

One stripping operation with a 50-ft sandstone overburden prepares its bank for shooting by drilling five rows of holes on 12 ft burdens with 15 ft spacings. Four rows are 6 ¼ in. diameter holes while on the outside row 7 ⅜ in. diameter holes are used to insure the desired movement of the toe. Hole depths average 45 ft. The powder factor averages 1.2 lb./yd³.

This blasting operation moves a minimum of 35 percent of the overburden without the use of equipment and uncovers 35-40 percent more coal in the same time. Even though drilling and blasting costs are higher, mining profits are greater because of the increased production.

Factors Favoring the Use of Explosives Casting
(1) Deep, hard overburden requiring extensive shooting.
(2) Dumping radius of primary stripping unit less than 150 ft.
(3) Narrow, steep cuts, 60-100 ft wide.
(4) Under-capacity of primary stripping unit.
(5) Overcapacity of coal mining unit.
(6) Ability to use least expensive AN-FO explosive.

**Factor Unfavorable for Explosive Casting**

(1) Overburden is shallow and easily excavated.
(2) Cuts are more than 100 ft wide.
(3) Condition are poor (water, etc.) for the use of bulk AN-FO.
   The haulage road may have to be run past the stripping pit.

2.6. Ripping Overburden

Experience in several limestone quarries indicates that ripping costs may run from 5.2 cents per yd$^3$ to 11.5 cents per yd$^3$ depending upon the hardness of the rock. In excavation sandstone the ripping costs have ranged from 2.1 cents per yd$^3$ in soft sandstone to 15.0 cents in hard rock. Ripping costs on these jobs were less than the costs for drilling and blasting methods.

Generally shale and sandstone which overlie bituminous coal beds are amenable to ripping and this method warrants consideration when mining methods are chosen.
UNDERGROUND MINING SYSTEMS

3.1. Introduction

Practically all coal produced by underground mining is mined by one of the following systems:

1. Pillar-mining systems. These include the room-and-pillar system and the block system as well as the bord-and-pillar system.
2. Long-wall systems. These include the long-wall advancing and the long-wall retreating systems.

3.1.1. Pillar-mining Systems

Entries, cross entries, panel entries, and cross cuts, or rooms are driven through the coal bed to divide it into pillars or blocks which may then be extracted on retreat. Figure (1) shows the general layout for a room-and-pillar system. At present, such hand loading methods have now been almost entirely replaced by mechanical loaders and 90 percent of the coal produced by underground mines is loaded mechanically.

3.1.2. Long-wall Systems

![Figure (1) Mining method at Mine 3 (room-and-pillar system)](image)
“Main roads” or “mother gates” are driven through the coal bed and are semi-permanent in nature and “gate roads” or “gate” leading to the long-wall face are maintained.

A long face or “long-wall” is usually several hundred feet long and is served by two or more “gate roads”; the long dimension of the working face being at right angles to the direction of the gate roads. Some long-wall faces are several thousand feet long while others are operated as a series of stepped faces, each several hundred feet long.

The long-wall is advanced continuously by extracting slices of coal from the face and transporting the broken coal to the gates from which it is transported to the main roads or mother gates and thence to the main shaft. “Cross gates” are maintained as angular cross-connections between gates in order to shorten haulage and ventilation distances.

Figure (2) shows a layout for a typical long wall mining system as employed in European mining practice.
3.2. Pillar-Mining Systems

These include the “room-and-pillar” system, the “block” system, and (in Britain) the “bord-and-pillar” system. With the room-and-pillar system entries, cross-entries, and panel entries are driven to “block out” large panels of coal and rooms are turned off (usually at right angles) from the entries. Rooms are driven as wide, or wider, than the entries. Pillars left between rooms may, or may not, be extracted. Different mine employ different sizes of pillar. Room pillars may commonly be from 20 to 40 ft wide and from 40 to 90 ft long. When room pillars are not to be recovered they are made as narrow as is feasible while still leaving enough coal to support the roof. Figure (3) shows an idealized room-and-pillar system.

Figure 3: Development by room and pillar system from the outcrop
With the block system a series of entries, panel entries, rooms, and cross cuts is driven to divide the coal into a series of blocks of approximately equal size which are then extracted on retreat. Development openings are most commonly driven between 15 and 20 ft wide. Pillars are most commonly from 40 to 60 ft wide and from 60 to 100 ft long.

Generally, the room-and-pillar system is favored for the thinner coal beds (that is, those less than 3-4 ft thick) while the block system is used more often in the thicker coal beds where equipment can move about freely without the necessity of “brushing” roadways to obtain headroom.

The bord-and-pillar system, such as used in some British mines, divides the coal into very large pillars or blocks by means of very narrow openings. “Bords” approximately 12-15 ft wide are driven at approximately right angles to the main coal cleat, and narrow openings (“wall”) about 6 ft wide are driven parallel to the main cleat. With the bord-and-pillar system pillars as large as 130-200 ft square are blocked out and extracted on retreat. This system may be considered a compromise between the block system and the retreating long-wall.

3.2.1. Mine Development

In coal mining there are two broad classes of underground openings: (1) mine development and (2) production.

Development work comprises driving entries, entry cross cuts, room necks, or other openings necessary to open up access to production places from which coal will be produced later.

Development openings formerly were driven narrower than production places and only a few men could work as a group in a development heading. Because the working places were so narrow the rate of coal production was low and the cost of development coal was high.

As mines became more mechanized and yielded larger quantities of coal at a faster rate per shift it became necessary to increase the quantity of ventilating air. In order to provide adequate airways it was necessary to increase the number of development headings in a group and/or to make the development headings wider. It is common practice now to drive development headings ten or twelve in a group and, where roof conditions will allow it, the headings are made wider. As the width of the
development approaches that of the production place the productivity form development approaches that from production places.

Production work in the room-and-pillar system involves the driving of rooms and the subsequent extraction of room pillars. With the block system the production work consists of the extraction of the blocks.

3.2.2. Ratio of Development to Production

For successful operation of a mine a certain ratio between development and production must be maintained. If development does not exceed production, mining in a production section could be completed before the next production section is ready for operation. In mines where the cost of coal mined in development is more than that from production sections such an imbalance between development and production would increase the overall cost of the mined coal.

3.2.3. Conventional (Cyclic) Operation

“Conventional mining” is a cyclic process which includes the following operations.

1. Cutting coal. A slot, usually horizontal, is cut for the length of the coal face. Usually this slot is cut at the bottom of the seam although coal is sometimes cut at an intermediate height or even at the top of the seam, and transverse (shear) cuts are sometimes used. This slot in the coal provides a free face to which the coal can break when it is shot.

2. Drilling. Drills for boring shot holes may be hand held, post mounted, or mounted on a rubber-tired drill jumbo. Coal drills are usually of the auger type rather than the percussion type as used in hard rock mining operations and they may be powered by electricity, compressed air, or may be operated from the hydraulic system of a mining machine.

3. Charging and shooting. Holes may be charged with an explosive or with a mechanical device which releases compressed air or carbon dioxide to break the coal.
Table 1. Bituminous coal and lignite mechanically loaded underground in the United States, by type of loading equipment (Case Study)

<table>
<thead>
<tr>
<th>Type of loading equipment</th>
<th>1960</th>
<th>Percentage of total</th>
<th>1961</th>
<th>Percentage of total</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Net tons</td>
<td></td>
<td>Net tons</td>
<td></td>
</tr>
<tr>
<td>Mobile machines:</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Direct into mine cars</td>
<td>8,137,606</td>
<td>3.3</td>
<td>5,931,074</td>
<td>2.5</td>
</tr>
<tr>
<td>Onto conveyors</td>
<td>11,195,270</td>
<td>4.6</td>
<td>6,755,764</td>
<td>2.9</td>
</tr>
<tr>
<td>Into shuttle cars</td>
<td>142,775,484</td>
<td>58.1</td>
<td>132,446,554</td>
<td>56.3</td>
</tr>
<tr>
<td>Continuous-mining machines:</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Onto conveyors</td>
<td>10,474,509</td>
<td>4.3</td>
<td>11,031,679</td>
<td>4.7</td>
</tr>
<tr>
<td>Into shuttle cars</td>
<td>67,453,771</td>
<td>27.4</td>
<td>73,289,572</td>
<td>31.1</td>
</tr>
<tr>
<td>Scrapers and conveyors equipped with duckbills or other self-loading heads</td>
<td>1,232,019</td>
<td>0.5</td>
<td>1,032,009</td>
<td>0.4</td>
</tr>
<tr>
<td>Hand-loaded conveyors</td>
<td>4,517,166</td>
<td>1.8</td>
<td>4,863,270</td>
<td>2.1</td>
</tr>
<tr>
<td>Total mechanically loaded</td>
<td>245,785,775</td>
<td>100.0</td>
<td>235,349,922</td>
<td>100.0</td>
</tr>
</tbody>
</table>

4. Loading coal. Formerly all broken coal was loaded by hand into rail-mounted cars. The efficiency of loading was improved when conveyors were introduced. However, most coal produced in the United States is now handled by mechanical loaders which load the broken coal onto conveyors or into shuttle cars which carry the coal from the working faces to the main transportation system for haulage to the surface.

5. Installing roof support. The roof may be supported by bolts, or by wooden props, or by steel props, or posts and beams.

The cycle of operations described above, that is cutting, drilling, shooting, loading and installing support, is termed “cyclic” or “conventional” mining as opposed to the newer “continuous” or “non-cyclic” mining in which coal is ripped or cut from the seam mechanically and the drilling and shooting operations are eliminated.

When conventional mining methods are used the various work phases cannot be performed simultaneously in the same working place and it is necessary to operate from three to seven working places so that at least one place is available for coal loading at all times. Even if the schedule of the phases in a cyclic is maintained, coal production ceases during the time required to move the loading equipment from one loading place to another.
3.2.4. Continuous (Non-cyclic) Operation

The continuous mining machine cuts or rips the coal from the face and loads it onto conveyors or into shuttle cars in a continuous operation. Thus the drilling and shooting cycles are eliminated, along with the necessity for working several headings in order to have available a heading in which loading can be in progress at all times.

3.2.5. Coal Haulage

Haulage refers to the process of moving the coal mined at the face to the mine tipple. It does not include the transportation of men and supplies that use the same equipment. For convenience the transportation system can be divided into four parts: Portal haulage (shaft, slope, and drift); mainline haulage; intermediate haulage; and face haulage. These subdivisions are not necessarily made in all transportation systems as the intermediate haulage may be the mainline and the portal haulage may be a continuation of the mainline.

The portal haulage used depends on the type of mine opening. In vertical shafts, hoisting of loaded mine cars on cages to the surface is being eliminated in favor of skips that are loaded automatically from a bottom dump. Slopes sunk on dips not exceeding 17° are equipped with belts in most newly developed mines and some older mines. Where slope belts are used a supplementary track system transports men and supplies. In drift mines this haulage is a continuation of the mainline haulage whether it is done by locomotives, belts, or even shuttle cars.

