IRON-ORE (HEMATITE) MINING PRACTICE
IN THE BIRMINGHAM DISTRICT, ALA.

BY

W. R. CRANE
CONTENTS

Introduction--------------------------------------------------------------- 1
Acknowledgments---------------------------------------------------------- 2
History and early development------------------------------------------ 2
  Early history of the district---------------------------------------- 2
  Early mining methods----------------------------------------------- 3
    Hand stripping and wagon haulage-------------------------------- 4
    Haulage by tram lines--------------------------------------------- 4
    Development by drifts and slopes-------------------------------- 6
    Mining of hard ores---------------------------------------------- 7
    Haulage methods------------------------------------------------- 7
    Mining methods------------------------------------------------- 8
Recent practice--------------------------------------------------------- 8
Character and occurrence of ore---------------------------------------- 10
  Number and thickness of beds--------------------------------------- 10
    Definition of ore----------------------------------------------- 10
    Big Seam and Irondale bed------------------------------------- 10
  Kinds and character of ore---------------------------------------- 11
  Grade of ores----------------------------------------------------- 12
Variation of constituents throughout district------------------------ 13
Physical properties-------------------------------------------------- 14
Occurrence of ore; structural irregularities-------------------------- 16
  Dip of ore beds--------------------------------------------------- 17
  Folds and faults--------------------------------------------------- 17
  Rolls, pots, and kettles------------------------------------------ 18
  Slips and cross-bedding------------------------------------------ 18
Crushing and compressive strength of ores---------------------------- 21
Mining methods-------------------------------------------------------- 22
  Development of mines in the upper bench, Big Seam---------------- 22
  Objections to abandonment of present mining methods----------- 22
Arguments in favor of change of method------------------------------ 23
  Changes of equipment-------------------------------------------- 23
  Abandonment of transportation system----------------------------- 23
  Abandonment of mining villages--------------------------------- 23
  Sinking shafts through water-bearing strata---------------------- 24
  Transportation of miners---------------------------------------- 24
  Prejudice of miners--------------------------------------------- 24
Development by vertical shafts---------------------------------------- 24
Development by slopes------------------------------------------------ 25
Development for trip or car haulage---------------------------------- 25
  Slopes------------------------------------------------------------- 25
  Manways----------------------------------------------------------- 26
  Headings---------------------------------------------------------- 27
Plan of development-------------------------------------------------- 28
Protection of workers------------------------------------------------- 29
Width and arrangement of slopes and headings------------------------ 30
Change in method of development for skip haulage--------------------- 30

III
<table>
<thead>
<tr>
<th>Mining methods—Continued.</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Development by inclined and vertical shafts</td>
<td>31</td>
</tr>
<tr>
<td>Plans for Shannon slope</td>
<td>32</td>
</tr>
<tr>
<td>Methods of driving slopes, manways, and headings</td>
<td>33</td>
</tr>
<tr>
<td>Definitions</td>
<td>33</td>
</tr>
<tr>
<td>Driving slopes</td>
<td>33</td>
</tr>
<tr>
<td>Driving manways</td>
<td>34</td>
</tr>
<tr>
<td>Driving headings</td>
<td>35</td>
</tr>
<tr>
<td>Extraction of ore</td>
<td>36</td>
</tr>
<tr>
<td>Driving upsets</td>
<td>36</td>
</tr>
<tr>
<td>Stoping practice</td>
<td>38</td>
</tr>
<tr>
<td>Changes in practice</td>
<td>39</td>
</tr>
<tr>
<td>Branch tracks</td>
<td>39</td>
</tr>
<tr>
<td>Gravity planes</td>
<td>41</td>
</tr>
<tr>
<td>Scraper loaders</td>
<td>41</td>
</tr>
<tr>
<td>Driving headings with scraper loaders</td>
<td>43</td>
</tr>
<tr>
<td>Development of stopes with scraper loaders</td>
<td>44</td>
</tr>
<tr>
<td>Working the lower-bench ores</td>
<td>45</td>
</tr>
<tr>
<td>Early mining methods</td>
<td>45</td>
</tr>
<tr>
<td>Present methods</td>
<td>46</td>
</tr>
<tr>
<td>Working the upper and lower beaches together</td>
<td>49</td>
</tr>
<tr>
<td>Mining at Shannon slope</td>
<td>49</td>
</tr>
<tr>
<td>Robbing pillars</td>
<td>50</td>
</tr>
<tr>
<td>Slabling pillars</td>
<td>51</td>
</tr>
<tr>
<td>Breakthroughs</td>
<td>51</td>
</tr>
<tr>
<td>Splitting pillars</td>
<td>52</td>
</tr>
<tr>
<td>A-headings</td>
<td>53</td>
</tr>
<tr>
<td>Splitting pillars in irregular areas</td>
<td>54</td>
</tr>
<tr>
<td>Robbing entire thickness of Big Seam</td>
<td>54</td>
</tr>
<tr>
<td>Drilling and blasting</td>
<td>55</td>
</tr>
<tr>
<td>Seams and slips</td>
<td>56</td>
</tr>
<tr>
<td>Drilling holes for blasting</td>
<td>57</td>
</tr>
<tr>
<td>Drilling equipment</td>
<td>57</td>
</tr>
<tr>
<td>Drills</td>
<td>57</td>
</tr>
<tr>
<td>Compressed-air supply</td>
<td>59</td>
</tr>
<tr>
<td>Drill bits</td>
<td>59</td>
</tr>
<tr>
<td>Explosives</td>
<td>60</td>
</tr>
<tr>
<td>Support of mine workings</td>
<td>61</td>
</tr>
<tr>
<td>Conditions affecting support</td>
<td>61</td>
</tr>
<tr>
<td>Water-bearing formations</td>
<td>61</td>
</tr>
<tr>
<td>Strength of formations</td>
<td>61</td>
</tr>
<tr>
<td>Relationship of beds</td>
<td>62</td>
</tr>
<tr>
<td>Dip of formations</td>
<td>62</td>
</tr>
<tr>
<td>Summary of conditions</td>
<td>64</td>
</tr>
<tr>
<td>Strength of ore</td>
<td>65</td>
</tr>
<tr>
<td>Weakness of associated formations</td>
<td>66</td>
</tr>
<tr>
<td>Beds of “slate” and sandstone</td>
<td>66</td>
</tr>
<tr>
<td>Joint planes</td>
<td>66</td>
</tr>
<tr>
<td>Cross-bedding planes</td>
<td>67</td>
</tr>
<tr>
<td>Props</td>
<td>68</td>
</tr>
<tr>
<td>Effect of dip on support</td>
<td>69</td>
</tr>
<tr>
<td>Effect of irregularities in occurrence</td>
<td>71</td>
</tr>
<tr>
<td>Effect of faults</td>
<td>72</td>
</tr>
</tbody>
</table>
Handling ore in mines—Continued.

Effect of variation in dip of bed on handling of ore.......................... 108
Use of engine or gravity planes............................................. 108
Rock tunnels.............................................................................. 108
Connecting separated parts of ore bed..................................... 104
Combined skip and trip haulage.............................................. 105
Mule haulage in mines............................................................. 106
Care of mules............................................................................ 107
Power-driven equipment for handling ore............................... 107
Surface handling of ores.......................................................... 108

Drainage .................................................................................... 110

- Conditions affecting drainage................................................. 110

Sources of mine water............................................................. 111

Surface water........................................................................... 111

Water from porous formations.................................................. 111
Water entering through fault planes........................................ 112
Water entering through prospect holes..................................... 112

Preventing entrance of water................................................... 112

Drainage of surface water....................................................... 113

Underground drainage............................................................. 113

Catch basins, reservoirs, and sumps........................................ 115

Group drainage systems......................................................... 117

Unfavorable conditions affecting drainage............................. 118

Natural conditions..................................................................... 118

Conditions resulting from mining............................................ 118

Caving of top rock.................................................................... 119

Disturbed formations............................................................... 119

Pumping equipment................................................................. 121

General conditions.................................................................... 122

Ventilation ............................................................................... 122

Conditions affecting ventilation.............................................. 122

Ventilation of shallow workings.............................................. 123

Ventilation of deep workings................................................... 123

Effect of outside temperature.................................................. 124

Effect of haulage....................................................................... 124

Ventilation of remote workings............................................... 125

Atmospheric impurities............................................................ 125

Summary of conditions affecting ventilation........................ 126

Suggested improvements......................................................... 127

Miscellaneous practice........................................................... 128

Lighting...................................................................................... 128

Signaling..................................................................................... 129

Sanitary conditions................................................................. 129

Fire hazards.............................................................................. 130

Concentration of high-silica ores............................................. 130

Objections to mining lower-bench ores.................................... 131

Rebuttal..................................................................................... 131

Suggested changes in mining practice.................................... 132

General considerations affecting work................................... 132

Control of influx of water into mine....................................... 133

Changes in development......................................................... 134

Advantages of vertical shafts................................................... 134

Changes in mining................................................................. 135
CONTENTS

Suggested changes in mining practice—Continued. ........................................... 136
Applicability of caving method ................................................................. 136
Change in ventilation ..................................................................................... 137
Publications on metal mining ........................................................................ 138
Index ...................................................................................................

ILLUSTRATIONS

Figure
1. Open-cut workings on outcrop, Red Mountain ........................................... 4
2. Removing entire thickness of Big Seam by open cut ................................... 5
3. Railroad tracks for ore cars on Red Mountain ........................................... 8
4. Surface works of large mine, Red Mountain .............................................. 9
5. Diagrams showing apparent and true specific gravity, percentage of porosity, and the principal constituents (Iron, insolubles, and lime) of ore from Red Mountain .......................................................... 14
6. Variation in constituents of ore on dip of seam, upper bench, Big Seam, Red Mountain, mine A .................................................................................. 15
7. True dip and direction of dip, taken along outcrop of Red Mountain ......<16
8. Fault plane exposed in stope ...................................................................... 18
9. Pot or kettle in draw-slate top rock ........................................................... 19
10. Filled pothole in ore bed ........................................................................... 19
11. Direction of major and minor slips and of inclination of Big Seam ......... 20
12. Slip planes cutting ore bed in open cut ..................................................... 21
13. Cross-bedding showing in roof of slope ................................................... 21
14. Barrier form of manway construction for steep dips ............................. 27
15. General plan of development of an iron-ore mine .................................. 28
16. Method of advancing slopes and manways ............................................. 29
17. Method of advancing manways and slopes where both manways are carried in advance of slope ............................................................. 30
18. Plan of proposed workings in Shannon slope ......................................... 32
19. Section and plan of face of slope, showing arrangement of holes .......... 33
20. Section and plan of face of manway, showing arrangement of holes for normal occurrence of ore ................................................................. 34
21. Section and plan of face of manway, showing arrangement of holes when ore bed is thin and readily broken ........................................... 35
22. Section and plan of face of heading and stope, showing arrangement of holes; overhand stoping ................................................................. 36
23. Section and plan of face of heading and stope, showing arrangement of holes; overhand stoping ................................................................. 36
24. Section and plan of face of heading and stope, showing arrangement of holes; underhand stoping ................................................................. 37
25. Common arrangement of slopes, manways, and stopes ......................... 38
26. Section and plan of face of upset, showing arrangement of holes .......... 38
27. Use of branch tracks in driving wide upsets on low dips ....................... 40
28. Use of branch tracks in working wide stopes .......................................... 40
29. Use of gravity planes in working large stopes .......................................... 41
30. Systematic timbering of large stopes ......................................................... 42
31. Arrangement of stopes for scraper loaders .............................................. 44
32. Early method of working lower-bench ore .............................................. 46
33. Working lower bench, Big Seam, by cross headings ............................... 47
34. View of stope, showing removal of lower-bench ore ............................. 48
35. Workings in lower bench, upper bench standing .................................... 50
36. Reduction in size of pillars by slabbing and by driving upsets .............. 51
37. Pillars left at regular intervals when robbing accompanies mining .......... 52
38. Irregular formation of breakthrough in arch pillar .......................... 52
39. Use of cross-heading and switchback tracks in mining pillars on high dips ........................................................................ 53
40. Use of A-headings in removing ore from pillars ................................... 53
41. Use of switchback tracks on high dips .............................................. 54
42. Use of cross headings in mining lower-bench and arch pillars on moderate dips ........................................................................ 55
43. Slope driven across instead of with slip planes .................................. 57
44. Drilling at stope face ........................................................................ 58
45. Ratio of distance between points cut by cross-bedding planes and top and bottom of beds, thickness of roof ore and sandstone top rock, and percentage of slate in first 5 feet above Big Seam ......................... 63
46. Roof ore left as support for draw slate .............................................. 64
47. Slip plane in top rock and fallen mass of slate .................................. 67
48. Mud slip crossing stope ..................................................................... 67
49. Failure of top rock along cross-bedding planes .................................. 68
50. Effect of cross-bedding on support by props ..................................... 69
51. Column of slate remaining above top of prop ..................................... 70
52. Ore bed dipping steeply ....................................................................... 71
53. Bad condition of top rock next to fault ............................................. 72
54. Extensive timbering in old stope ......................................................... 73
55. Use of props under normal condition of roof ...................................... 74
56. Long props used when all of Big Seam is worked ................................. 75
57. Pack walls of waste rock built between props ..................................... 76
58. Use of long props in high stope ......................................................... 77
59. Timbering in heading with weak roof ................................................ 78
60. Manway in iron mine ......................................................................... 79
61. Heave in slate footwall ....................................................................... 81
62. Pillar failing under pressure ................................................................ 82
63. Breaking of pillar under pressure ...................................................... 83
64. Rotary dump above ore pocket .......................................................... 90
65. Motor haulage in iron mine ............................................................... 91
66. Sledging ore in stope .......................................................................... 91
67. Racked or trimmed mine car ............................................................. 92
68. Use of balanced or gravity planes on moderate dips ............................ 95
69. Heading for scraper loaders ............................................................... 97
70. Cross heading extending through pillar ............................................ 99
71. Removing pillars by use of cross headings ........................................ 100
72. Steep-grade cross heading or "hill" track ........................................... 101
73. Extension of work beyond fault and method of handling ore ............ 104
74. Vertical-switch track in split slope ..................................................... 105
75. Combined skip and trip haulage on disturbed ore bed ..................... 106
76. Rock house or tipple for trip haulage ................................................ 108
77. Rock house or tipple, showing dumping rails and ore bin ................. 108
78. Skip and dumping rails above ore bin at rock house or tipple .......... 109
79. Drainage ditch extending through pillar .......................................... 114
80. Catch basin in ore mine .................................................................... 115
81. Pipe line for carrying water from one part of mine to another .......... 115
82. Main sump in ore mine .................................................................... 116
83. Discharge of mine water into main sump ........................................... 116
84. Making sump out of old stope .......................................................... 117
85. Brick stopping for holding back mine water ..................................... 118
86. Extensive caving of top rock, allowing admission of water ............... 120
87. Water entering mine through cave, as seen through upset ............... 121
IRON-ORE (HEMATITE) MINING PRACTICE IN THE
BIRMINGHAM DISTRICT, ALA.