Mainline haulage is the movement of mined coal from gathering stations in the mine to the portal and usually is accomplished by locomotives or by belts fed by other conveyors. Locomotives up to 50 tons operated singly or in tandem pull trips of as many as twenty-five cars ranging to 20 ton capacity. Track rails weighing up to 120 lb/yard are either bolted or welded and are laid on wood ties in a ballasted locomotive operators by two-way radio. Block signals and automatic switch-throws are operated from the cab of the locomotive without stopping. Speeds of 15 miles, or more, per hr are maintained by the mainline trips. Belts used in mainline transportation show a trend away from the rigid frame construction to the rope type.

Intermediate haulage is the transportation of mined coal from the face haulage to that point where it is accessible to the mainline. It is accomplished by conveyors,
belt, or locomotives and mine cars. Where gathering belts or conveyors are used, the end of this phase of haulage is at the transfer point of the coal to the mine cars or to a mainline belt. With locomotives and mine cars, it would end at the partings where the loads are assembled into trips for the mainline locomotives.

3.3. Long-Wall Mining Systems

In the long-wall system coal is won from a continuously advancing face which may be from a few hundred feet to several thousand feet in length. The long-wall system has been little used in American coal mines because coal seams are generally mined at fairly shallow depths and favorable roof conditions allow the use to the more flexible pillar mining systems. However, several recent installations of fully mechanized long-wall systems have given very high production rates.

3.3.1. Hand-worked Long-wall

Originally all long-walls were hand-worked with the coal being holed (cut) and drilled by hand labor, shot, and then hand loaded into cars running on tracks located just behind the face for transport to the loader gate.

In firm coal it is necessary that a slot be cut back underneath the seam in order that the coal will settle and fracture. Such a slot may be cut by a miner with a hand pick. If the seam contains dirt bands the cutting may be done in these bands.

In exceptionally soft or friable coal seams, such as many of those in Germany, the coal may be won by breaking it from the face by means of hand-held pneumatic picks. The coal, after it is broken down, is then loaded onto conveyors which are located parallel to and just behind the long-wall face.

3.3.2. Mechanized Cyclic Mining

Long-wall mining operations in which coal is first cut and broken down from the face and then loaded onto conveyors are cyclic in nature. Usually only one shift in each 24 hours can be devoted to loading of coal. The remainder of the time is spent in cutting and breading the coal and in moving and installing roof support, ripping roadways, and miscellaneous work.
The first step in the mechanization of long-wall operations was the introduction of the mechanical coal cutter for undercutting the face.

Next to be introduced was the conveyor installed parallel to the face, onto which coal was shoveled and which conveyed the coal to loading gates whence it was transported to the main haulage system.
4.1. Mining Cycles

The sequence of operations in cyclic mining is as follows:

(i) Cutting
A slot, usually horizontal and from 5 to 7 ft deep, is cut across the width of the coal face. This is usually cut at floor level and provides the coal a free face to which it can break when it is shot, as well as providing a plane of separation between the coal and the floor.

(ii) Drilling
Rotary auger-type drills are commonly used for drilling shot holes. Drills may be hand held, or may be mounted on columns, or they may be mounted on rubber-tired drill jumbos.

(iii) Shooting:
Holes may be shot with explosives, or the coal may be broken by means of compressed air or carbon dioxide cartridges.
(iv) **Loading coal:**

Most coal is mechanically loaded by means of crawler mounted loaders which operated by thrusting a wide steel pan into the broken coal which is then swept up the pan and onto a conveyor which transports the coal to the back end of the loader and discharges it onto another conveyor, or into shuttle cars. From the loader the coal is transported to the main transportation system which may be either main conveyor belt or may consist of large rail mounted car.

(v) **Root support:**

Temporary support may be placed as soon as the coal is shot, and before the loader approaches the face, or support may be placed over the loader as it advances.

![Cross-sections of typical cutter bars](image)

**Figure 2: Cross-sections of typical cutter bars**

Support may consist of props, or of props and bars, or of hydraulic props, or of roof bolts. Wooden roof props are still in extensive use but have been replaced to a large extent by roof bolts.

4.1.2. Coal Cutters

The basic cutting unit in all modern coal cutters is the “jib” around the perimeter of which a toothed chain travels somewhat in the manner of a chain saw. All coal cutters employ this same cutting principle.
4.1.2.1. Short-wall Coal Cutter

This machine is used in cutting wide headings and production places. The jib is fixed rigidly to the body of the machine and extends through the machine body to discharge the cuttings at the back end of the machine.

The machine maneuvers by means of two ropes one of which is used to pull the machine along while the other is used to control the angle which the machine makes with the face.

Usually short-wall cutters are equipped with jibs about 7 ft long and the rate at which the machine can progress along the face while cutting is about 2 ft/min.

Figure 2.3 illustrates the method of “sumping in” and cutting across a short-wall face.

A short-wall cutter may be used to cut a number of headings during the course of a shift. Rail mounted trucks may be used (in track mines) to transport the machine from heading to heading while a crawler-mounted or rubber-tire mounted flitting truck may be used for this purpose in trackless mines.

Figure 3: Cutting the face with a short-wall cutter
4.1.2.2. Arc-wall Cutters

As its name implies the arc-wall cutter makes a cut by sumping in and then swinging its jib in an arc. It is most commonly used for cutting narrow places, such as those in room and pillar work.

The arc-wall cutter is designed to make a horizontal cut on one horizon. Where it is desirable to make cuts at various heights machines are available with the body mounted on four hydraulic jacks.

Arc-wall cutters may be rail mounted, mounted on crawlers, or mounted on rubber tires.

![Arc-wall cutter diagram](image)

Figure 4: Arc-wall cutter

4.1.2.3. Universal Cutters

The machine can cut horizontally, vertically, or at any angle. The angle of the cut is controlled by rotating the head which carries the jib while the height of the cut is regulated in some machines by raising or lowering the body of the machine by...
means of integral hydraulic jacks, and in other machines is regulated by raising or lowering a head which pivots about a transverse horizontal axis.

Universal cutters may be track mounted, mounted on crawlers, or mounted on pneumatic tires.

![Figure 5: Universal cutter](image)

4.1.3 Coal Drills

In soft coal, or in thin seams, shot holes may be drilled with hand held auger-type rotary drills. Where the coal is hard or the seam is thick column-mounted drills may be used.

For highly mechanized operations multiple rotary drills mounted on booms which are mounted on a rubber tired chassis and are manipulated by hydraulic controls are used.

**Drilling and Shotfiring**

Shot holes should be spaced in accordance with the thickness of the seam. In thin seams one row of holes placed near the top of the seam and spaced a distance apart about equal to the thickness of the seam is usually sufficient.

In seams 5 ft or more in thickness it may be necessary to use a top and a bottom row of shot holes with the holes staggered in alternate rows.
4.1.4. Coal Loaders

Loading machines may be classified as (1) gathering-arm loaders; (2) duckbill or shaker loaders; (3) overshot or rocker shovels; (4) slusher or scraper-loaders.

Of these only the gathering arm loaders and the duckbills have been used extensively in American coal mining practice.

Rocker shovels are used extensively in metal mining work but require about 7-8 ft of headroom and find application in very few coal mines.

4.1.4.1. Gathering-arm loaders

These are self propelled and may be mounted on rails, on crawlers, or no rubber tires. The crawler-mounted loader is the most commonly used. It is made up of three main components (a) the gathering head which is thrust into the coal and which is fitted with two rotating arms to sweep the coal onto the conveyor: (b) a chain conveyor which runs down the middle of the machine and carries the coal to the rear; and (c) the crawler chassis which carries a separate motor for each of the crawlers.

In track-mounted machines the loading head has the ability to swing 45° to either side of the center line and the tail of the machine can swing 15-20° on either side of the center line.

Gathering-arm loaders are available in a height from about 30 in. up and in capacities ranging from less than 1 ton/min up to 12-20 tons/min.

4.1.4.2. Duckbill Loaders

Shaker conveyors can be converted to self-loaders by the addition of “duckbills”. The duckbill loader consists of steel-trough sections with a shovel head at the forward end and with gripping device at the rear end which can be clamped to the shaking conveyor so that the reciprocating motion of the shaker is imported to the duckbill.

The shovel head or “duckbill” rests on the floor and is forced forward into the loose coal and, by the nature of the shaker action coal is caused to travel up the trough and onto the shaker conveyor.
4.1.4.3. The Conveyor-loader

The Jeffrey conveyor-loader is mounted on rubber tires and is very maneuverable because the front wheels can be rotated through $90^\circ$. The machine has a gathering head which is provided with chains and flights which scrape the coal onto a chain conveyor which carries the coal back to discharge it at the rear of the machine.

The method of operation is as follows: Coal is cut and shot and the conveyor-loader is flitted to the face and maneuvered so that the gathering head is at the broken coal while the rear end is situated so that it will discharge onto the room conveyor. A hydraulic roof jack at the rear of the machine is dripped and tightened between floor and roof to provide an anchorage point about which the machine may pivot. The retractable rear wheels are then raised clear of the floor and the front wheels are rotated through $90^\circ$ so that the front end of the loader can travel in an arc across the
heading. The gathering head is then lowered and extended into the coal pile to start loading.

The machine has an overall height of 24 in. which makes it suitable for use in seams as low as 36 in. and has a rated capacity of 1½ tons/min with an average loading rate of 2/3 tons/min including clean-up.

4.2. Methods and Equipment Used in Development Work

A number of mines were studied by personnel of the U.S. Bureau of Mines to determine what methods and equipment were used in underground. Following are the descriptions of the methods and equipment used in development work at two of these mines and in addition there are shown diagrams of development methods.

Case Study

4.2.1. Mobile Loading onto Chain Conveyors (Mine 1, Figure 8)

This mine is operated in the No. 3 Elkhorn bed, which averages 35 in. thick in this area. The bed is flat lying, with some local dips. The coal is underlain by fire clay and overlain by sandy shale that usually provides good roof. The overburden average 315 ft thick and consists mainly of strata of shale and sandstones.

Figure 8: Geologic section and method of development at Mine 1
(Mobile loading onto chain conveyor)
The mine was operated two shifts a day and produced an average of 1070 tons of raw coal (1049 tons of clean coal), using the following units:

1. Two mobile-loading piggyback to chain-conveyor units developing panel entries; nine men each.
2. One mobile-loading piggyback to chain-conveyor unit mining rooms; nine men.

A detailed study was made of one entry development unit only. Entries 30ft wide were driven in sets of four on 52- and 68-ft centers, with cross cuts 20 ft wide, on 68-ft centers. Entries were developed by a unit consisting of nine men, one mobile-loading machine, two piggyback conveyors, two short-wall cutting machines, two chain conveyors (in entries), one mother chain conveyor, one elevating conveyor, two hand-held electric drills, one roof-bolting machine, one supply truck, and one car hoist.

**Operating Cycle**

The cycle of operation was: timber, drill and cut, extend conveyor, blast and load coal. Power to operate the mining equipment was furnished at 250 V d.c.

Entries were advanced in pairs, with a 38-ft coal pillar between each pair. The first pair (entries 1 and 2) were advanced 340 ft. Equipment was then moved out and set up in entries 3 and 4. While these entries were being advanced, the bottom was brushed, the main-line track was extended in number 1 and 2 entries, and a new loading station was established 320 ft ahead of the old station. When the entries 3 and 4 were advanced 340 ft, the equipment was moved back through the last open cross cut to entries 1 and 2.

**Roof Support**

The roof was supported by three rows of props on each side of the chain conveyor, spaced 4 and 5 ft apart. When the roof required it, ¾ in. expansion-shell type bolts 2 ft long were installed on 5-ft centers. The roof bolts were provided with 6 \( \times 6 \times \frac{3}{16} \) in. steel bearing plates.

**Cutting and Drilling**

Five equally spaced holes 9 ft deep were drilled at a distance of 1 ft below the roof. The coal was bottom-cut and the conveyor was extended. Each hole was charged with four sticks of permissible explosives and blasted individually.
**Loading and Haulage**

Broken coal was loaded by a mobile-loading machine onto a piggyback conveyor which transferred it to a chain conveyor which discharged coal into 2 ½ ton capacity steel, bottom-dump mine cars. Loaded cars were transported to the surface by a 15-ton trolley locomotive.

Rock dusting was done by hand on shift and by machine on week-ends, when required.

**Crew Required**

A unit crew consisted of the following men:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Section foreman</td>
<td>1</td>
</tr>
<tr>
<td>Loading-machine operator</td>
<td>1</td>
</tr>
<tr>
<td>Loading-machine operator’s helper</td>
<td>1</td>
</tr>
<tr>
<td>Cutting-machine operator</td>
<td>1</td>
</tr>
<tr>
<td>Cutting-machine operator’s helper</td>
<td>1</td>
</tr>
<tr>
<td>Conveyor man</td>
<td>1</td>
</tr>
<tr>
<td>Timberman</td>
<td>1</td>
</tr>
<tr>
<td>Utility or supply man</td>
<td>1</td>
</tr>
<tr>
<td>Boom man</td>
<td>1</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>9</td>
</tr>
</tbody>
</table>

A unit crew produced an average of 150 tons of raw coal per shift, or 16.7 tons per man-shift. The average advance in each entry in a development groups is 10 ft per shift, or a total of 20 ft for the group.

**“Case Study”**

4.2.2. Mobile Loading into Shuttle Cars (Mine 8, Figure 9)

**Equipment**

Main entries 20 ft wide were driven in sets of six on 50 ft centers, with cross cuts 20 ft wide on 75 ft centers. These entries were developed by a unit consisting of twelve men, two mobile loading machines, two short-wall cutting machines with caterpillar trucks, three shuttle cars, two hand-held coal drills, a 30 in belt conveyor. Power to operate equipment is furnished at 250 V d.c.
**Operating Cycle**

The cycle of operation was: Timber, drill, cut, blast, and load coal.

**Drilling and Blasting**

Six holes, each 7 ft long, were drilled in the coal. Three holes were drilled 1 ft below the roof, and three others were drilled 2 ft below the roof, with the lower holes being staggered between those in the upper group.

![Geologic section and method of development at Mine 8 – Mobile loading into shuttle cars](image)

**Figure 9:** Geologic section and method of development at Mine 8 – Mobile loading into shuttle cars

After drilling the entries were bottom cut and the drill holes were charged with four sticks of permissible explosive in each hole. Holes were then blasted individually.
Roof Support

Props were used to support the roof where it was sandstone. However where it was shale more timbering was required and 3 in.\(\times\)8 in.\(\times\)16 ft long wood crossbars were set on wood posts roof support.

Loading and Haulage

The coal was loaded by mobile-loaders into shuttle cars and transported to the 30-in. belt conveyor in the No.3 entry for transportation to the surface.

Rock dusting was done on shift by hand casting and offshift by machines.

Crew Required

A unit crew consisted of the following men:

<table>
<thead>
<tr>
<th>Role</th>
<th>Number</th>
</tr>
</thead>
<tbody>
<tr>
<td>Section foreman</td>
<td>1</td>
</tr>
<tr>
<td>Loading-machine operators</td>
<td>2</td>
</tr>
<tr>
<td>Loading-machine operator’s helpers</td>
<td>2</td>
</tr>
<tr>
<td>Cutting-machine operators</td>
<td>2</td>
</tr>
<tr>
<td>Cutting-machine operator’s helpers</td>
<td>2</td>
</tr>
<tr>
<td>Shuttle-car operators</td>
<td>3</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>12</strong></td>
</tr>
</tbody>
</table>

A unit crew produced an average of 509 tons of raw coal per shift or an average of 42.5 tons/man-shift. The average advance in each entry in a development group was 18.2 ft/shift, a total of 108.6 ft for the group.
CHAPTER 5
PILLAR MINING SYSTEMS
CONTINUOUS (NON-CYCLIC) MINING

5.1. Continuous Mining Machines

Continuous mining machines cut or rip the coal from the seam and load it onto conveyors, or onto shuttle cars, for transportation to the main haulage system. In continuous mining the cutting, drilling, and shooting cycles required in conventional mining are thus eliminated.

Continuous mining machines may be divided into two broad classes; that is the “rippers” and the “borers”.

(i) Ripping Machines

These are characterized by a series of parallel disks or chains which are equipped with coal cutting picks. These disks or chains rotate about and axis which is parallel to the face and the picks rip the coal from the face.

(ii) Boring Machine

These are equipped with cylindrical cutting heads with cutting picks set on the edges of the cylinders. These are advanced into the coal and cut a series of concentric annular grooves.

Most types of ripper machines are articulated, that is the head of the machine can swing to the left or right, within a limited angle, and these machines are therefore somewhat more flexible in operation than are the boring machines.

“Case Study”

5.2. Joy Continuous Miner

There are various models of Joy continuous miners to suit different seam conditions. One of the models (6CM) is shown in Figure 1. The machine consists of a cutter head or ripper head, a main chassis, and a loading unit. The ripper head is fitted with five multi-pick chains and is driven by 100 hp electric motors. A single disk
containing cutter picks is fitted on either side of the ripper bar. The 6CM is fitted with a 7-ft ripper bar and can cut up to a height of 10 ft. The ripper jib can be swung on either side and is attached to the main chassis, which is mounted on two tracks powered by two separated motors.

![Figure 1: Ripper-type continuous](image)

Cut coal is delivered to the conveyor by the ripper head and also by two gathering arms, which are fitted on either side of the conveyor. The rear conveyor can be swung 45° to either side of the machine.

**Method of Operation**

The machine, with its head in the retracted position, is moved forward on its crawlers until the cutting bits are just touching the face in the center of the room. The head is then swung to its limit to the right and lowered until the bits are about to touch the floor. It is then, with the cutting chains running, advanced 24 in. into the coal and is gradually raised vertically, taking out the coal above as it rises, until the desired roof line in the coal is reached (maximum 10 ft). The head is then retracted, taking out any loose coal left below roof level. Thus a block of coal, 42 in. wide and 24 in. deep, is removed from the seam.

The head is again lowered, swung about 3 ft to the left and again sumped forward 24 in. in the seam, and this operation is repeated across the whole width of the room.
5.3. Lee-Norse Miner

The Lee-Norse miner rips coal from the seam by means of a cutter head which consists of four rotating disks which are equipped with standard cutter picks. The arms holding the cutter head are caused to oscillate by means of motor-driven eccentrics.

The Lee-Norse miner is mounted on crawlers and cuts a width of 8 ½. The cutter-head does not swing sideways therefore it cuts on a straight line until it has advanced to the end of a cut. It then moves back on its crawlers and maneuvers into position for the next cut.

Method of Operation

The machine advances on its crawlers with the cutting head operating and sumps into the coal at the top of the seam. The coal is ripped downwards and is collected by gathering arms and delivered into the single-chain conveyor and discharged at the rear of the machine.

Characteristics

The Lee-Norse miner is a relatively simple low-cost machine which has a high productive capacity and the larger models of machines have produced an average in excess of 800 tons of raw material per shift.

Where mining conditions are severe, bit costs may be as much as 10 cents per tons but costs on the order of 3-5 cents/ton are more common.

Maintenance labor runs generally from 10-15 cents/ton and parts from 10 to 20 cents/ton.

Figure 2: Lee-Norse CM68 continuous miner
5.4. Goodman Continuous Miner

The cutting head of this machine consists of two large cutting arms rotating in opposite directions, and a trimming chain. All these are fitted with cutting picks. Each large arm is equipped with a double core barrel, two short arms, and two long hinged arms which lock in the extended positions. Wedge-shaped cutter bars at the top and bottom of the chain break off that section of the face which is not accessible either to rotating arms or the cutting chain.

The actual section cut by the machine may be varied up to 7 ½ ft high and 13½ ft wide. The machine moves on crawler tracks. A high speed chain conveyor collects the coal in front and carries it through the machine. The discharge height can be altered as desired.

The tail conveyor can be swung 40° in either direction.

![Figure 3: Goodman Type 429 borer continuous miner](image)

Method of Operation

The machine moves up to the face on its crawlers. The rotating units, geared together and equipped with bit-filled cutting arms, and a center core barrel, cut and break out the coal.

As the machine bores ahead, the trimming chain cuts free, at the top and bottom, the center wedges of coal not reached by the rotating arms. The chain also widens the path at the bottom on each side, and can be arranged for cutting a wider than normal top to allow for cross barring.

Coal taken from the face drops to the bottom, is pushed to the center and up the throat of the conveyor, by plows on the rotating arms. The discharge conveyor can be swung 40° to either side, a big advantage when turning cross cuts or pulling pillars.
5.5. Joy Twin Borer Continuous Miner

The Joy twin borer uses the boring principle to cut a full face at a maximum rate of 8 tons a minute. The machine is crawler mounted and has two sets of boring arms and two sets of trim chains, one of which cuts an arched shape for tool control. The miner’s full face cut varies from 5 ft 11 in. to 7 ft 11 in. high and 11 ft 8 in. to 12 ft 8 in. wide. It trams while boring at up to 4.5 ft/min; trams from mine openings at up to 28.5 ft/min. all cutting surfaces retract hydraulically for roof and wall clearance.

Boring-arm diameters can be changed in 4-in. increments from 6 to 7 ft by using arms of different lengths, and raising or lowering the main transmission 2 in. for each change. Additional heights up to 12 in. are cut with the upper trim chain, which is quickly raised hydraulically to any desired position.

Boring arms are equipped with internally mounted hydraulic cylinders which can retract each arm 12 in. overall. The center section of each arm and the cylinders remain the same, regardless of diameter bored. The boring arm assembly is driven by an 8-in. diameter, alloy steel output shaft from the main boring arm transmission.

![Figure 4: Joy twin borer continuous miner](image)

5.6. Auger Mining

The Cardox-Hardsocg auger is equipped with a cutting head 3 ft long and 24 in. in diameter. A number of picks are fitted to the periphery of the cutting head.
These cut an annular ring of coal which is broken up by a center bit and is then conveyed by archimedian screw sections to the mouth of the hole.