By W. R. Crane

INTRODUCTION

Mining of the red iron ores of the Birmingham district, Alabama, has gone on energetically for the past 50 years and has created a large iron and steel manufacturing center, the only important one in the South. The district produces about 10 per cent of all the iron ore mined in the United States and 80 per cent of the Alabama output is red ore; moreover 43 of the 419 blast furnaces of the country are in the district tributary to Birmingham and produce 7 per cent of the pig-iron output. The rapid growth of the district has been made possible by investigations that have led to radical changes in furnace practice; studies of methods by which the low-grade, high-silica ores may be treated by concentration\(^1\) will give still greater impetus to development.

The mode of occurrence of the ore has made mining practice comparatively simple, but with the rapid extension of the workings underground and the disturbed condition of the ore bed some distance from the outcrop, conditions tend to become more difficult. Stoping, handling, support, drainage, and ventilation are all affected, sometimes favorably and sometimes adversely, by the extended workings and increased depth of cover.

Should beneficiation make available the high-silica ores of the lower bench of the Big Seam, mining practice must necessarily be adapted to the new conditions and will be rendered more difficult as the weight of cover increases at long distances from the outcrop. The support of workings will require more attention as the weight of cover increases, and the efficient and economical operation of the mines will depend largely upon improvements in mining and handling the ore.

The mining methods discussed in this report have been considered historically—that is, from early to recent practice. In addition, sug-

\(^1\) Singewald, J. T., Concentration experiments with the siliceous red hematites of the Birmingham district, Alabama: Bull. 110, Bureau of Mines, 1917, 91 pp.
gestions are offered for improved and changed practices that investiga-
tions have shown may be necessary with future development. Exper-
ence and practice in other districts where conditions are similar or near-
ly similar have been drawn upon for these sugges-
tions and recommendations which, although they may not be wholly
applicable, may point the way to success in future practice.
Data for this report were collected before 1923.

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HISTORY AND EARLY DEVELOPMENT

EARLY HISTORY OF THE DISTRICT

The first explorers of Alabama mineral lands were blacksmiths
and mechanics mustered out of Andrew Jackson's army after the
War of 1812. Recognizing the "red rock" of the Birmingham
district as iron ore, these men in a very small way utilized it to make cooking utensils and farm implements.

Progress in mining and furnace practice for the iron ores may be shown to best advantage by a series of chronological statements. The first blast furnace was built and operated at Russellville in 1818 by Joseph Heslip, who also built a Catalan forge, a foundry, and a crude rolling mill on Cedar Creek and named the furnace after the creek. In 1827 coal was first mined in the Warrior field; in 1830 Daniel Hillman erected the Roups Valley forge, later known as the "Tannehill," and now owned by the Republic Iron & Steel Co. In 1854 the first coke was made from Alabama coal and was used in foundries; in 1858 the first rolling mill was built at Shelby; in 1864 the red hematite ore of Red Mountain was first used in blast furnaces, the ore being mined at Irondale and used in the McIlwain furnace south of Irondale in Shades Valley; in December, 1871, Birmingham was founded; in 1872 coal was first barged down the Warrior River by Mudd, Hanby, Hewitt, and Steele; in February, 1876, the first pig iron was made at Oxmoor in experimental furnaces fired with coke fuel; it also was made with coke in Shades Valley, although the furnaces first used charcoal. Frank P. O’Brien built the first coke ovens at Oxmoor and Levine S. Goodrich was the superintendent of furnaces. The successful use of coke in making pig iron marked the beginning of great industrial activity in the Birmingham district.

The Tennessee Coal, Iron & Railroad Co. made its first iron in this district in 1881 and the first steel in 1888. The first furnace of the Woodward Iron & Steel Co. was blown in in 1883. The Sloss-Sheffield Steel Co. was organized in 1887 and the Republic Iron & Steel Co. in 1899.

The first iron ore used in furnaces of the Birmingham district was the soft ore, leached ore from the outcrop of the Big Seam on Red Mountain; the ore for the Oxmoor furnace came from No. 1 and McIlwain mines, Ishkoda. The flux was obtained in Shades Valley near the furnaces. Only the first 3 feet of the Big Seam ore was mined and the ore was obtained by stripping.

Ore was first mined underground in 1875 in slopes 1 and 2, Ishkoda; it went to the Oxmoor furnaces. All pig iron made in the district in the early days was hauled in wagons to Selma, where it was made into iron. During the Civil War cannon and other munitions were manufactured there.

**EARLY MINING METHODS**

Before the value of the red hematite ores of the Birmingham district was appreciated, the furnaces made charcoal iron from the brown ores. Later the possibility of using the soft red ores was
demonstrated and coke pig-iron was produced. In a relatively short time the soft ores were nearly exhausted, and it was generally supposed that when they were gone the ores remaining would be too limy for furnace use. This apprehension soon proved to be unfounded, when experiments demonstrated that hard ore could be used as a mix with soft ores; later work demonstrated that the hard ore was suitable for pig-iron manufacture with coke fuel.

Before hard ores were used the soft ores were mined extensively by open cuts, and it has been said aptly that "the outcrop workings on Red Mountain stand as a monument to cheap labor." The surface workings followed the outcrop of the Big Seam for 30 miles along the crest of Red Mountain, and extended down the dip of the ore bed for several hundred feet or until the cover grew so thick that moving it was too costly, even with the cheap labor of that day.

**HAND STRIPPING AND WAGON HAULAGE**

The ore bed was stripped by hand and the waste rock transferred to either side of the ridge in wagons, cuts or ramps being made through the ridge to permit the wagons to pass from the strippings to the spoils bank. Stripping was carried on until the cover attained a thickness of 28 to 30 feet, which was believed to be the economical limit. As the Hickory Nut seam occurs about 28 feet above the
Big Seam, it was commonly used as a basis for the calculation of permissible depth of cover; a perpendicular line drawn from the outcrop of the Hickory Nut seam to the Big Seam gave the limit to which stripping should be carried.

The ores of the Big Seam were sparingly worked at first, only 3 feet being removed; the depth was later increased to 6 and 8 feet, a larger amount seldom being taken. As the ore in the Big Seam averages 18 to 20 feet in thickness, it is evident that a large tonnage of ore still remains unmined in the old open cuts, although in certain localities some of this ore has been mined from below in later underground workings.

To mine the ore after it had been exposed by stripping was comparatively simple. The ore was blasted and quarried, and being weathered it separated readily along bedding and joint planes.

![Figure 2](image-url)

**Figure 2.**—Removing entire thickness of Big Seam by open cut

The larger masses were sledged to such a size that the larger pieces could be loaded by hand and the smaller ore shoveled into wagons or cars. The cost of mining and handling the stripped ore was slight; the drillers were paid $1.10 per 10-hour day, and the shovellers $1 for a 10-hour shift. The letting of contracts for mining thousands of tons of ore at 35 cents per ton is striking evidence of the cheapness of the ore used for pig iron in the early days of iron making in the district. (See figs. 1 and 2.)

**Haulage by Tram Lines**

As the use of red iron ore grew the demand for increased production required larger operations and mechanical equipment; consequently wagon haulage was soon replaced by cars running on
tracks systematically arranged to permit the handling of large outputs. As the tram lines that received the ore from the open cuts were largely on the Shades Valley side of Red Mountain, especially in the early days, the small ore cars used in the open cuts were connected with the trams through ramps similar to those that were used in wagon haulage of waste rock from the stripplings; in fact, the same ramps were often used. Probably the most common as well as the most satisfactory method was to extend a cut from the tram line on the side of the mountain to the bottom of the open cut in which the tram-line tracks were laid, thus permitting the ore from the open cuts to be dumped directly into the tramcars; the latter as loaded were run from the cut on to a sidetrack and empty cars were set for loading. To bring the small open-cut cars to the loading point of the tramcars, tracks were variously arranged in the open cuts. Gravity and engine planes, commonly used, extended from the dumping point up the dip or diagonally to it and received, the loaded ore cars from benches or terraces formed in the ore bed; the empty cars were returned to the benches as needed.

**DEVELOPMENT BY DRIFTS AND SLOPES**

When open-cut work was curtailed by the thickness of the overburden, the development of mines by drifts and slopes began, and numerous short slopes were driven with headings on either or both sides. For convenience the slopes were usually driven along joint or slip planes and did not follow the dip closely; the headings were driven at short intervals, usually 35 to 45 but not more than 50 feet. No attempt was made to rob the pillars, although upsets were made in them at 150 to 200 foot intervals along the stopes. Ventilation was simple because the slopes were short and openings to the surface through stopes and upsets were numerous. Drainage was also simple, as the lower workings could often be connected to the surface by drifts run from lower ground such as the hollows and runs that form gaps in the mountain.

Full advantage was taken of the lay of the ground in order that drifts might be used instead of slopes and the hoisting of ore and pumping of water be eliminated. In consequence the whole outcrop, including open-cut workings and ground where no surface work had been done, was worked out in a relatively short time. Some of the numerous openings thus formed aided and others hindered the satisfactory ventilation of the workings when the slopes were materially lengthened as underground mining advanced. Drift mining of soft ores was particularly applicable in those localities where numerous runs cut the flanks of the mountain and left narrow ridges between. This lay of ground, which is common on the Shades Val-
ley side of the mountain, offered an easy mode of attacking the ore bed and was convenient when the ore was delivered to tram lines extending along the runs. Here, too, as in the open-cut workings, gravity and engine planes supplemented the ore-handling system and were often the essential link joining the workings and tram lines. Many of the planes were several hundred to more than a thousand feet long and connected the drifts with the points of delivery to the trams.

MINING OF HARD ORES

Before the soft ores were exhausted the hard ores had been encountered and the problem of their utilization solved, at least to the extent that they were used as a mix with the soft ores; in fact, so rapidly were the hard ores developed that they soon became the more desirable; then the mining of soft ore declined, and attention was directed to the hard ores.

Hard ore was first mined in 1879 in the Alice mine, which was operated by the Alice Furnace Co. and later by the Morris Mining Co. Power drills and air compressors were first used at the Alice mine. Formerly all drilling was done by churn drills and hand labor, as power drills would not work satisfactorily in the soft ores.

The hard ore was mined in the slopes and drifts, where it had been encountered in mining the soft ore, consequently mining was still by drifts and the numerous slopes were continued. However, as workings underground were extended, economical operation demanded the elimination of many openings. As a result, a few of the old slopes were maintained and new main slopes developed at more advantageous points. The same process of elimination still continues, unification and standardization of practice and equipment being factors controlling the number of openings and choice of location.

At present there are in the district 24 slopes, most of which are active.

HAULAGE METHODS

Ore was hauled out of the mines in cars, four to six cars to a trip, and the empty cars were returned to the headings. In short slopes cars were easily handled, but in long slopes of varying grades there were frequent derailments, with consequent delays. Trip haulage was therefore abandoned and skip haulage installed in a number of the larger mines. Edwin Ball, general manager of the Tennessee Coal, Iron & Railroad Co., who introduced the use of skips in the iron mines of the district, used them first at the Spring-Gap mine. The second installation was at East Ishkoda No. 2 mine and the third at Raimund mine No. 1. The skips were of sheet iron and
each held six to eight cars of ore; they ran on tracks of 60-inch
gauge and were hoisted unbalanced.

At present there are in the district eight slopes using cars or trips;
the remaining 16 slopes are equipped for skip haulage. Additional
skip installations are planned, which indicates further standardiza-
tion of practice.