This machine is equipped with a 25 hp. electric motor and can bore holes up to 80 ft deep at a rate of 2.7 ft/min.

An auger built for the Oliver Springs, Tennessee, mine of the Wind Rock and Coal Company by the Salem Tool Company bores 34-in. holes to depths of 100 ft. The machine has separate power and drilling units. The power unit weighs 8 ½ tons and is equipped with two 50 hp. motors, hydraulic pumps and a 250 gal. tank of hydraulic oil. Hydraulic power is transmitted to the drilling unit through 50 ft hoses. The power unit is self-moving on two hydraulic skids.

The drilling unit weighs 9 ½ tons. It is self-moving on two skids and when it is positioned for drilling it is secured in place by two roof jacks at the rear of the unit. Rotational speed of the auger may be varied from 0 to 37 rpm but the best results are obtained when the head turns at the maximum speed. Auger sections are 5 ft long.

Two men are needed to operate the auger while a third drives a shuttle car between the auger surge car and the loading ramp. As the cutting head advances auger sections are pulled from the previously drilled hole and added in the new hole.

When operating in clean coal the auger takes an average of 1 hr to drill a 100-ft hole. It takes an average of 20 sec to pull back the carriage and be ready to add a new auger section. The new auger section is added in 15 sec. each 100-ft hole yields about 23 tons of coal. Six inches of coal is left between adjacent holes. This gives a theoretical recovery of 50 percent of the coal.

Figure 5: Coal auger operating underground
The auger cannot be used for a full production face but is useful in the following cases:

1. Extraction of good quality coal from thin seams.
2. The partial extraction of pillars in mines which are to be abandoned.
3. The partial extraction of seams where subsidence cannot be tolerated.

Ventilation difficulties have retarded the use of augers in British mines.

5.7. Methods and Equipment Used in Development Work

A number of mines were studied by personnel of the U.S. Bureau of Mines to determine what methods and equipment were used in underground development (U.S. Bureau Mine I.C. 7813, 1957), and how continuous mining machines were employed (U.S. Bureau Mines I.C. 7696, 1954).

Following are the descriptions of methods and equipment used at some typical mines as well as diagrams showing the development patterns used.

**Case Study**

5.7.1. Boring-type Mining Machine (Mine 7; Figures 7 and 8)

This mine is operated in the Sewell coal bed, which averages 44 in. thick in this area. The bed is flat lying but some rolls and faults are encountered in mining.
The overburden averages 436 ft in thickness and consists mainly of shale and sandstones.

One of the continuous mining on entry development was studied in detail. Main entries 27 ft wide were driven in sets of nine on 60-ft centers, with cross cuts 18 ft wide on 75-ft centers (Figure 39). These entries were developed by a unit consisting of nine men, one boring-type continuous mining machine, one mobile loading machine, one piggyback conveyor, two chain conveyors, and a 30-in. belt conveyor. Power to operate the equipment was supplied at 440 V a.c.

Figure 7: Geologic section and method of development – boring-type continuous miner (Mobile loading onto chain conveyors at Mine 7)

**Operating Procedures**

Entries were advanced with a continuous mining machine by making three successive cuts, each 9 ft wide and 42 ft long (Figure 16). Cross cuts were advanced in a similar manner with two 9-ft cuts.
Figure 8: Sequence of cuts at Mine 7

**Loading and Haulage**

Coal mined with the continuous mining machine was deposited on the floor. The coal was reloaded by the mobile-loading machine and transported by the piggyback conveyor, two chain conveyors, the 30-in. belt conveyor, and the 36-in. main-haulage belt conveyor, to the preparation plant on the surface.

**Roof Support**

The roof was supported with wood props set on 5-ft centers.

**Crew Required**

A unit crew consisted of the following man:

<table>
<thead>
<tr>
<th>Position</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Section foreman</td>
<td>1</td>
</tr>
<tr>
<td>Continuous mining machine operator</td>
<td>1</td>
</tr>
<tr>
<td>Continuous mining machine operator’s helper</td>
<td>1</td>
</tr>
<tr>
<td>Loading machine operator</td>
<td>1</td>
</tr>
<tr>
<td>Loading machine operator’s helper</td>
<td>1</td>
</tr>
<tr>
<td>Supply man</td>
<td>1</td>
</tr>
<tr>
<td>Mechanic</td>
<td>1</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>7</strong></td>
</tr>
</tbody>
</table>
A unit crew produced an average of 299 tons of raw coal per shift, or 42.8 tons per man-shift. The average advance in each entry of a development group was 6.1 ft/shift, or a total of 54.5 ft for the group.

5.7.2. Boring-type Mining Machine (Mine 16; Figure 9)

This mine is operated in the Lower Kittanning Coal Bed which is from 48 to 59 in. thick in this area (42-53 in. mined). The overburden averages about 100 ft thick.

Figure 9: Main entry development with boring-type continuous miner - shuttle car haulage to conveyor (Mine 16)

Operating Procedures

Panel entries were developed with a boring-type continuous mining machine as shown in previous chapter. Entries were driven 18 ft wide by making alternate cuts 9 ½ ft wide on 75-ft centers.
Loading and Haulage

Two shuttle cars transported the coal from the continuous mining machine to an elevating conveyor, which discharged onto a 26-in. belt. The coal was conveyed by the 26-in. belt to a 30-in. main belt, which transported the coal to the tipple.

Roof Support

All entries were roof bolted on 5-ft centers with 5/8 in. bolts, 36-in. long set against 6×6×3/8 in. steel plates. The average advance of the mining machine in an 18-ft entry was 40 ft per shift.

Equipment Required

A unit crew was equipped with one boring-type continuous mining machine, two shuttle cars, one elevating conveyor, one 26-in. belt, one portable roof-bolting machine.

Crew Required

The crew consisted of eight men. The average production of raw per unit crew per shift on development was 101 tons giving an average production per man on a unit crew of 12.6 tons.

5.7.3. Boring-type Mining Machine (Mine 18; Figures 10 and 11)

This mine is operated in the Sewickley Bed in West Virginia which has a thickness averaging 72 in. in this area. The average thickness of overburden is 320 ft. The mine uses a block mining system with pillars being extracted.

Operating Procedures

A boring-type mining machine was used for all development work. The plan for development is shown in Figure 18. Entries and cross cuts were driven 19 ft wide by making alternate cuts 9 ½ ft wide, as shown by the sequence of cuts in Figure 19. The sequence of advance of each heading is indicated by letters in Figure 18.
Figure 10: Butt-entry-panel development with boring-type continuous miner - mobile loading into shuttle cars, Mine 18

**Loading and Haulage**

The continuous mining deposited the coal on the floor which provided adequate surge capacity, and allowed the continuous mining machine to operate with a minimum of shuttle car haulage delays. The mobile-loading machine reloaded the coal into 6-ton capacity shuttle cars which transported it to the unloading station where it was discharged into 5-ton steel mine cars.

**Roof Support**

Steel timber jacks, set on 4-ft centers, with 3 by 5 by 23-in. wood cap pieces, were used for safety posts to protect men and equipment. These timber jacks were
reset near the opposite rib for each alternate cut (see Figure 43. Permanent timbering generally was not required for roof support.

Figure 11: Timbering plan and sequence of cuts at Mine 18

**Equipment Required**

A unit crew was equipped with one continuous mining machine, one mobile-loading machine, and three cable-reel shuttle cars.

**Crew Required**

A continuous mining crew consisted of eight men as follows:

<table>
<thead>
<tr>
<th>Position</th>
<th>Number</th>
</tr>
</thead>
<tbody>
<tr>
<td>Section foreman</td>
<td>1</td>
</tr>
<tr>
<td>Continuous mining machine</td>
<td>1</td>
</tr>
<tr>
<td>Loading machine</td>
<td>1</td>
</tr>
<tr>
<td>Shuttle-car operators</td>
<td>2</td>
</tr>
<tr>
<td>Timber men</td>
<td>3</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>8</strong></td>
</tr>
</tbody>
</table>

The average production per unit shift was 565 tons. This gave an average production per man-shift of 70.6 tons for each man on a development unit crew.
5.8. Full-dimension Mining

A full-dimension system is a haulage system which provides an uninterrupted flow of coal from a loader or continuous miner at the face to the main line transportation system. The equipment required for this system consists of a series of interconnected conveyors which are mobile and articulated and which will retract or extend a sufficient distance for the development of a five-entry system.

Such a system with its components is shown in Figure 12.

![Figure 12: Method of developing a five-entry panel](image)

5.9. Methods and Equipment used in Room Mining

Following are the descriptions of room mining methods and equipment used in conjunction with continuous mining machines at some typical mines.

5.9.1. Ripper-type Mining Machine (Mine 20; Figure 13)

This mine is operated West Virginia which has an average thickness of 62 in. in this area (58 in. mined). The average thickness of overburden is 600 ft. Mining was
by room-and-pillar system with pillars not extracted. Total recovery of coal was about 60 percent.

Figure 13: Room-mining with a ripper-type continuous miner – shuttle car haulage to mine cars, Mine 20

Operating Procedure

Rooms were turned on 25-ft centers at 60° from the butt entries and driven 16 ft wide and 275 ft deep. Cross cuts between rooms were 16 ft wide and were turned on 80-ft centers at an angle of 45°.

Loading and Haulage

One of the two shuttle cars was used behind the continuous mining machine as a surge car, while the other transported coal from the surge car and discharged it into mine cars of 1.9-ton capacity at unloading stations. The unloading stations consisted of wooden ramps in cross cuts which had been top-brushed and roof bolted.

Roof Support

A 50-ft pillar of coal was left between each two adjacent groups of rooms. In mining, 4 in of top coal were left to support the roof. Four-inch steel H-beams, 13 ft long and spaced on 3-ft centers, were set on wood posts to support the roof. These H-beams and posts were recovered when a room was finished.
Equipment Required

Each unit crew was equipped with one continuous-mining machine and two cable-reel shuttle cars.

Crew Required

A continuous-mining crew consisted of 7 1/6 men as follows:

<table>
<thead>
<tr>
<th>Position</th>
<th>Number</th>
</tr>
</thead>
<tbody>
<tr>
<td>Section foreman</td>
<td>2/3</td>
</tr>
<tr>
<td>Continuous mining machine operator</td>
<td>1</td>
</tr>
<tr>
<td>Shuttle-car operators</td>
<td>2</td>
</tr>
<tr>
<td>Timber men</td>
<td>2</td>
</tr>
<tr>
<td>Electrician-mechanics</td>
<td>1 1/2</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>7 1/6</td>
</tr>
</tbody>
</table>

The average daily production of raw coal per unit per shift for the continuous mining units was 141.7 tons. This gave an average production of 19.8 tons of raw coal per man-shift for each man in a unit.

5.9.2. Ripper-type Mining Machine (Mine 10; Figure 14)

This mine is operated in the Lower Kittanning Bed in Pennsylvania. This bed is 48 in. thick in the locality of this mine and is overlain by about 500 ft of overburden. A room-and-pillar system was used with pillars extracted to give an overall recovery of about 90 percent.

Operation Procedure

Rooms 16 ft wide were turned on 56-ft centers. As shown in Figure 23, the two in by rooms were mined simultaneously by making cuts 1-6 inclusive. The pillars between these rooms were mined by extraction as shown by cuts 7-15 and Room C was mined by making cuts 16, 17 and 18.

Loading and Haulage

One of the two shuttle cars was used behind the continuous mining machine as a surge car while the other hauled the coal to the unloading point.
Roof Support

A single row of wood props on 4-ft centers is set in rooms 4 ft from the rib, and in pillar extraction a single row of wood props is set along the rib.

Equipment Required

Each unit crew was equipped with one continuous mining machine, two cable-reel shuttle cars, and a 30 in. belt.

Crew Required

A continuous-mining crew consisted of six men as follows:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Section foreman</td>
<td>½</td>
</tr>
<tr>
<td>Continuous mining machine operator</td>
<td>1</td>
</tr>
<tr>
<td>Shuttle car operators</td>
<td>2</td>
</tr>
<tr>
<td>Timberman</td>
<td>1</td>
</tr>
<tr>
<td>Car trimmer</td>
<td>½</td>
</tr>
<tr>
<td>Electrician-mechanic</td>
<td>1</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>6</strong></td>
</tr>
</tbody>
</table>
The average daily production of raw coal per unit shift was 133.3 tons. This gave an average production of 22.2 tons of raw coal per shift for each man in a unit.

5.9.3. Ripper-type Mining Machine (Mine 1; Figure 15)

The mine is operated in the American Coal Bed in Alabama. This bed is 54 in. thick and is overlain by about 216 ft of overburden. A room-and-pillar system was used with pillars not extracted. Overall coal recovery was 65 percent.

Operating Procedure

The plan for room development is shown in Figure 24. Rooms A and B were driven making cuts 1-5 inclusive. Succeeding rooms were driven by repeating the sequence of cuts 6-17.

Loading and Haulage

The continuous-mining machine discharged the coal to the floor from where it was loaded by a mobile-loading machine to a 3 ½ ton capacity shuttle car, used as a surge car. These operations permitted the continuous-mining machine to be operated with a minimum of delay for alignment with or waiting on shuttle cars.

Roof Support

Wood crossbars, 14 ft long by 8 in. wide by 3 in. thick, set on steel screw jacks were used to support the roof. The crossbars were set on 4-ft centers as the continuous-mining machine advanced and were recovered (along with the steel screw jacks) when the room was completed.

Equipment Required

Each unit crew was equipped with a continuous mining machine, one mobile-loading machine, two shuttle cars, and one 26-in. or 30-in. panel belt.

Crew Required and Productivity

Travel time at this mine was 1 hr; no time lost for lunch (crew lunch period staggered); leaving 7 hr face time per shift. The average delay caused by mechanical failure was 39 min, 9.3 percent of face time. Other delays, as maneuvering the
continuous mining machine, timbering, power failure, bit change, greasing, and waiting for empty cars, amounted to 1 hr 24 min, 20 percent of the face time. Actual productive time was 4 hr 57 min, 70.7 percent of face time.

A crew of eight men in the typical continuous mining unit produced an average of 369 tons of raw coal per shift. This was an average of 46.1 tons of raw coal per man-shift per man in the production unit.

![Figure 15: Room-Mining with ripper-type continuous miner-mobile loading into shuttle cars, Mine 1](image)

5.10. Method Used for Pillar Extraction in Continuous (Non-cyclic) Mining

There are two general methods used for pillar extraction. These are (1) the open-end method; and (2) the “pocket-and-fender” or “split-and-fender” method.

(1) The open-end method involves taking lifts of coal from the side of the pillar adjacent to the mined-out area. In some cases no enders are left between the mining machine and the gob or goaf while in other cases thin pillars known as “fenders” are left to support the roof and protect the mining machine and crew from the caving roof-rock in the goaf.

(2) The “pocket-and-fender” or “split-and-fender” method involves splitting the pillar and then taking slices or lifts from the interior of the pillar.
Generally the open-end method, where it can be used, gives a higher percentage of coal recovery than the split-and-fender method because less coal is left un-recovered in the fenders or stumps. However, there are many mines where the open-end method cannot be applied because of adverse roof conditions.

Where pillars or stumps are left it is frequently necessary to blast them to secure proper caving of the roof after mining of a pillar is completed.

**Auxiliary Support**

Auxiliary supports used during the extraction of the last stump of a pillar or the final stump of a lift may include cribs, additional props, 20-ton hydraulic props, or 80-ton yielding-type steel props. Ropes are attached to the 20-ton hydraulic area when pillaring is completed.

**Examples of Pillaring Methods**

Figures 16 - 17 inclusive are taken from U.S. Bureau of Mines.

**General Requirements for Pillaring Operations**

(U.S. Bureau of Mines R.I. 5631)

1. After extraction of a pillar is begun speed in completing the operation is essential. If necessary it may be wise to finish a shift in another pillar, rather than leaving a small block stand to be recovered later.

2. Complete extraction of pillars if preferable to hogging or splitting as strong remnants tend to throw weight of the roof forward onto remaining pillars and cause difficulties with roof control. It may be necessary to blast remnants to remove them as supports.

3. An unobstructed runway is essential to allow quick removal of men and equipment if roof falls threaten.

4. Where a thin-fender or open-end method is employed long lifts are hazardous; it is preferable to take slices from alternate sides of a pillar rather than from the same side.

5. Hydraulic and yielding steel props appear to be safer than breaker props and cribs since they may be tripped and recovered from a safe area out by the pillar split.
(6) In roof-bolted area where pillars are being extracted with ripper-type continuous miners, occasional timbers set at outby intersections and in the intervening approaches to an active pillar, would warn of strata separation above the bolts, or of other roof movement not usually indicated by the bolts. Abnormal or fragile roof outby pillar places should be supported in accordance with the need.

Figure 16: Open-end pillaring with a boring-type continuous miner in Lower Kittanning coal bed using post timbering

Figure 17: Open-end pillaring with a ripper-type continuous miner in Lower Kittanning coal bed using post timbering.
CHAPTER 6
LONG-WALL MINING – CYCLIC OPERATIONS

6.1. Introduction

A full mining cycle at a long-wall face includes the following operations:
(1) undercutting the coal;
(2) drilling shot holes, loading them with explosives (or with mechanical pressure breaking devices), and shot firing to break down the coal;
(3) loading (filling) the broken coal onto the face conveyor;
(4) moving the conveyor over;
(5) moving forward the line of chocks or props which determine the caving lone of the roof, moving forward the back props, ripping the roadways, and building packs.

When coal is exceptionally soft or friable and/or when roof pressure can be brought to bear on the face coal to fracture the seam then coal may be gotten with hand-held pneumatic picks without the necessity for undercutting, drilling, or shot-firing.

6.2. Machines for Cyclic Long-Wall Mining

The coal cutter was the earliest form of mechanization applied to face operations in coal mining. The modern machines have evolved from early day machines which employed a horizontal rotating disk as the cutting element. The basic cutting mechanism in all modern cutters is a bar or “jib” with a toothed chain running around its perimeter.

6.3. Long-Wall Coal Cutters

There are a number of types of chain coal cutters in use on long-wall faces. The design varies from one to another but the basic principle is similar in all cases. The typical long-wall coal cutter consists of three parts as follows:
(1) The cutting unit,
(2) The hauling unit,
(3) The driving unit.
These parts are located by spigot-faced joints and are held rigidly together by high-tensile steel bolts and studs.

The cutting unit drives the cutter chain on the jib. Different makers have different specifications for cutting speeds, rate of travel, etc. The length of jib may be anything from 3 ½ to 9 ft. The length of jib used depends on the nature of the coal seam, geological conditions, the amount of output which can be handled, etc.

The haulage unit consists of two rope drums, one on each side of the machine, driven by the motor through a series of gears. Each drum carries 75 ft of 5/8 in. wire rope or 120 ft of ½ in. wire rope. Cutting speed can be adjusted between 1 and 5 ft/min. Flitting speed is about 23 ft/min.

The driving unit may be either electrical (a.c. or d.c.) or compressed air.

The height of the machine may be anything from 12 to 21 in. and the width between 2 and 2 ½ ft.

The speed of the cutter chain around the jib varies from 320 to 650 ft/min. The width of the kerfs cut may be between 3 and 8 in. but is usually 5 in.

Figure 1: Long-wall coal cutter

Figure 2: Universal arc-shearer
6.3.1. Height of Coal Cutters

In thin seams the height of a coal cutter is an important factor and account must be taken of the thickness of the seam, the amount of face convergence which will take place and the types of roof support bars in use. The width of the coal cutter will affect the width of the cutting track and the ease of turning the machine.

“Case Study”

Gummers

Cutters should be fitted with gummers which will effectively remove the cuttings from the cutter chain and will leave a clean undercut. Effective removal reduces the load on the driving motor and assists in the preparation of the coal. Gummers may be either paddle type or propeller type.

Height Adjustment

The height of the cutting jib may be adjusted to suit seam requirements. For a height of up to 13 in. the machine is raised bodily by a jack so that it rests on timber flats. For a height of 13 in. or over the gear head is inverted.

For greater heights of cutting jibs may be mounted on hydraulic turrets. The hydraulic lifting and lowering arrangement is housed in the turret itself and is completely isolated from dirt or damage.

Types of Jibs

Single-jib machines are used successfully in many collieries but difficulties are sometimes encountered where roof is sticky and there is need for both overcutting or cutting a dirt band simultaneously. In such cases coal cutters fitted with twin jib or with stepped jibs may be used. The stepped jib is useful in allowing roof bars to be extended over the face track while cutting at roof level.

In cases where there is need for the coal cutter to be mounted on an armored conveyor, the under-cutting at floor level is facilitated by the use of bent or slightly curved jibs.
6.3.2. Mushroom-Jib Coal Cutters

The coal-cutter jib may be so designed as to give a shear cut at the back of the web in addition to the usually horizontal cut. This shear cut may be made by using either a mushroom jib or a curved jib.

The mushroom head is incorporated in a special jib in such a manner that it can be fitted directly to any standard coal cutter. The mushroom head and the sprocket at the end of the jib are combined and thus the mushroom receives its drive direct from the standard cutting chain. The mushroom and the sprocket are mounted on a stationary vertical shaft which is supported by the bottom jib plate: this plate, together with the top jib plate, is of special construction and ensures maximum rigidity at the jib end.

The mushroom is mounted on ball bearing arranged so as to give rigidity as well as maximum protection against dust.

The A.B. mushroom jib can shear up to a maximum height of 10 in. from the floor and extension pieces are available in 2-in. steps by means of which the vertical shearing height can be extended an additional 14 in. (A mushroom jib fitted with an extension is designated as a “turret jib”.)

When a shear cut is required in the middle of a web as well as at the back, a twin turret jib is useful. This jib is used in conjunction with a special deep undercut gumstower.