**MINING METHODS**

Mining methods were also modified as the workings grew larger. The
distances between headings (the heading intervals) were in-
creased to 65 and 85 feet, thus reducing development work and in-
creasing the amount of ore mined from the pillars. As ventilation
of the workings grew more difficult, blowers were used to deliver air

to the advanced slope and manway faces. As the slopes were ex-
tended, basins and faults in the ore bed required the installation of
auxiliary haulage systems for delivering ore to the slopes. Shaking
chutes and gravity planes have replaced human labor for handling
ore in the stopes, and the amount of ore handled has been greatly in-
creased and the costs reduced thereby. The whole trend in practice
has been to employ mechanical means for handling ore, thus making
possible more complete control of the output.

**RECENT PRACTICE**

During the last few years the changes in practice outlined above
have been still more marked, necessitating radical alterations in
development and mining methods. The production of the mines has
materially increased and the number of mines actively operating has decreased, indicating the adoption of labor-saving machinery and greater efficiency in working.

The heading intervals have been lengthened to 200 feet or more and the width of the stopes run with the headings has been reduced to 18 and 20 feet. In consequence these stopes have ceased to be any great source of ore supply and the bulk of the output now comes from stopes in the large pillars. Earlier methods of handling ore in the stopes, such as by chutes and planes, have been largely abandoned and mechanical loaders are being installed, which are working satisfactorily and are materially increasing the outputs, already large, although they may not have reached the highest degree of efficiency.

![Figure 4.—Surface works of large mine, Red Mountain](image)

No attempt has been made to improve the ventilation of the mines aside from the connecting of stopes to permit free passage of air currents caused by natural ventilation and the use of blowers and piping. Collapse of pillars in old workings and the robbing of pillars have caused extensive caving of top rock. This caving has tapped water-bearing formations and let large amounts of water into the mines, thus increasing the operating cost. The driving of intercepting ditches and the installation of larger pumping plants have been the only improvements that will tend to lower the cost of handling this water.

Railroads to connect the mines and the furnaces have been extended rapidly, and new and important improvements in the delivery of ore to the furnaces are under way. (See figs. 3 and 4.)
CHARACTER AND OCCURRENCE OF ORE

NUMBER AND THICKNESS OF BEDS

Red Mountain hematite, commonly called "red ore," occurs in the Clinton formation of the Silurian system. This formation is 200 to 300 feet thick, and is composed largely of shale and sandstone.2

Of the four ore beds in the Clinton formation only two are now supplying ore for the manufacture of iron and steel. These beds, in order of occurrence downward, are the Ida, Hickory Nut, Big Seam, and Irondale. The Big Seam is most important both as to grade of ore and tonnage available. The Irondale bed ranks next in importance, and although the output is not large it is of high enough grade to be considered ore, and as such has some importance in the production of pig iron. The Ida bed has been worked in a few localities, but in a very small way; in fact, the Ida and the Hickory Nut beds are not considered commercial producers of ore.

DEFINITION OF ORE

Under the present classification of iron ores, any ferruginous material that has a high enough iron content to permit its use in the manufacture of pig iron may be considered an ore. The Ida and Hickory Nut beds, therefore, do not contain ore, but as such a classification necessarily is tentative and depends largely on costs of production it is impossible to say how soon these ferruginous materials may become ores through improved methods of treatment and reduction in costs and thus assume relatively important roles in the production of pig iron in the Birmingham district. However, under present conditions the Big Seam and the Irondale are the only producers of ore and they are considered in the following pages.

BIG SEAM AND IRONDALE BED

The Big Seam and the Irondale bed outcrop on the west face of Red Mountain and dip southeast at about 20°. The ore beds strike northeast and southwest and maintain that direction with remarkable uniformity.

The Big Seam is the more important throughout a relatively large part of the district, but in the northern part the Big Seam degenerates into a low-grade ferruginous sandstone and the Irondale bed rises to first importance. To the southwest the Big Seam persists, though decidedly thinner, but the Irondale bed disappears.

---

The Big Seam is 15 to 22 feet thick, but as it is divided into two distinct parts—the upper and lower benches—by a layer of shale that increases from a mere separation in the northern part to 30 inches in the southern part of the workable area, it is not possible to name a reliable average thickness for the whole bed. Furthermore, in the southern part of the field the lower bench is broken up into a series of thin layers of ore, shale, and sandstone, low in iron and unworkable; the upper bench of the ore bed is then all that remains and is known as the Big Seam. The thickness of the upper bench averages about 10 feet, and of the lower bench less than 8 feet.

The Irondale bed is 3 to 8 feet thick but averages under 6 feet where it has been worked. It attains its maximum thickness and value in the northern part of the field, where the iron content is higher than that of the Big Seam.

The most important occurrence of ore is on East Red Mountain between Birmingham on the north and Bessemer on the south. Beyond these limits the workable deposits extend for some distance but are thinner and usually of low grade. Little information is available on the extent and thickness of the ore beds beyond the outcrops on Red Mountain; however, it is generally conceded that the beds vary somewhat in thickness and value both on the strike and dip. The Big Seam changes to sandstone to the north and to thinly bedded shale and sandstone to the south and is thinner, if not of poorer grade, to the southeast in the direction of the dip.

As this study of mining practice concerns the active part of the district, the workable ore beds may be said to lie within a semi-elliptical area whose major axis extends from Sadler Gap on the north to Sparks Gap on the south, and whose minor axis is approximately one-half the length of the major, being prolonged to the east beneath the coal measures.

KINDS AND CHARACTER OF ORE

Although other classifications of ore are based on minor characteristics, in so far as they affect the grade of the ore they are more apparent than real. For example, the ore does not materially differ in grade when in the fossil or the oölitic forms, probably because both kinds of ore occur in the same bed and are not confined to any particular locality, being more or less indiscriminately mixed throughout the vertical dimension of the Big Seam. 3

At present all the ores mined are hard, as the leached (soft) ores have been largely exhausted by mining. The occurrence of the soft ores is of interest, however, if for no other reason than that the leaching action made prominent by change in character of ores has also

largely affected the associated overlying formations, weakening them and complicating the drainage problem by making impervious beds porous.

The leached ores occur at the outcrop and extend down the dip of the beds for distances ranging from a few feet to a maximum of 400 to 500 feet. The average lower limit of leached ores is probably between 200 and 300 feet; the contact between the leached and unleached parts is exceedingly irregular, the greater depths following the line of stream beds that have cut to or nearly to the ore beds themselves. The terms "soft," "semihard," and "hard" ores, occasionally heard in the district, are misleading, as they indicate different degrees of leaching of the ores. Leached ore may be nearly or quite as hard as the hard ore, although much badly disintegrated and soft ore occurs along watercourses.

Soft ore may also occur in small quantities along and adjacent to fault planes some distance from the outcrop in ground disturbed by excessive folding, along joint planes or slips, and wherever the leaching effect of water has changed the character of the ore, and particularly the content of lime.

GRADE OF ORES

Entirely apart from the effect of leaching, the lime and silica content of the ore may vary widely throughout the thickness of the bed, because of the concentration of lime along planes of deposition that may be horizontal or cut across the bed at high angles, as along cross-bedded layers. High concentrations of lime may also occur at the junction of the ore bed with the top rock. However, the grade of the ore is remarkably constant for any given horizon, although occasionally a slight but persistent change is noted along the strike or the dip of the ore bed.

The grade of the ore in the upper and lower benches of the Big Seam varies greatly; in fact, if the classification applied to the Ida and Hickory Nut beds is applied here, the lower bench does not contain ore, although it is mined in large quantities and is used in the manufacture of pig iron. The ores of the lower bench are higher in silica than those of the upper bench, but as the margin between usable and nonusable ore is slight, there is hope for the utilization of the lower-bench ores in the near future if an economical method of treatment can be devised. Fortunately the upper bench of the Big Seam is much thicker and extends much farther laterally than the lower bench, the iron content varying but slightly throughout the actively operated part of the field. The average iron content of the ores now mined is 35.84, or approximately 36, per cent. The range in percentage of constituents is as follows: Metallic iron,
34.36 to 36.41; lime, 15.74 to 19.10; silica, 9.68 to 12.78; alumina, 2.75 to 3.38; phosphorus, 0.267 to 0.325; sulphur, none.

**VARIATION OF CONSTITUENTS THROUGHOUT DISTRICT**

In general, the content of metallic iron throughout the actively operated part of the district from north to south may be said to vary about 2 per cent. The lime and silica in the upper bench show wide but rather constant variations; the alumina remains remarkably constant throughout the district. The lime and silica graphs (see fig. 5) cross 6½ miles south of Red Mountain Gap, this being the approximate locality where the ores change from nonself-fluxing to self-fluxing; that is, where the lime content equals or exceeds the insoluble content, thus giving a balance between the two. (See fig. 5.) Probably more than one-fourth of the actively mined ores of the district are self-fluxing.

Systematic sampling in a number of the large mines shows that the ores of the upper bench vary per 100 feet on the dip as follows: Metallic iron, 0.055 per cent, decrease; calcium oxide, 0.050 per cent, increase; silica, 0.045 per cent, decrease; alumina, practically constant. The maximum distance along the line of the dip was approximately 6,000 feet; no samples were taken in the leached-ore zone. (See fig. 6.) Furthermore, a mining company reports that an analysis of ore representing the average for the upper bench of the Big Seam at mile 12, 14,000 feet from the outcrop, shows a content of metallic iron 1.88 per cent higher than ore at the outcrop, an increase in iron of 0.013 per cent per 100 feet. However, owing to the limited number of series of samples taken and the comparatively short distances covered on the dip the figures given merely suggest conditions, and it is not certain that such conditions persist throughout the area of workable ores.

Associated with the ore beds are numerous beds of ferruginous sandstone that differ greatly in thickness and in iron, lime, and silica content; however, in certain localities the iron content is surprisingly high, as determined by sampling several sections at the most promising points. The average metallic iron content for three locations, with sections 69, 37, and 34 feet thick, respectively, are 21.23, 22.97, and 32.30 per cent. Although large quantities of waste materials, principally shale, occur in these sections, they are usually in one part of the bed, permitting all or a large part of the worthless material to be eliminated at one operation if desirable.

It is evident, therefore, that the perfecting of methods of beneficiating these highly ferruginous materials may make possible their use at a profit, thereby adding greatly to the available reserves of ore.
PHYSICAL PROPERTIES

Ore samples taken from all operating mines for 21 miles along the outcrop, mile 7 being at Red Mountain Gap, were tested for porosity, true and apparent specific gravity, metallic iron, lime, and silica. The results are shown in Figure 5. The average porosity of the ore is 3.48; the average true and apparent specific gravities are 3.61 and 3.48. (See fig. 5.) Although the porosity of the ores varies somewhat from point to point along the 21 miles over which samples were taken, the graph indicates that the porosity diminishes from the
north to the south, which is natural, as the mines are deeper in the southern part of the field. This explanation may not be completely satisfactory, however, because the increase in depth is not as great as the change in porosity would indicate, but work with the diamond saw in preparing test cubes of the ores plainly showed that the ores to the extreme southeast were not only denser but much tougher than the ores farther north. Cuts made by the saw showed smooth, almost polished surfaces for the southeastern ores, while ores from the north were rough and granular. (See fig. 5.)

![Diagram](image_url)

**Figure 6.**—Variation in constituents of ore on dip of seam, upper bench, Big Seam, Red Mountain, mine A

The range in apparent specific gravity is fairly wide, but the average of 3.54, roughly 3.50, shows the normal condition. (See fig. 5.) The weight per cubic foot is 221 pounds plus, or 10 plus cubic feet to the long ton. These figures will be used in calculations relating to mining practice.

The values given for specific gravity of the ore were obtained from tests of the series of samples cited above; the results for the various places are shown in Figure 5. The apparent specific gravity was found by the ordinary submersion method, in which the samples were weighed first in air, then in water. The results are less accurate,
but meet working conditions better than the results obtained from calculations based on data obtained in the porosity tests. Figure 5 shows true and apparent specific gravities calculated from data obtained in the porosity tests; the averages were 3.61 and 3.48, respectively. Although the difference between the two sets of apparent specific gravities is only 0.06, it is more evident when the two graphs are compared, the former being much more uniform and regular than the latter.

The number of cubic feet of broken ore in a ton depends upon the size of the pieces and the character of the ore. Furthermore, if the ore is left in stock piles for some time or is shipped to distant points, thus permitting it to settle thoroughly, the percentage of void space will vary appreciably. The number of cubic feet of ore in a ton under the full range of conditions from freshly broken to settled ore will range from 17.2 to 20 cubic feet.

**OCCURRENCE OF ORE; STRUCTURAL IRREGULARITIES**

The grade of ore in the various beds and in the different parts of beds is the principal factor in determining whether the ore can be profitably mined, and the part of a bed to which mining must be limited. This fact is particularly true of the
Big Seam. The upper bench is extensively worked throughout the district, and the lower bench only to a limited extent. The following additional factors, however, affect mining practice more or less, depending upon their occurrence, singly or combined: 1. The dip and strike of the ore beds; 2, folds and faults, which cut and in many places displace the beds; 3, rolls and pots, which weaken the top rock and break the continuity of the ore beds; and 4, slips and cross-bedding, which break and weaken the ore bed and the top rock.