For satisfactory jib performance it is necessary to use a gummer to remove the cutting to leave a clean undercut. Mushroom jibs are not recommended for over-cutting with turret turned downwards because there would be no method of removing cuttings from the shear cut and they would tend to jam the machine.

Figure 3: Hydraulic turret over-cutter with down-curved jib for roof cutting
6.3.3. Advantages of Curved Jibs and Turret Jibs

Some advantages claimed for curved jibs and turret jibs are:

1. The shear cuts reduce the amount of shot firing required to break the coal and in some cases entirely eliminate the necessity for shot firing.
2. The coal sizing may be improved with a larger percentage of larger sizes being produced.
3. Output per man-shift is increased.
4. The shear cut gives a straight face line and makes installation of support easier.

The following disadvantages of curved or turret jibs are noted:

1. Great care is needed when starting and finishing a cut at the corners of the face.
2. Power consumption is high.
3. Chains must operate at slower speeds.
4. A gum stower is essential.
5. Gas may accumulate in the shear cut.

Applications for Curved Jibs

Curved jibs have been applied in a variety of ways. Some of these applications are indicated in Figure 5. In some instance in a 4-ft thick seam on a face 600 ft long where the coal cutter was mounted on the armored conveyor and was equipped with a curved jib an overall face output of 7 tons/man-shift was obtained. The face was hand loaded. The cutter made a 2 ft 3 in. undercut and a 2-ft shear cut.
6.4. Power Loaders

Power loading machines were at first designed to load the strip of coal cut by a standard coal cutter. This strip was usually several feet wide and required wide loaders. Power loaders for long-wall faces did not achieve a real break-through until the mining system was changed so that a narrow web of coal was cut along the face and narrow loaders were designed to load it. These machines allowed the space between the last row of props and the face to be narrowed so that the roof could be supported by cantilever bars and an unobstructed space (prop-free face) provided in which the face conveyor could be located and along which the power loader could pass. The conveyors used were of a semi-flexible type which could be snaked over to the face as the advanced.

6.4.1. Types of Loaders

Power loaders for long-wall faces may be divided into three classes:

1. Machines which are designed especially for loading.
(2) Coal cutters which may also be used for loading by the addition of flights to the cutter chain.

(3) Combination machines which both cut and load the coal during one pass along the face.

**Case Study**

**(1) Loading Machines**

The Huwood loader is designed especially for loading. Earlier models of this machine were rope hauled along the face but more recent models are propelled by two hydraulic rams which push against hydraulic jack posts which are set between floor and roof.

Loading action is provided by a series of arms which project in front of the machine and which operate with a reciprocating motion to sweep the broken coal away from the face onto the face conveyor. This conveyor may be either a chain or an armored conveyor, or it may be bottom belt conveyor.

The Huwood loader can load at the rate of 1 ton/min; however, the coal must be well prepared to enable the machine to function successfully.

The loader can travel at a maximum speed of about 220 ft/hr but a speed of only 90 ft/hr is considered to be sufficient for the completion of loading out a face on shift. It is claimed that 360-420 ft of face can be loaded during a working shift. It has been recommended that the cutter should be kept at least 150-180 ft in front of the loader.

A seam from 2 ¼ to 3 ½ ft thick is most suitable for this type of power loading, although a low type of loader has been made to be used in seam as low as 1 ft 11 in.

![Figure 6: Face support system with Huwood double-jack-post loader](image)
(2) Loading with Modified Coal Cutters

The simplest form of power loader is created simply by reversing the picks on the cutter chain of the conventional coal cutter. A more efficient loader is created, however, by attaching loading flights to the cutter chain. The pick holders on the chain are adapted for attachment of the loading flights.

This form loading has been successful in the thinner hard seams, such as those from 2 to 3 ft thick where conditions are not suitable for plows.

Figure 7: Huwood loader.

Figure 8: Face support system for flight loading
The whole operation was based on a rigid cyclic system, involving undercutting the coal, drilling and blasting the coal during the preparation shift, and then attaching flights to the chain ready for the next shift.

In using these loaders the face is first pre-cut and suitably prepared for loading. To convert the cutter to a loader one pick is withdrawn from one of each pair of the flight-carrying pick holders, and flights are attached by means of drop-in pins. Usually one flight per foot of jib length is sufficient for loading purposes. The converted coal cutter then traverses the face in the direction opposite to the cutting direction, with the jib leading and the chain running in reverse so that the broken coal is scraped onto the conveyor.

In the second shift the coal was loaded by the flight loader and then permanent supports were set in the new track behind the machine. New stables for the cutter were also made on this shift.

The third shift consisted of moving the conveyor forward and withdrawing back supports. The road head was also advanced and the face packed during this shift.

6.5. The Scraper-Box Loader

The scraper-box is a relatively simple form of power loading used underground. The coal is cut and broken in the usual manner and is then scraped to the end of the face by dragging a scraper box to and fro along the face.

The box is attached to wire ropes which are actuated by a winch placed in the tail-gate road.

To hold the box against the face and in the coal a skid rail is usually set parallel to the face and is advanced by means of ratchet pushers as the face advances.

The scraper-box system eliminates the need for a face conveyor but it does have a low efficiency. This is partially offset, however, by the relative excellency of its performance in very thin seams where working conditions are very unfavorable.
A narrow design of scraper box with collapsible arms was introduced in a Scottish colliery and this enabled supports to be erected almost as close to the face as in ordinary practice. The scraper box was powered by a 100 h.p. winch with double drums. The specified rope speed was 200 ft/min and the maximum pull on the 7/8 in. diameter rope was to be 15,000 lb. The equipment was installed on a double-unit face in a 19-in. thick seam, each unit being 280 ft in length. The face was cut by a 12-in. double jib machine and the coal was scraped to the center loading road by scraper boxes on each side.

6.6. Semi-cyclic operation

The preceding discussion has dealt with cutting and loading machines for cyclic operations in which cutting, breaking down the coal, and filling it onto the face conveyor were separate operations.

Non-cyclic operations will be discussed in the next chapter and involve the use of machines which rip or cut the coal from the seam and load it onto the face conveyor without the necessity of drilling and shooting to break down the coal. Generally in long-wall “continuous” or “non-cyclic” mining a relatively thin slice of coal is removed from the face at each passage of the mining machine. Since these machines are relatively narrow, are usually mounted on an armored face conveyor, and only a narrow web of coal is removed at each pass, the roof at the face can be supported by bars cantilevered from a row of props set about 3 ft from the face. Thus coal getting operations are conducted on a “prop-free face”. A continuous mining (non-cyclic) system attempts to produce coal at a fairly steady rate during two, and sometimes all three shifts in contrast to cyclic operations where all coal is loaded out on one shift. The non-cyclic system tends to produce a relatively smooth flow to coal through the transportation system in contrast to the cyclic system which imposes peak loads on transportation during one shift.

Semi-Cyclic Systems

Between the “cyclic” systems and the “non-cyclic” systems there are some machines which operate on what might be called the “semi-cyclic” system. In these operations the coal is gotten without having to drill and shoot to break it down.
One such system is that known as “waffling”. This system may be used when the coal is of such nature that machine cutting of the face produces adequate breakage and no shot firing is required. In such a situation the coal is cut by one machine and is loaded by a flight loader which follows the cutter along the face.

The distance which the loader follows the cutter is determined by the nature of the coal and, whether it will settle and break immediately or whether some time must be allowed for breakage.

“Case Study”

6.6.1. The Meco-Moore Cutter Loader

The Meco-Moore cutter loader simultaneously cuts the coal and loads it onto a face conveyor. Initially the machine was designed to cut and load the coal alternately but it was later modified for simultaneous cutting and loading. The machine consists of two main sections: (1) the cutting units; (2) the loading unit.

(1) The cutting unit consists of a special long-wall coal cutter which is equipped with two horizontal jibs – one for undercutting and the other overcutting. The chains in the two jibs travel in opposite directions. This helps to stabilize the machine and to break up the coal.

(2) The loading unit comprises the following:

(a) Left and right-hand gimmers.
(b) Left and right-hand loader gear boxes with loader bars.
(c) Loader structure with belt.
(d) Shear jib.

Figure 10: Meco-Moore cutter loader
Since the machine is designed for work in either direction of travel the gear boxes for the gummers, the loader bars, and the shear jib are arranged symmetrically about the center line of the loader belt. This makes it possible for the cutter unit to be attached to either end of the loader unit.

The Meco-Moore cutter loader can be used under variable conditions. It cannot be used, however, in seams less than 3 ft thick and it requires a good roof and strong floor. Where the coal does not fall easily from the roof a picked drum may be fitted to the cutter loader. The picked drum also helps to break up large lumps of coal. The machine requires stable holes 15-30 ft long at each end of the face.

6.6.2. Cycle of Operation for One Coaling Shift per Day

The organization of the working cycle for one coaling shift per day may be as follows:

(1) Day shift: The face is loaded out by the cutter loader and the stables are made,

(2) Afternoon shift: The work consists of ripping, packing, and conveyor shifting. The cutter loader is turned and made ready to load the next cycle. The machine is examined and lubricated.

(3) Night shift: All work is completed and made ready for the loading shift.

The normal face crew during the production shift consists of fifteen to sixteen men.

6.6.3. Requirements for Efficient Operation

The Meco-Moore cutter loader runs on the floor of the seam and does not use the prop-free support system.

The efficient application of the machine depends upon the following factors:

(1) A definite system of operation and roof support should be adhered to.

(2) The face and props should be kept in a straight line. The conveyor should be laid close to the face props.

(3) The face conveyor should be adequate to deal with a large output.

In view of the rather cumbersome build of these machines and also because the Meco-Moore machine has to be turned around each time it traverses the length of the face it is not likely that there will be any great increase in the number of these machines working in the future.
CHAPTER 7
LONG-WALL MINING – NON-CYCLIC OPERATIONS

7.1. Machines for Non-Cyclic Mining

Machines for non-cyclic (continuous) operations on long-wall faces are designed to remove a relatively thin slice of coal (3 or 4 in. up to 30 in.) from the face during each passage along the face.

In contrast to this most cyclic or semi-cyclic machines cut relatively wide webs – some machines cutting webs up to 5 ft wide. Because the cut is so wide, such machines ordinarily can traverse the face only during one shift in each 24 hrs. The remainder of the time is used in moving the conveyor forward, moving and setting supports, ripping and packing, and turning the machine around in its stable hole. Principal disadvantage of this method of operation is that transport equipment must be large enough to handle a large volume coal on one shift but may be only partially employed during the remainder of the 24 hr. In addition the removal of a wide web of coal tends to stress the roof to a greater degree than does the removal of narrow webs, and the difficulty of roof control is increased.

With the non-cyclic system coal may be cut and loaded on two or even three shifts each 24 hr and a steadier flow of coal is maintained which can be handled by a smaller transportation system.

Non-cyclic machines are relatively narrow and most are designed to ride on the face conveyor so that the space required between the front row of props and the face is not normally more than 3 ft. Thus a “prop-free front” can be used and the roof between the front row of props and the face is supported with bars cantilevered from the front row of props.