DIP OF ORE BEDS

The dip of the ore beds of Red Mountain varies widely both along the outcrop and at right angles to the strike, but the range of dip along the outcrop is fairly indicative of the conditions beyond the outcrop and in the mines. Figure 7 shows numerous determinations of the dip of the undisturbed footwall at the outcrop and the direction of the dip.

FOLDS AND FAULTS

The ore bed has been warped into anticlinal and synclinal folds. Secondary folding has also taken place, complicating the conditions that result from normal folding and faulting. Canoe-shaped basins occur, which vary in width and often terminate abruptly, thus seriously impeding the regular development of the mines. Not the least difficulty that results is the termination of main slopes, and the consequent need of establishing auxiliary slopes and haulageways. Wide variations in dip of the ore beds mark the occurrence of hills and basins produced by folding of the strata, which may in themselves seriously affect mining conditions.

The ore-bearing formations are cut by numerous faults that roughly parallel the outcrop, although a number of the larger faults cut across the mountain, causing gaps. Most of the fault planes dip eastward, but occasionally the dip is reversed. Most of the faults are normal; that is, the ore on the hanging-wall side of the fault is displaced downward; there are, however, a few thrust or reverse faults in which the displacement is downward on the footwall side of the fault. Displacements of ore beds range from a few feet to several hundred and much rock work is needed to continue the slopes and to change the grade of tracks. The latter interferes with the speed of hoisting more or less seriously. (See fig. 8.) Although a fault of large displacement seriously affects mine development it often does not affect operations as a whole as much as a series of faults of relatively small displacement, 10 to 20 feet. A number of approximately parallel faults along the line of a slope, but
normal to it, give a stepped arrangement to the parts of the beds lying between them and complicate the development of the mine, particularly the handling of ore. Under such conditions the switchback arrangement of mine track is common, but gravity and engine planes are also employed, making the economical handling of ore more difficult.

**ROLLS, POTS, AND KETTLES**

In many places rolls, pots, and kettles in the top formations reduce the thickness of the ore bed, but primarily they weaken the roof, as they break the regularity of the top rock. Rolls and pots in thinly bedded shales make mining conditions extremely bad, necessitating an increased number of props or, more commonly, causing a reduction in the width of stopes. (See fig. 9.) Similarly, potholes, which are water-formed cylindrical or conical openings filled with rock fragments (see fig. 10), break the continuity of the top formations and weaken the roof; however, when the pots are left intact in the workings and are mined around they cause no great inconvenience. The pots may serve well as supports in the workings. They interfere with regular working of the stopes, but this is not, however, a serious consideration.

**SLIPS AND CROSS-BEDDING**

Throughout the entire minable area of the Big Seam, joints or slips affect the development and especially the working of the mines.
In certain localities the slip planes, like incipient cracks cutting the ore bed, are hardly noticeable, while in other localities they may be numerous and prominent. A large number of observations along the outcrop, many of them checked by observations underground, give the results shown in Figure 11, which also indicate the relation between the direction of the slips and the
direction of dip. Although the direction of the slip planes on the surface differs somewhat from that in the workings, marked persistency in direction is usually observed, so much so that in opening many mines the slopes followed the slips. (See fig. 12.)

There are two series of slips practically at right angles; one series roughly parallels the strike and the other the dip of the ore beds. This feature is utilized in mining the ore; shots placed back of the slips force off slabs of ore that readily separate along the slip planes. Similar work on the lower face of arch pillars is called slabbing, as the ore comes off in slabs parallel to the slips. Slips in the ore, or lines of weakness parallel to the slips, make it comparatively easy to break up large masses in the stopes by sledging or by blockhoming.

Cross-bedding of top rock tends to shorten the span of the beams of rock in the roof; by cutting across the top rock at fairly high angles it greatly weakens the support that the top rock would otherwise afford.
Cross-bedding in the ore bed, usually indicated by concentrations of lime along the bedding planes, probably affects the compressive strength of the ore little, but like the slips the bedding planes offer lines of weakness and greatly facilitate the breaking of the ore into pieces of convenient size. (See fig. 13.)

**Figure 12.—** Slip planes cutting ore bed in open cut

**Figure 13.—** Cross-bedding showing in roof of slope

**CRUSHING AND COMPRESSIVE STRENGTH OF ORES**

Preliminary tests to determine ultimate or crushing strength show the ores will probably sustain with safety a static load of 12,000 pounds per square inch, but will fail under loads of 14,000 to 15,000 pounds. The effect of the slip planes is unknown, but their presence undoubtedly weakens the ore.

It is evident from the tests of compressive strength that the ore is strong enough to support a considerable thickness of overburden, if the cross section of the pillars regularly conforms to standard minimum dimensions, which would, of course, vary with conditions.
MINING METHODS

DEVELOPMENT OF MINES IN THE UPPER BENCH, BIG SEAM

Development and mining of the Red Mountain iron ore are in many respects relatively simple owing to its occurrence in beds of remarkably uniform thickness and dip. The usual methods employed will be discussed first, then the modifications necessitated by variations in number and thickness of the benches, their position with respect to dip and strike, and their relation to faults and folds.

The upper bench of the Big Seam—the workable part of the principal ore bed in the Birmingham district—outcrops without a break for 25 miles or more along the west slope of Red Mountain. Unless there is a specific statement to the contrary, descriptions of mining practice in this district apply to work in the upper bench. Methods of working the lower bench are described under “Working the lower-bench ores.” The ore was first mined by open cuts which were carried to the economical limit for such work, then underground mining had to be done. Because of the moderate dip and uniform occurrence of the ore bed development was by slopes.

Underground mining began about 1875 with a relatively small output. At present the annual production is more than 5,000,000 tons. Since 1910 the yearly output of red ore has been as follows:

<table>
<thead>
<tr>
<th>Year</th>
<th>Tons</th>
<th>Year</th>
<th>Tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>1910</td>
<td>3,110,000</td>
<td>1917</td>
<td>5,049,000</td>
</tr>
<tr>
<td>1911</td>
<td>2,983,000</td>
<td>1918</td>
<td>4,985,000</td>
</tr>
<tr>
<td>1912</td>
<td>3,814,000</td>
<td>1919</td>
<td>4,444,000</td>
</tr>
<tr>
<td>1913</td>
<td>4,370,000</td>
<td>1920</td>
<td>5,170,000</td>
</tr>
<tr>
<td>1914</td>
<td>5,003,000</td>
<td>1921</td>
<td>4,644,000</td>
</tr>
<tr>
<td>1915</td>
<td>4,974,000</td>
<td>1922</td>
<td>4,019,000</td>
</tr>
<tr>
<td>1916</td>
<td>5,672,000</td>
<td>1923</td>
<td>*5,500,000</td>
</tr>
</tbody>
</table>

* Estimated.

The development methods employed in the early days of mining are still in use, and the opinion is general that material changes are not likely to be made for many years. Extensive permanent improvements now under way corroborate this statement.

OBJECTIONS TO ABANDONMENT OF PRESENT MINING METHODS

Authorities justify the continuance of operations through slopes on the west face of Red Mountain by saying that abandonment of the present operating methods would involve:

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1. The loss of all equipment now in use in the Red Mountain mines; 2, the loss of work on transportation lines on the west side of Red Mountain; 3, the loss through abandonment of villages in Jones Valley near the present mine openings; 4, the expense of developing mines by vertical shafts and the installation of new and different types of equipment necessitated by the changes in developing and handling the ore; 5, the excessive cost of these shafts penetrating formations that are water-bearing and difficult to sink through; 6, the expense of transporting miners to and from the mines; 7, increased freight charges on ore and supplies; and 8, overcoming the prejudice of miners to working in mines where ingress or egress is by cage or skip only.

ARGUMENTS IN FAVOR OF CHANGE OF METHOD

These objections seem well taken on the whole and are conclusive to many persons; however, they can be overcome and are not wholly defensible when the future of the Birmingham district iron industry is considered.

CHANGES OF EQUIPMENT

In the first place it is not to be expected that sweeping changes will be made at one time. They will be made as equipment wears out or becomes obsolete and slopes require extensive repairs.

Faults, folds, and other irregularities in the ore bed complicate haulage in slopes and, coupled with the maintenance of long lines of haulage, make the operation of slopes and auxiliary planes cumbersome and expensive.

ABANDONMENT OF TRANSPORTATION SYSTEM

The abandonment of the present transportation system would not be necessary, but it could be linked up, if found desirable, with lines from the new mines, operating through tunnels connecting the valleys on both sides of the mountain.

ABANDONMENT OF MINING VILLAGES

The abandonment of mining villages is often a logical step in the betterment of operations, and removal of the population from ready access to a large city like Birmingham would probably greatly improve labor conditions. However, removal of the villages would not be necessary, as slopes abandoned for handling ore might readily be used for manways, or electrically operated trams to carry miners could be run from the villages through the mountain to the mines.
That villages can be maintained at mines in Shades Valley has been demonstrated.

**Sinking Shafts Through Water-Bearing Strata**

It is definitely known that extensive water-bearing strata overlie the ore bed throughout Shades Valley, particularly a mile or more from the outcrop. Methods of sinking shafts through water-bearing formations are well established, and although the work costs somewhat more than sinking through relatively dry formations it can be done safely and rapidly. Here again the extra cost is incidental to the extensive and economical development of a large ore body and would be very small when charged to a large tonnage distributed over a long period. The same argument applies to the installation of equipment that is new and different from that formerly used; for the reduced cost of operation and greatly increased output would warrant the change.

**Transportation of Miners**

If the transportation systems were connected with branch lines operating across or through the mountain, it is doubtful if the cost of transporting miners would be increased materially.

**Prejudice of Miners**

The prejudice of men against working in mines entered on cars or skips has been already overcome at a number of the large mines so equipped, where no serious objections have been raised. However, there is a well-founded prejudice against working in mines that have only one opening, with ingress and egress by cars or skips alone.

**Development by Vertical Shafts**

Many slopes on Red Mountain have reached or are approaching the limit of economical and efficient operation and a number of main slopes terminated when folds were met that would necessitate installing auxiliary hoisting and haulage systems. Such systems would undoubtedly be necessary if vertical shafts should be employed, because folds and faults are known to exist east and south of Red Mountain, but if motors were used for collecting from the auxiliary systems and delivering to the foot of vertical shafts ore could be handled more efficiently than through slopes of restricted capacity. Moreover, if systematic exploratory drilling preceded the location of mine openings, a large area could probably be worked to best advantage through vertical shafts. (See the section on
"Suggested changes in mining practice.") On the Rand, in the Transvaal, the "reef" dips steeply at outcrops, but flattens to 30° at depth, and was worked by inclined shafts or steep slopes in the earlier mines, but farther down the dip the ore is developed and mined to advantage through vertical shafts at the "deep" and "deep deeps" mines.

DEVELOPMENT BY SLOPES

All but two mines are developed by slopes. In a third mine a vertical shaft is connected with the slope to facilitate the handling of ore and supplies; the part of the slope that lies between the foot of the vertical shaft and the surface has been abandoned.

Formerly all development work was done in the ore bed, but as methods of handling ore were improved a change was necessary, and part of the development work was done in the formations underlying the ore.

As only the upper bench of the Big Seam is worked in most of the mines the development methods used therein with trip haulage—that is, by cars—will be considered first; skip haulage, the latest and most efficient method now in use, will then be discussed.

DEVELOPMENT FOR TRIP OR CAR HAULAGE

As stated before, variations in dip and the presence of faults control development, often changing what would otherwise be a simple and symmetrical plan to one exceedingly complex and difficult. When modern equipment is installed and large tonnages handled, the direction of joint planes or slips and minor disturbances are ignored in the general plan of development.

In the early development, when slopes were seldom more than 3,000 feet long, the slopes were usually driven parallel with the major slips for ease of driving; if the mine were developed beyond the preliminary stage, however, these slopes were almost invariably abandoned and others were driven on the dip, so that workings were developed on both sides of the slopes with equal ease.

SLOPES

Slopes are driven on the ore bed, beginning with the outcrop if possible, and spaced more or less regularly along it. The distance between slopes in the various groups of mines along the outcrop on Red Mountain is 1,333 feet, 1,480 feet, 2,350 feet, and 2,475 feet, or an average of 1,909 feet.
A number of slope portals have been moved from the outcrop into the stronger underlying formations some distance below the outcrop, in order to insure permanence by avoiding disintegrated and weakened ground, which ultimately caves. Driving the upper part of a slope in the footwall reduces the grade of that section of the slope and makes hoisting at uniform speed difficult, and is only a minor cause among several that occasion changes in the grade of slope tracks.

When the situation of slope portals and the distance between them have been determined, the slopes are driven on the dip of the ore bed. Uniform dips permit regularity in the direction and grade of slopes and give maximum advantage in handling output. Folds, cross folds, and faults necessitate radical changes in the driving of slopes, with resulting irregularities in grade.

A broad synclinal basin requires the main haulageways to follow the dip and rise of the ore bed across the basin, but a narrow basin is often “bridged” by continuing the slope at about the original grade through the formations overlying the ore.

Faults of small displacement do not affect the grade of the slopes materially unless the dip changes decidedly below and beyond the fault.

Faults of large displacement necessitate radical changes that often involve much rockwork and long stretches of track of steep grade.

Slopes are usually driven 14 to 16 feet wide and the full height of the ore bed, except where the top rock is weak; in such workings the slopes are 7 to 8 feet high, with 2 to 3 feet of ore left as support.

**MANWAYS**

Passages called manways parallel the slopes and are driven on either side of them 12 to 15 feet wide, and the thickness of the workable ore bed high. The manways are spaced 75 feet from the slopes, leaving pillars for protection of the main haulageways. The mine crews pass through the manways on foot, except in the larger mines where the miners are hauled in cars or skips. Many manways are used for mule travel and serve to a limited extent as ventilation passages.

Where mules do not travel the manways the latter are easily kept in good condition. Manways often have roughly formed steps, consisting of squared timbers about 4 by 6 inches in section and 6 feet long, spaced 18 to 20 inches center to center, and fastened together and kept from slipping by boards or planks nailed to the ends. The irregular spaces between the timbers and the floor are filled with fine ore or waste, which when well packed forms with the timbers
a good footing, the timber edges maintaining the stair form. (See fig. 60, p. 79.) Round or half-round timbers which are also spaced and held the desired distance apart by narrow strips of board or plank may also be used. On slight inclinations any regularly spaced, rigid member will give a toe hold and prevent a man from slipping on a footwall, which is often wet and slippery with clay. On steeper inclinations the more substantial construction mentioned above is preferable, but in many places is replaced by the barricaded construction.

On steep dips are driven manways 16 to 18 feet wide, in which a series of barricades is built of props and timbers erected on the footwall to a height of 3 to 4 feet. Above these barricades waste rock and ore are filled in, the whole making a continuous but zigzag passage from heading to heading at a moderate grade. The men traveling such a passage are protected from falling and rolling ore spilled from cars in the headings. (See fig. 14.)

Hoisting slopes and manways may also be protected by concrete supports, and mortar walls have been built along the sides of a few slopes.

**HEADINGS**

From the slopes—the main connections between the underground workings and the surface—other passageways called headings are driven into the ore bed on either side and are extended several hundred feet laterally at a slight grade.

Headings divide the ore bed into panels or lifts and in these the stopping of ore is done. The distance of 65 feet between headings probably represents the most common practice at present, and the intervals of 80, 100, 120 feet, and upward indicate a marked change in method of working. Ultimately intervals of about 200 feet will
probably be standard. The adoption of mechanical loading devices, such as scrapers, is largely responsible for the change indicated by the greater intervals between headings.

The headings are driven from the slopes to the manways; they have a width of 15 feet and a height equal to the full thickness of the ore bed. Through them the ore is delivered to the stopes. Beyond 75 to 100 feet from the slopes the face of the heading is widened to 30 or 35 feet and the identity of the heading as such is lost, both heading and stope face advancing as one and in a single operation.

**PLAN OF DEVELOPMENT**

Figure 15 shows the general development of a mine, including slopes, manways, and headings. Any mine map clearly shows the changes in strike and dip of the ore bed by the change in direction of the headings and stopes, since a grade of 2.5 to 4 per cent must be maintained in favor of the loaded cars to allow them to run to the slopes by gravity.

The development of a mine is best done symmetrically—that is, on both sides of a slope—in order that mining may progress both
horizontally and on the dip with uniformity. If, however, proper protection is to be given to the working force, the extension of a slope and manways can not be symmetrical, consequently manways are generally driven in advance of the slope. One manway is usually carried somewhat ahead of the other manway, which in turn is often one or two headings in advance of the slope. Again, one manway may be driven in advance of the slope, which is at least one heading in advance of the second manway.

Regardless of the method employed, one manway forms the pilot passage, establishes the direction of the development work, reveals changes in dip of bed or the presence of faults, and provides ventilation as soon as connection can be made with the slope through the connecting headings. A not uncommon practice is to carry the manways several hundred feet in advance of the slopes and drive headings from the manways and work the headings before connection with the slope is made.

**PROTECTION OF WORKERS**

In order that the working force may be protected and that the material excavated may be handled to the best advantage, the slopes are raised instead of sunk—that is, they are driven up the dip. Each successive advance of a slope is made by driving upward from the next heading below, cross-cut from the manway carried in advance. (See fig. 16.) Where both manways are carried in advance of a slope the arrangement does not differ materially, except that the ground ahead is explored to both sides of the slope, and as a further advantage ventilation is established ahead of the slope, thus insuring better working conditions when connection is made between the slope and manways. (See fig. 17.)
When slopes are driven up the dip instead of down it, the miners are in no danger of being hit by ore or rock falling down the slope or by runaway cars or skips, but are protected by an unmined block of ore or "pentice" between the end of the slope and the uncompleted part below. (See figs. 16 and 17.)

**WIDTH AND ARRANGEMENT OF SLOPES AND HEADINGS**

Slopes are usually driven 14 to 16 feet wide, but are often 18 and 20 feet wide at the headings to give room for connecting the heading tracks with the slope tracks. Similarly, in the headings extra width is given where sidings, usually extending to the manways and often to the slope, are provided for loaded and empty cars.

Where haulage in slopes is by car or trip, the headings may be staggered along the slopes. This arrangement is advisable to lessen the danger of derailments at the heading switches.

The numerals designating the headings are usually similar for both sides and increase as the slopes are driven inward. The side of a slope is designated by R or L for right or left; for example, 24R indicates the twenty-fourth heading on the right side. However, it is not unusual to have even numbers on one side and odd numbers on the other, the numerals on the last headings indicating the total number to a slope.

**CHANGE IN METHOD OF DEVELOPMENT FOR SKIP HAULAGE**

When haulage on the slopes is by car or trip, slope and heading tracks are both on the footwall or bottom rock of the ore bed, and
the cars run directly from the headings to the slope tracks. This arrangement may seem simple and ideal, eliminating, as it does, the additional handling of the ore in transit from the stope to the surface. However, for efficient and economical operation the use of cars in the stopes and of skips on the slopes is much to be preferred. Undue wear of cars, rope, and track, damage of equipment, and injury of workmen indicate the wisdom of speedy and universal change to skip haulage.

Most Red Mountain mines now use skip haulage, and have changed development methods accordingly. The principal changes in development that have been necessitated by the use of skips are: The slopes are driven in the formation underlying the part of the ore bed worked; and headings are turned off at equal distances and opposite each other along the slopes. In new work the slopes, headings, and manways are driven practically the same as for car or trip haulage, except as the installation of new equipment necessitates minor changes to meet new conditions. The changes include (1) taking up the bottoms of the slopes to such a depth as to permit cars in the headings dumping into the skips and (2) establishing loading stations along the slopes, one for each pair of opposite headings, thus reducing the amount of development required and facilitating the handling of ore.

The usual cross-section of slopes intended for skip haulage is 10 by 18 or 20 feet. Where the headings cross the slopes, stations are cut with an over-all height of opening of about 20 feet, which is the height of the slope plus the thickness of the ore bed worked.

DEVELOPMENT BY INCLINED AND VERTICAL SHAFTS

To mine ore at considerable distance from the outcrop of the bed, vertical or inclined shafts must be employed. Only three of these, two vertical and one inclined, have been made in the district, consequently no well-defined practice has yet been established. The use of slopes for main development openings is so thoroughly established and the methods of handling ore in them so well developed that to date little thought has been given to the use of shafts.

The two vertical shafts were sunk by the Woodward Iron Co. The deeper of the two (No. 4) is about 8,000 feet southeast of the outcrop of the Big Seam at the Muscoda division of the mines on Red Mountain. The shaft is 13 by 22 feet and is 1,400 feet deep. The second vertical shaft is at the Songo mine, 1,000 feet from the outcrop and about 384 feet deep. This shaft has a cross section of 10 feet 4 inches by 7 feet. It was sunk to eliminate the old slope, which was becoming difficult to maintain, and to facilitate
the handling of the ore and its delivery at a point accessible by railroad.

Practically no development work or mining has been done in the deep shaft of the Woodward Iron Co., consequently no definite system of development has been adopted. The Songo mine is developed like most car or trip haulage mines.

**Plans for Shannon Slope**

The only inclined shaft sunk and in use in the district is the Shannon slope of the Gulf States Steel Co., which is about 14,000 feet southeast of the portal of No. 7 mine of the Tennessee Coal, Iron & Railroad Co. (See fig. 18.) The ore at this point is about 1,900 feet below the surface, but the shaft, which has an inclination of $52^\circ 15'$, is 2,600 feet long. The shaft has a cross section of 13 feet 6 inches by 26 feet 6 inches; it is lined with concrete and is 9 by 18 feet in the clear.

The low dip of the ore bed and the need for large pillars have necessitated changes in the plan of development, but in the future slopes will be driven on or approximately on the dip from the main haulageway, which passes through the foot of the main slope. All development will be confined to the lower bench of the Big Seam. The slopes will be spaced about 500 feet apart along the strike and will be protected by pillars 75 feet wide on each side. Headings driven...
from the slopes at 100-foot intervals will be 9 by 9 feet in section until they pass the slope pillars.

Beyond the slope pillars the width of the headings will be increased to 15 feet. The rooms turned from the headings will have necks 9 feet wide and 20 feet long; they will be spaced 60 feet apart and driven 20 feet wide at an angle of 45° with the headings. They will break through into the next heading above and the connecting passage will be 9 feet wide and 10 feet long. By this arrangement of development, passages of moderate grades for handling ore will be insured and full advantage taken of conditions favorable to the maintenance of workings.

Although the system described is being followed as closely as possible, modifications will undoubtedly have to be made, as in the past, but conditions seem to favor its ultimate success.

**METHODS OF DRIVING SLOPES, MANWAYS, AND HEADINGS**

**DEFINITIONS**

The driving of slopes, headings, and manways constitutes development, whereas stoping and forming upsets and breakthroughs may be considered mining. As headings and stopes are driven in one operation except where the ore occurs under unusual conditions, it is difficult to separate development and mining. However, as it is becoming the practice to increase the distance between headings, and consequently to increase the amount of ore removed by stoping beyond the usual amount obtained by stoping in conjunction with the headings, the driving of stopes will be discussed in connection with the driving of headings.

**DRIVING SLOPES**

Slopes 8 feet high and 16 feet wide will be taken as an average size, although in practice the dimensions differ considerably because of differences in methods of mining. Two series of nine holes
each, arranged in sets of three, are drilled in the face in more or less vertical lines, inclosing a wedge-shaped mass of rock between the prolonged sides of the slope and the holes. (See fig. 19.) For the most efficient work, advantage is taken of the condition of the face of the slope, consequently the arrangement of the holes is often rather irregular and unsymmetrical. From 9 to 12 shots are employed per round, or 18 to 24 for each advance, as drilling is done in two shifts, first on one side and then on the other.

Holes are drilled both upward and downward to insure the maintenance of the full section of the slope and are 4 to 10 and 12 feet long. The drills, mostly of the water type, are mounted on tripods. The footage drilled per machine shift ranges from 60 to 80 feet.

The ore broken down at the end of each shift is cleared away by the next shift and that part of the face is thus kept free for drilling. Mucking and drilling are often done simultaneously when they are continuous operations. An average of 45 to 50 tons of ore is broken for each 4 feet of advance of slope. In a 10 by 20 foot slope about 80 tons of ore is broken by 25 to 30 holes of an average length of 7.5 feet.

Disposal of the ore and rock broken during the driving of slopes is discussed under “Handling ore in slopes and manways.” Slopes are usually driven to the rise and in consequence clearing the face of broken ore and rock is materially facilitated.

**DRIVING MANWAYS**

Manways are driven much like slopes, and in fact contracts for driving are often made on the same basis of costs. The two operations differ mainly in dimensions, in direction with respect to dip, and occasionally in alinement.

When the dimensions of the manways are similar to those of slopes the method of driving may be the same, but the usual practice and arrangement of holes shown in Figure 20 are probably used more
often when conditions are normal. Holes are arranged as in Figure 21, when the ore bed is relatively thin and the ore is more readily broken, as where slips are present. In the first arrangement 15 holes, and in the latter 10 are used. The amount of ore broken for each advance of 4 feet ranges between 40 and 45 tons.

**DRIVING HEADINGS**

Only rarely are headings driven separate and distinct from the stopes, but when driven separately they are like drifts or crosscuts passing through barren or unworkable ground and are small in section. However, the first 75 to 100 feet of headings driven from the slopes through the slope pillars is narrower than the combined headings and stopes, averaging 15 feet, but is often much wider where two sets of tracks are used to handle loaded and empty cars. As all heading work is done on the same basis of cost, the narrow work of driving headings next to the slopes need not be discussed. Holes used in driving headings are arranged like those in manways.

Headings and stopes are usually as high as the workable bed of ore, although roof ore is sometimes left to support the top rock or draw slate. Roof ore may be and often is mined later, but such work is not part of the original stoping operation. Stopes are 9 to 14 feet high, averaging about 10 feet for all the mines, and from 25 to 45 feet wide; a width of 35 feet is most common. The combined heading and stoping face is therefore 10 feet high and 35 feet wide and is carried forward in one operation, the method of attack varying mainly with dip of the ore bed.