Non-cyclic systems are most efficient when used in conjunction with powered self-advancing roof supports which require a minimum amount of man-power for operation.
7.2. The Anderton Shearer

The Anderton shearer is the machine most widely used for continuous long-wall mining in Great Britain.

The cutting head on this machine is a drum which rotates about a horizontal axis which is approximately perpendicular to the face line. Coal picks are set around the circumference of the drum. The drum revolves so that the picks cut upward into the coal and the broken coal is thrown back over the drum onto the plow portion of the machine from whence it falls onto the conveyor. Coal which does not come down during the shearing run is picked down, or shot down and falls into the cutter track.

The machine is used in conjunction with armored conveyors and is hauled along the face by a wire rope. When the machine completes a cutting run it is pulled back down the face the plow portion plows onto the conveyor any coal which may have been missed, or which has fallen onto the shearer track.

The depth of cut taken on each run may vary from 16 to 24 in. or more, depending upon thickness of the seam and the support problems which result from roof conditions.

Where a thick seam is to be worked, or where coal sticks to the roof, the Anderton shearer may be equipped with a jib for making a top cut, or with a curved jib which will make a top cut and also a shear cut. Because of the nature of its ripping action the Anderton shearer produces a low proportion of large coal.

Figure 1: Anderton shearer on a C20 armored conveyor, 150 h.p.
**Mode of Operation**

The Anderton shearer shears and loads from the buttock and during the shearing traverse of the face it travels in a direction opposite to the conveyor travel. It starts from the stable hole at the loader gate and proceeds along the face with the shearing drum revolving so that the picks cut upward into the coal and throw a large proportion of it over the drum and onto the plow which deflects it onto the conveyor.

When the machine reaches the stable at the tail gate one of two systems can be adopted:

1. The direction of the machine’s travel is reversed, during which it plows the coal onto the conveyor. This is the common method of operation.

2. In exceptional cases before starting the reverse haulage the drum may have to be taken off. This may be necessary in thin seams where roof convergence at the coal face may be enough to impede the drum progress if the drum is left in its original position. If the drum is not removed it is not possible to set bars right up to the face when the machine has passed in the cutting cycle. To overcome this difficulty segmented drums have been designed.

As the machine plows back the armored conveyor is moved forward. In thick seams where the coal does not part readily from the roof an overcutting jib may be employed.

**Shearing Unit**

The original machine was equipped with a series of disks with the cutting picks set on their perimeters. These disks were later superseded by a drum and more recently segmented drums have been used as the rotating elements on which picks are mounted. Segments may be removed from these drums to provide clearance when the machine is flitted back along the face after a cutting run.

The largest drum attempted was 72 in. in diameter but generally if the drum diameter of 50 in. gave good results without pre-cutting. Machines are now available up to 150 h.p.

Speed of travel during the cutting cycle may be from 5 to 30 ft/min, although the average is probably between 6 and 10 ft/min. Speed of travel during the plowing back cycle may be from 30 to 60 ft/min.
Advantages

The following advantages are claimed for this machine:

1. It is simple and versatile.
2. It can be adapted to variable geological conditions. The nature of the roof and floor does not affect its operation; it can be used in faulted areas.
3. It can be used for hard coal.
4. The price is comparatively low.

The only disadvantage is that it causes degradation of the coal and a large proportion of the production is less than 2 in. in size.

### 7.3. The Anderson-Boyes Long-Wall Trepanner

The A. B. long-wall trepanner, which produced some 23 million tons of coal in Great Britain in 1961, is designed for narrow web non-cyclic mining in thin seams. It travels on the floor of the seam alongside the conveyor and can cut and load coal when traveling in either direction. This machine attacks the coal on the buttock.

The trepanner has a length of steel channel bolted to the bottom which bears on the face side of the conveyor and serves as a guide. The machine pulls itself along (at speeds which may vary from 0 to 6 ft/min) by means of a stationary chain which is anchored to the conveyor at both ends.

The coal cutting and breaking mechanisms are the trepanning wheels which rotate on axes parallel to the coal face. The trepanning wheel has two cutting arms which carry the cutting tools. These cutting tools cut a thin annular groove which causes a cylindrical core to form. Heavy picks are provided on the trepanning wheel to break up this core.

A bottom jib is provided to cut a level floor for the machine and shearing jibs are provided to make side cuts. The gummings (coal cuttings) from the bottom jib are loaded onto the conveyor by paddles mounted on the back of the trepanner.

Water jets are mounted on the machine and sprays are directed at the coal buttock. The machine is equipped with a roof-cutting disk which has the function of cutting roof at the desired horizon or of dressing down sticking top coal.

The depth of the web is 27 in. and normal working height is 3 ft 3 in. to 4 ft although seams up to 4 ½ ft are worked where the top coal falls freely.
**Method of Operation**

The trepanner starts from a prepared stable and hauls itself along the face by means of a driving sprocket which winds on an anchored chain. It is guided by the face-side edge of the conveyor. The trepanner wheel takes off a 27-in. buttock of coal and discharges it onto the conveyor. The shear jib and the roof cutting disk cut the remaining top coal causing it to fall onto the sloping top of the machine body and slide onto the conveyor.

The floor cutting jib not only levels off the floor but also pre-cuts the face for the next run of the machine.

After the passage of the trepanner the space between the conveyor and the face is cleared of any spillage and the conveyor is snaked up to the new face again by means of power-operated rams.

New supports are set and the old supports are withdrawn from the waste edge.

When the trepanner enters the stable at the end of its run it is made ready for the return run.

Actual machine handling requires only an operator and a cable man. Almost all other personnel are engaged in setting and removing roof supports, and in preparing the stables.

9.3.1. Support Systems for Trepanner Faces

The support system for the A. B. long-wall trepanner should have the following basic characteristics:

1. It should allow for a conveyor advance of about 27 in. during each cycle.
2. It must be advanced progressively as the machine travels along the face and the rate of this advance should be at least 6 ft/min.
3. The supports should be so placed that the trepanner operator has good access to the machine controls at all times.

Some of the systems which are being successfully used in conjunction with the trepanner include the following:

1. Hydraulic Props and Steel Straps
   
   In this method the steel straps may be either 5 ½ or 7 ft long. Where conditions permit the face is supported on one series of 7-ft steel straps with three
props to a strap. As the machine advances along the face another series of straps is erected.

As an alternative method 5½ ft straps may be used. In this case the face is supported on two series of such straps, each supported by two props. As the machine advances a third series of straps is erected.

Figure 2: A.B. trepanner at work underground

Figure 3: Self-advancing supports on a trepanner face
(2) Hydraulic Props and Slide Bars

In this method hydraulic props carry special bar-slide heads and three such props carry a 7-ft joist. The joist is held against the roof by the wedge action of special heads. When these wedges are released the bar is slid along towards the newly exposed face and tightened against the roof by operating the wedges. This is less expensive in manpower than the application of props and steel straps described above, but requires that the roof be relatively smooth and without undulations.

(3) Link Bars and Hydraulic or Friction Props

This method is used with either hydraulic or friction props and the bar used is normally twice the depth of the web taken so that alternate settings are extended for each cut. The bar size must be carefully chosen to conform to the depth of web and under normal conditions 4 ft 4 in. bars are chosen.

7.3.2. Mechanized Support Systems

The A. B. trepanner is suitable for use with systems of mechanized support. The self advancing supports have been used in conjunction with a trepanner and the general layout for the operation of such a system is shown in Figure 5.3.

At the commencement of the cycle of operations the rear ends of the roof supports are all in line, with the roof beams set close to the face and providing cantilever support over the conveyor.

As the trepanner moves along the face both props of the two-leg hydraulic support unit immediately behind the machine are retracted and the whole unit is drawn up to the conveyor. The props are then re-pressurized and the roof bar brought up to bear on the newly exposed roof at setting loads which may be varied up to 10 tons/prop.

The conveyor is then snaked over to the face by extending a jack on the two-leg unit which pushes the conveyor over to the face.

After the conveyor is in its new position the props of the three-leg unit are retracted and the jack is closed to draw it up to the conveyor. The three props are then re-pressurized and the units are once again in line, completing a cycle.
Two men are required for the operation of the supports on a face equipped with the self advancing supports, while a third man advances the conveyor.

7.4. New Types of Continuous Mining Equipment

“Case Study”

7.4.1. The Mawco Cutter Loader

The Mawco cutter unit is designed so that it can be attached to any standard long-wall coal cutter which is equipped with a horizontal drive shaft for disk shearing.

As the cutter is drawn along the face the single cutting chain cuts a kerf completely around the perimeter of a strip of coal, freeing it from the solid at the top, bottom, and back, while at the same time a horizontal rotating breaker bar breaks the strip of coal and a plow attached behind the cutter deflects the broken coal onto the face conveyor.

The machine has shown an ability to produce a large proportion of plus 2-in. coal, and for that reason the number in use will probably increase markedly during the next few years.

7.4.2. The Dranyam Power Loader

The cutting unit in the Dranyam loader is a drum, set with picks, which rotates about a vertical axis. The machine is mounted on an armored conveyor and as it is pulled along the face rotating picks rip coal from the face and deflect it onto the face conveyor.

The principal point favoring this machine is that it makes its own stables, and therefore allows a reduction in the large proportion of the face manpower which is normally required for this operation.

7.4.3. The Dawson Miller Stable Hole Machine

The Dawson Miller stable hole machine is a new development and has been designed to mechanize the extraction of stable holes thereby reducing the face manpower requirements and increasing the rate of face advance.
The machine consists of a cutting unit which travels on a specially constructed rigid conveyor frame. The cutting unit consists of a motor and a gearbox which drives a rotating cutting disk the diameter of which determines the height of the stable hole. This unit travels from end to end of the rigid conveyor frame which extends for the full length of the stable.

The machine is essentially a very narrow web Anderton shearer, which cuts a ½ in. web each time it traverses the face. The cut coal is conveyed on the rigid frame conveyor and discharged onto an auxiliary conveyor which conveys it to the main system.

Figure 4: Dawson Miller stable hole machine (right-hand)

Power Units

The conveying and traversing mechanism is driven by a 7 ½ h.p. motor and gearbox arrangement. The 7 ½ h.p. motor also drives a hydraulic power pack which is built into the drive head section of the conveyor and which supplies oil for push rams, for lifting jacks for horizon control, for chain tensioning rams, and to drive the auxiliary conveyor.

A four-speed gearbox fitted to the drive unit allows a choice of cutting speeds between 16 and 41 ft/min depending upon the hardness of the coal. The cutting unit is driven by a 15 h.p. electric motor and speed-reducing gearbox.
Cutting Unit

The rotating cutting disk equipped with six arms, each holding a cutter pick, is mounted on the output shaft of the speed-reducing gearbox which is driven by a 15 h.p. motor. The whole assembly is mounted on a carriage which is traversed continuously and automatically from end to end of the rigid frame conveyor along guide rails.

Traverse Mechanism

This mechanism automatically traverses the cutting unit from end to end of the frame and automatically reverses its direction at the end of each traverse.