The holes for driving headings are usually arranged as in Figures 22, 23, and 24; but even more than in driving slopes and manways advantage is taken of the condition of the working face, that is, the shape of the face and the number and direction of slip planes, especially the latter. Two holes are placed in a more or less vertical line and spaced 2 feet and 4 to 5 feet respectively from the bottom of the
stope, the lower sloping downward and the upper upward; this arrangement constitutes a unit in the system of blasting followed. Three or four such units are usually enough to break down the face for the full width and advance the stope 3 to 4 feet. Miners can always choose between a larger number of holes or longer holes; the latter are favored, as fewer set-ups are required and thus time is saved. Two eight-hour shifts are required for completing a round of holes and consequently one full advance of the heading and stope face, but to maintain a continuous supply of ore from each working face shooting is done at the end of each shift. From 50 to 65 tons of ore are broken down per shift or 120 to 130 tons for a full advance of face to a depth of 4 feet. Wider stopes will, of course, produce a tonnage correspondingly larger.

Other details of development are discussed under "Drilling and blasting."

**EXTRACTION OF ORE**

**DRIVING UPSETS**

A mine developed for hand shoveling has headings at 65-foot intervals. As the combined headings and stopes are 35 feet wide, an arch pillar 30 feet wide is left between
stopes. At 150 to 200 foot intervals along the stipes, upsets 25 feet wide are driven in the lower face of the arch pillar, breaking through or nearly through the pillar. When these openings do break through the pillar they are called “breakthroughs.” The shape is irregular, narrowing with advancement, until the original width has been reduced about one-half when the openings break through into the headings above.

Primarily breakthroughs provide outlets for air currents, through which dust and fumes from explosives may be promptly removed from the working places; they also permit men to pass from heading to heading. Where it is necessary or desirable to run air and water lines from one heading to another, breakthroughs are of decided advantage. The driving of upsets allows the removal of as much ore as possible in preliminary mining operations. The two principal operations in mining are therefore stoping and driving upsets. (See fig. 25.)

As most ore mines in the Birmingham district follow this procedure, it may be considered standard. A larger tonnage of ore, however, is probably produced by methods adopted more recently.

Upsets may be driven by one and sometimes two crews of drillers, one crew being more common. Figure 26 shows the arrangement of holes, which is similar to that used in stoping. The amount of ore broken by a complete round of holes averages 90 to 110 tons for each 4 feet of advance. Upsets are driven much like slopes, particularly as both types of openings are driven up the dip, but differ in that the full height of the workable ore bed is always removed. Upsets are, however, similar in practically all respects to raise stipes.
STOPING PRACTICE

Stoping practice has already been described, in a general way, without reference to the dip of the ore bed. Although stoping is more or less independent of handling ore, yet it is necessarily of prime importance that the ore when broken down should be readily accessible to the cars into which it is to be loaded. Full advantage must therefore be taken of all conditions that will facilitate the handling of ore, foremost among these being the dip of the ore bed.

When dips are high no special arrangement of holes is necessary to insure prompt delivery of ore to the track level, unless the dip is as high as 25 to 30°, when retarding the downward movement of ore may be desirable. Breaking ore from above downward may prove most satisfactory, as the steps formed in the stope face hold back the ore somewhat. Underhand stoping should be employed with high dips.

When dips are as low as 5 to 10°, working from below upward permits the ore to fall unimpeded from the stope face to the track level; gravity and the force of the explosive place the bulk of the broken ore within easy reach of the shovelers loading into cars. Overhand stoping is therefore best adapted to low dips.

The arrangement of holes for overhand and underhand stoping is shown in Figures 22 and 24; holes in underhand stoping are largely drilled downward or are wet holes, while practically all those in
overhand stoping are dry. The figures also show the vertical arrangement of holes, the two holes of a set being directed toward the top and bottom of the bed to exert uniform pressure on the whole face in shooting.

Bedding and slip planes help to determine the arrangement of holes in the stope face, and full advantage is taken of them to effect a material saving in drilling and explosives.

**CHANGES IN PRACTICE**

Mining practice has changed greatly during the past 10 years, particularly with respect to increased width of stopes and methods of attacking the ore. The distance between headings has been increased from 55 to 200 feet and the tendency is to increase the interval still more. Headings and preliminary stopes have not been widened appreciably but remain at or close to 35 feet wide, although, with the 200-foot interval, 20 to 25 foot headings are now used. The increased interval between headings has evidently resulted in much larger pillars, from which the larger part of the ore mined is obtained.

Increasing the size of stopes requires more attention to the support of the top rock, which is particularly true in areas where the top rock makes a bad roof. It is recognized that there is a limit to the size of the area that can be worked out, even with such temporary support as can be provided, and it has been found that methods will have to be modified to meet conditions. Changes in development and mining methods have followed changes in ore handling, particularly in stopes and in hoisting ore to the surface; the cost of mining and handling the ore has been the controlling factor in these changes.

**BRANCH TRACKS**

In contradistinction to pillar robbing, the driving of wide stopes was secondary to the original mining; such stopes were opened from the preliminary stopes and driven directly up the dip of the ore bed. (See fig. 27.)

With heading intervals of 70 to 75 feet and headings 35 feet wide, the 35 to 40 foot pillars were cut into blocks 25 to 35 feet long by upsets or raise stopes 25 to 30 feet wide. The upsets or raise stopes were connected with the headings above by 8 to 10 foot break-throughs. The ore broken in the stopes was loaded into cars that ran on tracks extending from the headings to the stope faces. Usually the pillars in adjacent stopes were offset and did not form continuous lines throughout the mine.

In later practice the stopes were driven 75 to 100 feet wide, spaced 75 to 100 feet apart, and carried to the heading above. At first the
heading intervals were 75 to 80 feet and the pillars in the adjacent headings were offset or staggered to give maximum protection. (See fig. 28.) A wide face of ore in the stopes enabled full advantage to be taken of all conditions favorable to drilling and blasting, but in order that the ore might be handled efficiently tracks were laid from the main headings to the raise stopes. In some places where the dip was low, gravity planes were employed, but the small width between pillars hardly warranted the establishment of such a system. With high and moderately high dips, and with better con-

**Figure 27.—Use of branch tracks in driving wide upsets on low dips**

**Figure 28.—Use of branch tracks in working wide stopes**

dition of top rock, the stopes were driven in line for several adjacent stopes, and the ore was drawn from the stopes to their respective headings or even drawn through two or more stopes to a heading track which served as a main haulageway to the slope.
Gravity Planes

After large stopes had been worked by this method more or less successfully, it was extended to heading intervals of 160 feet, giving pillars 125 feet wide, and to fairly high dips. The preliminary raise stopes are driven 25 to 30 feet wide and entirely through the pillars to the headings above. The ore is handled by gravity planes following the full dip of the ore bed. When the preliminary raise stopes are completed a new stope is started at the bottom of the pillar and carried upward. Successive strips or slices of pillar parallel with the dip are thus removed until the stope is increased to a width of 130 to 150 feet, or to such a point that further increase in the size of the stope threatens to weaken the roof unduly. (See fig. 29.)

![Diagram of gravity planes in working large stopes](image)

Numerous props are employed in the stopes. These props are placed close together and in rows along the dip, regularity in timbering being required by the installation of gravity planes. (See fig. 30.)

After the preliminary raise stopes are driven the subsequent work in the stopes is rendered relatively much easier by the ore being free on two sides. Holes can be drilled to advantage in the sides instead of the face of the stopes; the set-up of the drills is practically unimpeded by broken ore, and the direction of the slip planes also favors the work.

Scraper Loaders

The use of scrapers and other mechanical loading devices has allowed much more extended application of the wide-stope method. A relatively long movement on the dip of the ore bed and conse-

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sequently a long heading interval are required for the successful use of scraper loaders. A fairly wide stope is needed for economical operation. The lower limit of stope interval for scraper loaders is 150 feet; the upper limit has not been definitely determined, but will vary largely with local changes in the character of roof. The dimen-

![Image of a mine heading]

sion of the stopes commonly spoken of as length, as it parallels the preliminary stopes, has necessarily been increased to correspond with the increased width of the stopes. The greater the output from a given stope the lower the cost per ton, consequently the tendency is to make the stope larger in both directions. This condition of
working is well understood, and every effort is being made to determine the safe working limits as influenced by condition of overlying formations, occurrence of ore, and dip of ore bed. Although scrapers reduce the cost of driving headings they introduce in connection with support an element of danger that previously was slight. This danger can be obviated to some extent by the judicious use of pillars left in the stopes, but the presence of pillars in stopes where scrapers operate is a decided disadvantage.

Although scraper loaders have been used in the mines for a number of years, their development has only recently reached a stage that assures their success. During their experimental stage scrapers were used in relatively few localities, but now that permanence of operation is assured they are being introduced in a larger number of mines and under much wider variations in operating conditions.

Although there is marked similarity in method of operation of scraper loaders in the different mines, the differences are important enough to note here.

**Driving Headings with Scraper Loaders**

The simplest use of the scrapers is where headings are driven 18 to 20 feet wide and 150 feet apart. At 75-foot intervals stopes are driven in the pillars; they are 20 feet wide for 20 feet and then are widened to 50 feet, the ribs extending up the dip at an angle of approximately 60°. When the full width of the stope has been developed, it is carried up the dip by overhand stoping, the stope face maintaining, as nearly as possible and practicable, a line parallel with the headings; it ultimately breaks through the pillars throughout practically the full length of the stope. As the length of these stopes is much less than the breadth, it might be better where the heading intervals are longer to consider the longer dimension the length, then the stopes break through the pillar along its full width.

In Figure 31 the broken line \(ab\) indicates the working face of the stope. The stubbing bar of the drag line is set in this face and is moved along it as the operation of the scraper shifts from one side to the other of the stope. When the stope breaks through into the heading above, the stubbing bar is shifted to the roof without materially affecting the operation of the scraper.

The width of the working face enables full advantage to be taken of slip planes and other conditions that affect mining and the ore can be mined at relatively low cost. Scrapers work to best advantage on dips of 15 to 20°.

With scraper loaders being used under top formations that differ in strength it is necessary to have the method of mining more or
less elastic in order that the size of the stopes may not exceed the safe working limit with respect to support. Of the two dimensions, the length as determined by the heading interval is fixed for any given place, but the width or the horizontal dimension can be varied to suit the conditions developed by mining.

DEVELOPMENT OF STOPES WITH SCRAPER LOADERS

The stopes as usually developed are large for iron mines, about 50 to 60 feet wide by 180 to 150 feet long and 100 feet wide by 150 to 200 feet long.

![Diagram of stopes for scraper loaders](image)

**Figure 31.**—Arrangement of stopes for scraper loaders

Failure of top formations may be obviated, at least temporarily, by leaving pillars in the stopes. Pillars provide better support than props but are objectionable because they must be worked around in both mining and scraping out the ore. The use of scraper loaders will probably bring about other and possibly more radical changes in mining methods, requiring longer stope intervals and possibly much wider stopes; but such developments will depend on better and more extended knowledge of the conditions affecting support and how existing conditions can be utilized in the safe working of the mines.
Figure 31, left, shows a simple stope with only one opening or way; stopes with two or more ways are similar in virtually all respects except width.

Figure 31, broken lines, also shows a multiple-way stope. In such a stope the heading interval is 175 to 200 feet. The headings are 20 to 25 feet wide and provide room for three standard-gauge tracks. At given intervals along the headings, depending upon the number of openings to a stope, two or more 20-foot upsets spaced 35 feet apart are driven in the pillars. Pillars 20 to 25 feet wide are left between lifts or panels. After the upsets are driven 20 feet they are gradually expanded until they connect to form the stope. The stope face, ultimately the full width of the stope, is carried upward until only 20 or 25 feet of pillar remains between the stope and the heading above. Upsets, usually similar in number, size, and arrangement to those at the bottom of the stope, are driven through the intervening pillar.

As mentioned before pillars are occasionally left in the stopes to insure adequate support of the top rock. The inconvenience caused by these pillars will probably make smaller stopes preferable.

**WORKING THE LOWER-BENCH ORES**

The lower bench of the Big Seam has been and is still being worked extensively at a few places on Red Mountain. In localities that have been mined most successfully the condition of the top rock is good, permitting large stopes to be worked with few or no props. As a result of the mining of the lower bench, stopes 30 to 35 feet wide, 20 feet high, and hundreds of feet long are formed with practically no other support than small pillars, remnants of the arch pillars, along the upper rib. Many such stopes have stood for years with slight indication of collapse, a fact which speaks well for the strength and resistance of the top rock.

**EARLY MINING METHODS**

The first attempt to mine lower-bench ore, where minable, was entirely inadequate and unsatisfactory, and is mentioned in this connection only to show how an unsuccessful attempt has been developed into a well-systematized and successful method.

When lower-bench ore was first mined, branch tracks were turned into the stopes of the upper-bench workings and extended along the middle of the stopes. In order to facilitate the handling of ore across the full width of the stopes the bottoms of the new stopes were carried horizontally, and working faces therefore varied from no height to several feet next to the foot of the arch pillar. As this
irregularity was unsatisfactory, the height of the ore bench worked was increased by increasing the grade of the track as the work advanced. This procedure was also unsatisfactory, as the grades tended to become excessive as stops grew steeper, consequently only short sections of the old stope bottoms were worked. (See fig. 32.)

PRESENT METHODS

After this attempt at mining the lower bench, the present method was developed, has been successfully employed for a number of years, and has produced large tonnages of ore at very moderate cost.