Advance Mechanism

The conveyor frame is automatically advanced by means of hydraulic pushing rams attached to it. The amount of advance or depth of web is controlled by toe plates at floor level, one at each end of the frame, which are maintained in contact with the stable face by the action of the pushing rams. As the cutting disk cuts by the toe plates, the resistance offered to them is removed. This allows the pushing rams to automatically advance the conveyor frame until the toe plates again contact the face, thus sumping in the cutting disk for a further web to be taken.

Conveying System

The rotating disk loads the cut material onto the deck plate of the conveyor. A 14 mm round-link chain with cantilever flights then conveys the material to a discharge port in the deck plate of the conveyor through which it passes onto the auxiliary conveyor. This conveyor is designed to allow the stable hole machine to advance 4 ½ ft without requiring movement of the face conveyor or stage loader.

7.4.4. Joy-Sullivan Road Ripping Machine

The Joy-Sullivan road ripping machine has the following characteristics: (a) It can be mounted and operated in a standard roadway producing an excavation 13 ft wide by 10 ft high. (b) It incorporates provision for advancing the machine. (c) The cutting drums are designed for sumping as well as shear cutting. (d) Dust suppression
is catered for by the use of integral water sprays: (e) Debris disposal is arranged for by scraper packing or by transference through the center of the machine by conveyor.

The machine consists of:

(a) A skid mounted base, moved and positioned by wire ropes actuated by hydraulic cylinders. (b) A cutter head frame or carriage mounted on the base advance or retracted hydraulically in relation to it, over a range of 2 ft 3 in. (c) A large diameter tube supported in the cutter head frame, which provides the pivot for the cutter arm and the mounting for the cutter drive. (d) A transmission drive consisting of a 60 h.p. electric motor, fluid coupling, reduction and drive shafts to the three cutter drums. (e) A cutter arm mounting the three cutter drums which is positively controlled in the vertical plane by hydraulic cylinders. (f) Three cutter drums each fitted with tungsten carbide insert picks on the periphery and face, and staggered in relation to each other to assist in the clearance of cuttings. Integral water sprays provide a jet of water at each cutter pick point. (g) A twin jack arrangement mounted at the top of the cutting head frame, giving a positive loading between the roof arch girders and the machine, to give complete stability to the machine when cutting and also to assist in the setting of the roof girders. (h) An hydraulic pump driven from the main motor and operating the hydraulic cylinders.

Figure 5: Joy-Sullivan road ripping machine
Operation

(1) The machine is positioned with the center line of the drive on the center line of the roadway, the cutter drums touching the face of the rip with the cutter head retracted on the base, and the cutter arm in the horizontal position to cut at the bottom of the rip on the operator’s side.

(2) The roof jacks are raised against the roof to stabilize the machine.

(3) The machine is sumped into the face up to 12 in. by advancing the cutter head frame using the carriage jacks. An indicator is mounted on the base to show the depth of the sumping.

(4) The cutter arm is turned in the vertical plane through 180°, cutting the whole face, using the cutter arm jacks.

(5) The arm is returned to its initial position.

(6) Operations 3, 4 and 5 are repeated to take a further 12 in. cut.

(7) Chains are fastened to the front of the extension rods of the carriage jacks from anchor points under the lip, the roof jacks lowered and the machine base advanced forward.

The approximate times for the above operations are:

- Sumping in 12 in. – ½ min;
- cutting – 2-3 min;
- returning cutter arm – 1 min;
- advancing base after fixing chains – 1 min.

Roadway Excavation Sizes

The machine is made suitable for three sizes of roadway excavation. In addition packing pieces can be used to increase the cutting height up to a further 12 in. The following list shows the various excavation sizes and the minimum height required under the ripping lip.

<table>
<thead>
<tr>
<th>Model</th>
<th>RR.227</th>
<th>RR.245</th>
<th>RR.246</th>
</tr>
</thead>
<tbody>
<tr>
<td>Width of excavation</td>
<td>ft</td>
<td>in.</td>
<td>ft</td>
</tr>
<tr>
<td>Standard</td>
<td>13</td>
<td>15</td>
<td>17</td>
</tr>
<tr>
<td>with 6 in. packing</td>
<td>10</td>
<td>11</td>
<td>12</td>
</tr>
<tr>
<td>with 12 in. packing</td>
<td>11</td>
<td>12</td>
<td>13</td>
</tr>
<tr>
<td>Width of finished roadway</td>
<td>12</td>
<td>14</td>
<td>16</td>
</tr>
<tr>
<td>Height of excavation</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Standard</td>
<td>10</td>
<td>11</td>
<td>12</td>
</tr>
<tr>
<td>with 6 in. packing</td>
<td>10</td>
<td>11</td>
<td>12</td>
</tr>
<tr>
<td>with 12 in. packing</td>
<td>11</td>
<td>12</td>
<td>13</td>
</tr>
</tbody>
</table>

Minimum heights of ripping lip are as follows: No packing – 2 ft 6 in.; with 6 in. packing – 3 ft; with 12 in. packing – 3 ft 6 in.
**Stone Disposal**

In a 3 ft 6 in. seam a 12 in. cut gives 5 tons of stone. In a 2 ft 6 in. seam the tonnage is 6 tons. This is produced in 2-3 min plus ½ min for sumping, so that the rate of production of stone may be at 2 and even more tons per minute. In considering method of stone disposal it is necessary to provide a system which can operate at these rates or provide bunkerage capacity, or the cutting will have to be intermittent. It is undesirable to use the machine cutting at slower speeds because this results in increased dust production.

![Figure 6: Slusher packing of stone from a road ripping machine](image)

Four methods of stone disposal have been tried:

(a) Loading on to a center conveyor passing through the machine. This is capable of dealing with maximum rates of cutting. Chutes from the sides of the roadway to the center conveyor eliminate all handfilling. Alternatively, a conveyor along the side of the roadway may be used. A cross conveyor can be fixed to the machine immediately below the cutting arm to give transverse loading of the stone. This cross conveyor is driven hydraulically from a power take-off from the machine.
(b) Slusher packing. This system is well known, there being about 700 units in operation. It provides a very effective means of stone disposal with the ripping machine. The rate of loading is dependent on the length of run of the packing bucket. On average the rate of packing is about 1 ton per min, but in practice the floor immediately below the cutting arm forms bunker.

The team required is two for a 5 or 6 ft advance per shift and three for a 10 or even 15 ft advance per shift. In the first case one man is positioned on the face at the pack-hole and one man operates the two machines. About one-third of the cut is taken and then the stone cut is slusher packed. The two operations are repeated with the second and third sections of the cut. With three men operating, one man again is on the face, one man on the ripping machine, and one man on the slusher. Cutting and packing is continuous and slushing is continued whilst the cutting arm is returned to its original position and until all the stone is packed.

(c) Gob flinging. A standard long-wall coal cutter fitted with a gum-stower head was sited alongside the ripping machine. The stone was delivered to it by a cross front conveyor and then flung into the pack hole. In practice the results were not successful, mainly due to the lumps produced causing blocking, and wet dirt sticking to the gum-stower. The speed of stowing when operating was under ½ ton/min, which necessitated a very slow cutting arm speed of 10-12 min. A 5 ft advance was obtained in 5 ½ hr with four men.

(d) Pneumatic stowing. In this system the stone was again loaded by cross-conveyor onto a conveyor alongside the machine and then to a Markham crusher stower. The stowing pipe was carried through the machine to a 90° bend for stowing in the pack-hole. The results showed that a 5 ft advance may be obtained with three men per shift and a 10 ft advance with four men per shift. The main disadvantage experienced with the apparatus used was that the rate of stone disposal does not keep up with the rate of cutting. Again a cutting arm speed of 10 min was necessary giving a maximum loading rate of about ½ ton/min. Either some form of bunkering is required or a greatly increased stowing capacity.

From the various time studies taken, rates of advance of 30 ft per shift are possible in a solid heading where the stone is loaded away from the machine by conveyor, assuming the support setting can be accommodated in 20 min/yd. On long-wall faces slusher packing can give an advance of up to 15 ft/shift.
In seams greater than 4 ft thick, slusher packing of roadside packs is not normally practiced in that the length of pack is not sufficiently long to give efficient operation. It is probable in these seams that pneumatic stowing will prove the best method of stone disposal. As explained the standard crusher-stower has a limited capacity and a larger unit of at least 1-1½ tons/min is advisable, incorporating a crusher unit to break down any oversize obtained.

Sequence of Operations

The following is a sequence of operations for a 5-ft advance using scraper packing and is repeated for a 10-ft advance. Normally two men are required for a 5-ft advance per shift and three men for a 10-ft advance per shift.

The following shows a time study of a typical operation.

Breakdown Operations

10-ft. advance – 2 packs of 5 ft. – 3 men.

<table>
<thead>
<tr>
<th>Operation Description</th>
<th>Time (min)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Set staker props, prepare pack hole, thread ropes for scraper packing</td>
<td>25</td>
</tr>
<tr>
<td>2. Prepare lip and set lip supports</td>
<td>20</td>
</tr>
<tr>
<td>3. Advance base</td>
<td>2 ft</td>
</tr>
<tr>
<td>4. Rip and scraper pack</td>
<td>1 ft</td>
</tr>
<tr>
<td>5. Advance base</td>
<td>1 ft</td>
</tr>
<tr>
<td>6. Move lip supports</td>
<td>2 ft</td>
</tr>
<tr>
<td>7. Rip and scraper pack</td>
<td>1 ft</td>
</tr>
<tr>
<td>8. Rip and scraper pack</td>
<td>1 ft</td>
</tr>
<tr>
<td>9. Advance base</td>
<td>2 ft</td>
</tr>
<tr>
<td>10. Move lip supports</td>
<td>2 ft</td>
</tr>
<tr>
<td>11. Rip and scraper pack</td>
<td>1 ft</td>
</tr>
<tr>
<td>12. Rip and scraper pack</td>
<td>1 ft</td>
</tr>
<tr>
<td>13. Draw rope from pack, clean-up and finish packs</td>
<td>10</td>
</tr>
</tbody>
</table>

Total for 5-ft advance: 125 min

Total for 10-ft advance: 250 min

Setting girders – 3 at 25 min each: 75 min

Approx, total time for 10-ft advance: 5 ½ hr

(1) Set machine chain staker-props, return sheaves, pull forward ripping machine base, then set slide bars and props in position. (2) Cut 1 ft and scraper pack.

**Supports**

The roof jacks with the girder support can carry the center section of a three piece arch, to support the roof between the face of the rip and the last permanent setting. When a new arch requires to be set, the center section is then already positioned for bolting of the side legs. Alternatively, two piece arches may be used and the extension piece can carry the forward temporary support. With the profiled section of roadway cut, setting of the arches is a simple and rapid operation.

The timbering system for support of the roof under the ripping lip, when stone disposal is by scraper packing, may use slide bars and cabbage heads on hydraulic props, bull-rails being added if required. When another form of stone disposal is used, straight bars or girders across the roadway supporting the lip are preferable.