![Diagram of mining method](image)

**Figure 32.**—Early method of working lower-bench ore

The mining of lower-bench ore or upper-bench stope bottoms in any particular stope begins in the stope immediately below, from which a cross heading is driven to the stope above. A branch track is turned off the main-heading track, and its grade is maintained as determined before and as required by the dip of the ore; it extends diagonally across the stope through the cross heading in the arch pillar to the bottom or middle of the stope above, where it is run parallel with the main-heading track. (See fig. 33.) A stope face worked thus is practically the full height of the lower bench, except where the ore bed dips steeply and the track is in the middle of the stope, when a wedge-shaped mass of ore will remain unmined at the bottom of the stope, just below the former heading. No ore
is left unmined when the cross-heading track is at the bottom of the new stope and the stope has a bottom similar in all respects to that of the original stope above. These stopes may be carried the full length of the stopes worked in the upper bench.

Lower-bench ore is mined by underhand stoping at slight cost, as the benches worked are wide and usually 8 to 10 feet high. The ore

is broken down by holes placed on the bench above or in the vertical face of the bench, if shooting off that face is preferred. Six holes in three rows of two each are usually employed; the holes are 10 to 12 feet long and advance the face 8 feet per round of shots. About 160 tons of ore is broken for each round of shots in a stope 30 feet wide by 8 feet high. One eight-hour shift drills the six holes, but two shifts are required to muck the ore.
The removal of pillars in connections with the mining of the lower bench is discussed under "Robbing pillars." Where conditions are favorable—that is, where both benches of the Big Seam are mined and the pillars robbed—the extraction of ore approaches 95 per cent;

but all conditions are rarely favorable for such high extraction; in fact, at only a few places can 80 per cent of the ore be mined from the upper bench, for the character and condition of the top formations prevent.
MINING METHODS

WORKING THE UPPER AND LOWER BENCHES TOGETHER

Working the entire thickness of the Big Seam by one continuous operation is a development in recent mining practice that is especially interesting, now that attempts are being made to render available by beneficiation the low-grade high-silica ores not utilized at present.

MINING AT SHANNON SLOPE

Of several localities in the district where lower-bench ore of the Big Seam is of high enough grade to be suitable for the furnaces, there is one locality where the full thickness of the ore bed is now being mined.

The ore bed opened and developed by the Shannon inclined slope has been mentioned. The method of mining remains to be described, although it is still in the experimental stage. (See fig. 18, p. 32.)

Beginning at the breakthroughs connecting the rooms with the upper headings or mined grounds, the pillars will be worked by taking one-half on either side of the rooms; thus operations in two adjacent rooms will remove all of the ore in that pillar. No two faces will be worked in a continuous line, but the working faces of the adjacent rooms will be arranged in steps, the most distant being attacked first, with the adjacent face 15 to 25 feet behind the previous one. Removing pillars, like all previous work, will be done in the lower bench.

After pillars are removed and temporary supports, such as props, are placed, the upper-bench ore will be attacked and broken down by shots until caving starts. The broken ore will fall upon the stope floors and will be loaded into cars running on the original room tracks. Upper-bench ore will be caved after the lower-bench pillars are removed and will be taken down until the presence of waste rock indicates that it is too lean for use. Figure 35 pictures two stopes where the ore has been mined from the lower bench, and the upper bench provides support for the workings.

The work as planned is evidently a panel system; all available ore in a panel is removed and the worked-out area filled by the top rock caving. This procedure has the following advantages: 1, All preliminary or development work will be narrow, thus insuring against falls; 2, all work will be done in the lower bench and the ore of the upper bench will serve as top rock; 3, the removal of the pillars in the lower bench will also be under the protection of the upper bench, with such temporary protection as may be found desirable or necessary; 4, as the ore of the upper bench in broken down by caving, mining will be done cheaply and protection will be supplied
by unmined pillars in the lower bench and by props supporting the upper bench; 5, if the worked-out areas are systematically filled by caving of the top rock, the unmined part of the ore bed will be relieved of great pressure.

Further reference to panel mining will be found under "Suggested changes in mining practice."

**ROBBING PILLARS**

Mining involves stoping or breaking ore, in contradistinction to operations in which the breaking of ore is merely an incident, as in development. The usual stoping methods employed in iron-ore mines have been discussed, and the removal of about 60 per cent of the ore thereby may be assumed. All work needed for removing the 60 per cent of ore may well be considered mining proper; robbing pillars, a secondary operation that may be called "percentage work," involves two other operations that are usually employed, namely, slabbing and forming breakthroughs. Splitting pillars may also be logically included with these two.

Pillar robbing is the final operation in removing ore from the stopes, and the ultimate extraction depends upon it. More skill and care are required than in stoping, as the pillars have taken weight and the top rock has been weakened by being long exposed to air and moisture and having less support; moreover the ore is more difficult and consequently more expensive to handle, because it is farther from the heading tracks.

Robbing pillars, which marks the completion of mining, must necessarily weaken still further an already weak top, therefore it is begun at points farthest from the main haulageways or slopes. In mines where roof ore is left to support the draw slate above and
can be removed with safety, the first operation is usually to quarry or shoot down the roof ore, then other robbing is commenced.

Falls of rock, the threatened collapse of pillars, and occasionally the heaving of the bottom rock may greatly reduce the amount of ore recovered in robbing pillars, and may even cause abandonment of the work with little accomplished.

**SLABBING PILLARS**

Slabbing of pillars involves attacking the pillars on the lower side and shooting off slabs of ore of varying width, depending largely on the thickness of the pillar worked. Slip planes facilitate the work materially, for large amounts of ore can be easily slabbed off with quarrying bars when the slips have been opened by pillars taking weight. Slabbing usually starts at upsets or breakthroughs and extends to adjacent upsets. It may also be done on the side of upsets and breakthroughs, enlarging many of them to twice their usual width. The condition of the ore and the wide face that can be attacked make slabbing ore from pillars probably the most economical form of mining. (See fig. 36.)

**BREAKTHROUGHS**

Breakthroughs are driven through the pillars after slabbing has reduced their width one-half or more; long arch pillars are thus broken into a series of small pillars that usually are very irregular in size and shape. Upsets 50 feet wide may be driven without previous slabbing of pillars; then pillars are left about 25 feet square, which are usually more uniform in shape and arrangement than when slabbing precedes or accompanies the driving of upsets. (See fig. 37.)

Where the upsets and breakthroughs in the preliminary mining in the pillars have been spaced at irregular intervals the final ex-
traction of ore by driving upsets may be smaller than if the upsets had been spaced at regular intervals and at distances that were multiples of 25 feet. Driving upsets will not readily divide arch pillars of different lengths into small pillars standing at regular intervals, as there will be numerous large pillars that can not be formed into smaller ones without unduly reducing the support of the roof. (See fig. 38.)

**SPLITTING PILLARS**

The term "splitting pillars" is often applied to the subdividing of pillars by upsets driven into and through them. It would seem more logical to apply the term only to dividing a pillar into two parts by driving a passage through it along its greater dimension, or, in its more general application, parallel with the headings, which is in fact the direction taken in splitting.

The pillars to be robbed by splitting are usually laid off into sections, each served by a branch or cross-heading track from the main-heading track. The tracks shown in solid lines in Figure 39 are ar-
ranged for low and moderate grades; for steeper grades a switchback may be employed, as shown by the broken lines. This practice will be mentioned again in connection with special methods of mining the upper and lower benches of the Big-Sewam. Fully 50 per cent of the ore in a pillar can be mined by splitting, particularly if the splitting is done in conjunction with the driving of upsets.

A-HEADINGS

A-headings, also formerly used in splitting, are secondary openings driven parallel to and halfway between the main headings. The A-headings were formerly employed extensively to extract the relatively large blocks of ore left in mining by wide stopes, already mentioned. (See fig. 40.) These blocks were cut by the A-headings, which extended from the manways through pillars and stopes, and had suitable grades which usually favored the loaded cars. Stopes
25 and 30 feet wide were cut through the pillars along the lines of the A-headings, and in a subsequent operation the pillars were reduced in size as shown. When the A-headings were ultimately abandoned in a section of a mine, the arch pillars directly above them had been reduced to small more or less regularly spaced blocks of ore, and a second row of wall pillars some distance below represented all that was left of the lower part of the pillars. The extra expense of driving the A-headings, and especially the difficulty of handling ore and waste, were largely responsible for the abandonment of robbing pillars by such means.

**SPLITTING PILLARS IN IRREGULAR AREAS**

The splitting of pillars is a common practice when irregular areas resulting from a change in dip of the ore bed are worked, the main difficulty being the laying of track to the pillars at grades suitable for handling the ore. Gravity planes have occasionally been employed, but switchbacks are preferred when practicable. (See fig. 41.)

**ROBBING ENTIRE THICKNESS OF BIG SEAM**

Robbing pillars that include the entire thickness of the Big Seam, as when the lower bench is worked, is relatively much easier than robbing in the upper bench alone, because there is a larger surface to attack, and the pillars have stood for some time under a heavy roof pressure, which has weakened them considerably. Pillars are usually robbed after lower-bench ore has been removed by upsets and cross headings.

When upsets are used to reduce the size of pillars, branch tracks are run along the lower side of the arch pillars to handle the ore coming from the upsets. Upsets 25 feet wide and spaced 25 feet apart are driven into the pillars, many of which are still further reduced in section by slabbing.

Cross headings are formed as already described and as shown in Figure 39; they follow the mining of the lower-bench ore in the
stopes below. With heading intervals of 85 feet and stopes 35 feet wide, the remaining 50-foot pillars are split by cross headings 30 feet wide, which leave 10-foot pillars of ore on either side. Frequent breakthroughs reduce the thin pillars to stubs measuring 8 by 10 feet (many of them are smaller) as the final support for the roof. The pillars soon collapse, and in order to obtain as much ore from them as possible, many are broken down by shots; then if the top rock does not follow, the greater part of the ore is loaded out. (See fig. 42.)

Pillars 10 to 15 feet across and 18 to 20 feet high are readily broken by placing two shots in the base and one two-thirds of the way up. The amount of powder used need not be large, but in any event the quantity of ore broken is considerable and is cheaply obtained; however, removal of the remnants of the arch pillars or stubs

![Diagram](image)

**Figure 42.**—Use of cross headings in mining lower-bench and arch pillars on moderate dips

is hazardous, for injury or loss of life to miners, as well as serious damage to workings, may result from falls of roof.

**DRILLING AND BLASTING**

Ore can not be drilled and broken by explosives according to any fixed rule; practice must be flexible enough to meet the different conditions encountered. Aside from the comparatively slight variations in physical character of the ore and the dip and strike of the ore bed, bedding and joint planes probably exert the greatest influence on the breaking of ore in development and mining.

As stated before, the Big Seam is divided into two distinct parts or benches by a shale parting that ranges from paper thinness to over 2 feet. The bond between the upper and lower benches or between the benches and the shale parting is exceedingly weak, and there is no difficulty in separating the ore from the "slate." In fact, so weak is the bond between the several parts that much waste caused by breakage of the parting must often be sorted out. In gen-
eral, it may be said that all other distinct partings of shale or sandstone in the workable areas of the Big Seam are local and inconsequential. A relatively large number of seams or separations is, however, particularly prominent in the soft ores and in the hard ores at certain localities.

SEAMS AND SLIPS

Stoping operations often reveal seams which are of importance because they facilitate reduction of the larger masses of ore by sLEDGING. The presence or absence of more or less pronounced bedding planes between the top of the ore bed and the shale top rock, or between the roof ore and the ore immediately below it may have a most decided effect upon the shooting of ore from the stope face.

Where there is little or no connection or bond between the ore bed and the top rock, the separation of the two is easy, but where the various parts are closely knit together great care must be taken in placing holes and shooting; otherwise much damage may be done to the top formations, for they may be seriously weakened, often causing premature and serious falls.

Joint planes, or so-called slips, in the ore bed probably have the greatest influence upon development and mining. The systematic and orderly occurrence of the slip planes and their persistence throughout the whole extent of the ore bed make their influence on mining operations of the greatest practical importance. (See fig. 11, p. 20, and fig. 12, p. 21.)

When mining was confined largely to surface work in the soft ores and underground mining was beginning, full advantage was taken of slips, particularly in quarrying operations and in driving slopes. The occurrence of slip planes at more or less definite intervals and their persistence in a given direction permitted slopes to be driven easily and cheaply. Difficulties were experienced, however, in following the slips, chiefly because the slips do not follow the dip of the ore bed and because they extend into the top formations, thus weakening the roof. (See fig. 43.) More recent practice ignores the occurrence of slips in the development of mines, and they are now important chiefly because they facilitate the breaking of ore in stoping.

Miners designate slips in stopes as "mining" and "face" slips, according to direction; the former parallel the headings and the latter parallel the stope face or are normal to the headings.

The ore is remarkably uniform in texture throughout the full thickness of the bed, and only a moderate amount of explosive is needed to break it when shots are placed to take advantage of the shape and condition of the face.
DRILLING HOLES FOR BLASTING

Series of two holes each are almost invariably drilled for blasting; the unit is therefore two holes placed virtually one over the other. Where the dip of the ore bed is low the upper holes of the series are dry (they are usually so in any place), but the lower holes may be wet; with high and moderately high dips the lower holes are dry. The holes are 4 to 12 feet long, rarely more or less except where necessitated by special conditions of the face. Short holes are usually employed in squaring up and maintaining the full width of workings, as in trimming. The upper holes are directed toward the top and the lower toward the bottom of the ore bed, but are seldom run nearer than 10 to 12 inches, in order that the top or bottom rock will not be fractured.

![Figure 43.—Slope driven across instead of with slope planes](image)

DRILLING EQUIPMENT

DRILLS

Drills are mounted on tripods, no columns being used, and the two holes of a unit are drilled from one set-up when possible. Largely because of this practice long holes are required, although placing holes to take advantage of slip planes is a contributing cause. Holes are usually drilled so that shots may be placed back of slip planes, never in the slip itself; otherwise, much of the force of the explosive would be wasted.

Mounting drills on tripods gives the drill men wide range in the direction of hole drilled, as nearly the full 180° vertical arc is available, and by regulating the distance of the drill from the working face a wide horizontal range is obtained. Full advantage can there-
fore be taken of the shape and condition of the working face with respect to their effect upon drilling and breaking ore. A tripod is usually stood on two legs braced to steady it and the third leg is set against the face of the ore above the machine to keep the drill in alignment. (See fig. 44.)

Until about 5 years ago practically all drilling in the iron-ore mines was done by piston drills mounted on tripods and using solid steel. Many such drills are still used, but the latest practice calls for the Leyner type of water hammer drills. Makes of drills commonly found in the mines are the Ingersoll-Rand, Denver Rock Drill, Sullivan, and Chicago Pneumatic Tool, which differ little in size and weight. The 160-pound drill is ordinarily used.

Holes are usually drilled dry, but water drills are now employed, almost without exception, in narrow work, such as in driving slopes and manways and turning off headings. There is a strong sentiment in favor of water drills, however, and at least one mine is fully equipped with them for all drilling. There can be no question of the injurious effect upon miners of breathing the highly siliceous dust caused by dry drills, and it is safe to say that when this fact is more widely appreciated wet drilling will replace dry.

In certain mines throwing water into the hole by hand during drilling is the nearest approach to the use of water in stoping, but the water is so used not to allay dust but to enable cuttings to be
removed from the holes more readily. It is very difficult to clear dry holes of the fine, heavy dust, even by the use of compressed air. Often the relatively large end of a tamping rod is forced into the hole, and the dust or sludge is expelled by the air currents set up by the plungerlike action of the rod, which also loosens the packed-in cuttings. Compressed-air jets, however, are usually sufficient to clean both wet and dry holes.

Hand hammer drills are widely used, but not in large numbers. They are employed in trimming, cutting hitches for timber, and blockholing; recently they have been mounted on tripods by means of a special carriage and used in stoping. Hand hammer drills are used both wet and dry, but rarely as water drills. A serious objection to dry drilling with hand hammer drills is the large amount of dust thrown into the air; indeed, the mines where they are extensively employed are exceedingly dusty.

**COMPRESSED-AIR SUPPLY**

Drills receive compressed air through pipes laid in the manways and branch lines to the respective headings. The air pressure at the drill is 80 pounds, although it may fall to 50 or 60 pounds, and at the compressor is about 100 to 125 pounds. As the compressors are at the surface in the power plants, they are driven by steam or electricity, electric drive being a recent innovation. There are belt and direct drives. Alternating current (2,200 volt, 3 phase, 60 cycle) is supplied to the motor. As an illustration of the tendency in mine practice, the last three compressor units to be installed have a capacity of 5,000 cubic feet per minute each, when running against 100 pounds pressure and at a speed of 180 revolutions per minute. The makes of compressors commonly used in the district are the same as those of the rock drills.

**DRILL BITS**

The size of hole commonly drilled to hold the explosive is 2 to 2$\frac{1}{8}$ inches, a stick of powder being 2 inches in diameter. A set of drill bits for a 12-foot hole ranges in size as follows:

*Sizes of drill bits for 12-foot holes*

<table>
<thead>
<tr>
<th>No. of drill</th>
<th>Length, feet</th>
<th>Diameter of bit, inches</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>2</td>
<td>2$\frac{3}{4}$</td>
</tr>
<tr>
<td>2</td>
<td>4</td>
<td>2$\frac{5}{8}$</td>
</tr>
<tr>
<td>3</td>
<td>6</td>
<td>2$\frac{1}{2}$</td>
</tr>
<tr>
<td>4</td>
<td>8</td>
<td>2$\frac{3}{8}$</td>
</tr>
<tr>
<td>5</td>
<td>10</td>
<td>2$\frac{1}{4}$</td>
</tr>
<tr>
<td>6</td>
<td>12</td>
<td>2$\frac{3}{8}$</td>
</tr>
</tbody>
</table>
Probably a more representative set of drill bits for use with smaller sticks of powder is as follows:

Sizes of drill bits for holes taking smaller sticks of explosive

<table>
<thead>
<tr>
<th>No. of drill</th>
<th>Length, feet</th>
<th>Diameter of bit, inches</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>3</td>
<td>2 1/2</td>
</tr>
<tr>
<td>2</td>
<td>5 1/2</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>8</td>
<td>1 3/4</td>
</tr>
<tr>
<td>4</td>
<td>10 1/2</td>
<td>1 5/8</td>
</tr>
<tr>
<td>5</td>
<td>13</td>
<td>1 3/8</td>
</tr>
<tr>
<td>6</td>
<td>15 1/2</td>
<td>1 1/2</td>
</tr>
</tbody>
</table>

The diameter of the bit can readily be altered by a power drill sharpener, one-eighth to one-sixteenth inch change being made for each 2 1/2-foot run of steel.

The steel used ranges from seven-eighths inch to 1 1/4 inches; its cross section may be round, hexagonal, or octagonal. With water drills 1 3/8-inch to 1 3/4-inch steel is used. Hollow round steel is used in the drifting machines that drill to depths of 12 to 20 feet. Hand hammer drills use seven-eighths-inch to 1-inch hexagonal steel; the 1-inch size is for wet drills.

EXPLOSIVES

In narrow work 60 per cent dynamite is usually employed in breaking ore 35 to 40 per cent. Charges of explosive for breaking ore vary from 8 to 10 one-half pound sticks up to 16 sticks for the upper holes, while 12 to 18 sticks may be necessary for the lower holes. The lower holes are usually fired first to free the lower part of the face, consequently they require heavier charges. Fuse and caps are employed to fire shots in all work except the driving of slopes and manways, where electric firing is the practice.

Cartridges are split before being inserted in the holes, which are thoroughly cleaned before charging. One or more cartridges ("sticks") are placed at a time and rammed home with a wooden tamping rod. The primer is placed in the charge some distance from the top end. After the powder is placed, the hole is usually filled with moist drill cuttings, which are ordinarily made into sticks or "dummies" by being packed into paper shells. Fuse cut into lengths to give the firing time desired is supplied to miners. Ordinarily a blasting foreman indicates the amount of powder to be used in each shot; if there is no such official the regular foreman gives directions as to the position of the hole and the amount of powder to be used.
SUPPORT OF MINE WORKINGS
CONDITIONS AFFECTING SUPPORT

The support of the workings in the Red Mountain ore mines has been relatively unimportant up to the present time. The growing tendency to leave smaller areas of unmined ore as support is bound to make trouble as the workings go deeper and the weight of the overlying formations increases. In fact, where mining is being done at a vertical depth of 1,500 to 2,000 feet, much trouble is experienced already.

WATER-BEARING FORMATIONS

The problem of roof support is complicated by the water-bearing formations that overlie the ore beds in many places. The distance of the principal water-bearing formations from the ore bed and the impervious formations lying between prevent any great amount of water from seeping down into the mines, but the breaking of the impervious beds by fissures caused by subsidence permits water from reservoir formations to drain into the workings. The millions of gallons of water pumped daily from the mines emphasizes the importance, if not the absolute necessity, of preventing falls that would tap the water-bearing formations above the ore beds.

The Big Seam is about 150 feet below the Fort Payne chert, the first and the chief of these water-bearing formations. The Hartselle sandstone is fully 500 feet above the Big Seam. Falls of rock in the workings need not be extensive to reach a vertical height of 100 or 150 feet, when the chert bed would be broken and any water therein would readily escape into the mine. In fact, numerous falls have reached the surface, a distance of several hundred feet.

Although the Fort Payne chert is known to carry water at various points fairly near the outcrop of the Big Seam, it has not been proved to be water bearing at all points along the mountain and at considerable distances from the outcrop. The Hartselle sandstone is known to carry large quantities of water, and is used extensively as a source of water supply for power plants. Where the water menace exists, however, adequate support should be provided to prevent falls of top rock in the mines.

STRENGTH OF FORMATIONS

The formations that overlie the ore bed comprise shales ("slates") and sandstones, which in themselves are strong and would usually give adequate support to workings of any reasonable size in the ore bed, but unfortunately their strength is materially reduced by slips or joints which extend some distance above and below the ore bed. These slips reduce the compressive strength of the ore and the bending strength of the top formations.
The top rocks are, moreover, cross-bedded, and in many places the length of what would otherwise be strong beams of rock is reduced to only a few times the thickness. Although cross-bedding planes occur commonly, they are not as persistent or as prominent as slip planes. (See fig. 13, p. 21.)

RELATIONSHIP OF BEDS

When the formations overlying the ore bed were studied, the first bed of sandstone thick enough to be of chief importance was chosen as the element controlling support at that point, and the thinner and weaker formations overlying the controlling member were ignored. The formations below that sandstone and directly above the ore bed require temporary support.

The ratio of distance between points cut by cross-bedding planes and the top and bottom of the controlling members and the thickness of the members was also determined, giving a relative comparison of the value of the supporting members.

Furthermore, slip planes were considered as an element of weakness in the top rock. The number, direction, and especially the prominence of slips in the mine workings were noted.

Finally, the percentage of slate in the first 5 feet of rock above the ore bed was calculated. This figure in itself might be taken as an index of the strength of top rock.

Grouping the results of these observations, a composite value was obtained for the relative strength of top formations as factors in mine support. It was shown that, beginning about 3½ miles north of Red Mountain Gap, and continuing south along the line of Red Mountain for nearly 6 miles, the condition of the top formation is poorest. From 1½ miles south of Red Mountain Gap to 7 miles farther south the conditions of the top rock are best; from the last point—8½ miles from Red Mountain Gap—to a point 3 miles still farther south conditions rank second; and beginning 11½ miles from Red Mountain Gap and extending south for 2½ miles the third-best conditions exist. This study of the relative strengths of the top formations covers the actively worked part of the field, extending along Red Mountain for 4 miles north and 14 miles south of Red Mountain Gap. (See fig. 45.)

DIP OF FORMATIONS

The dip of the formations decidedly affects methods of working as well as support. When the beds are flat the entire weight of the cover must be supported by pillars or timbers; when they dip the vertical component of the load to be supported falls more or less obliquely across the pillars until it passes entirely into the walls when verticality is attained.
Figure 45.—Ratio of distance between points cut by cross-bedding planes and top and bottom of beds, thickness of roof ore and sandstone top rock, and percentage of slate in first 5 feet above Big Seam.
The true dip of the Big Seam, as observed on Red Mountain, ranges from 10 to 35°, the average being about 20°. The direction of the dip ranges from 32 to 58° SE., the mean being S. 50° E. (See fig. 7, p. 16.)

**SUMMARY OF CONDITIONS**

The general conditions that affect roof support are: 1. Character and condition of ore and top formations; 2. occurrence of slip and cross-bedding planes; 3. amount and direction of dip of ore bed; 4. occurrence of irregularities in formations; and 5. occurrence of faults.

At the top of the Big Seam and an integral part of it is a layer of ore or highly ferruginous sandstone which may or may not be separated from the ore bed by a distinct seam or line of separation.

![Figure 46. Roof ore left as support for draw slate](image)

If this top material is good ore it is designated "roof ore" and in places is removed after the underlying ore is mined and before pillars are robbed. Where the line of separation is well defined the upper part can readily be left to support the slate above, but where the separation is not well defined, to leave a protecting top for the slate is much more difficult. A relatively smooth plane of separation indicates a well-defined seam; a rough, irregular surface indicates little or no separation. (See fig. 46.)

In certain places the roof ore or sandstone is of remarkably uniform thickness, and is a reliable support; in other localities the
thickness varies so much as to render dependence upon it unsatisfactory and undesirable. Rolls in the roof ore usually cause the great variations of thickness.

Where the formations overlying the Big Seam are 2,000 feet or more thick, it has been found desirable to leave the entire upper bench to support the top rock, and to mine the bench later.

Aside from the effect of varying thickness of roof ore upon the support of the slate top an added difficulty arises in connection with leaving the ore in the preliminary mining operations, particularly where little or no separation from the main body of the ore occurs. Holes are often drilled too long, or too large charges of explosives are used; in either case the roof ore fractures or breaks down, exposing the slate above. Disastrous falls of roof usually follow such breaks.

**STRENGTH OF ORE**

The ore of the Big Seam is remarkably uniform in hardness, density, and texture, but slight differences are observed. At depth the ore is somewhat more dense and usually much harder. Preliminary tests of the crushing or ultimate strength of ore from various points on Red Mountain are given in the following table:

*Ultimate strength of iron-ore cubes, by groups and classes, pounds per square inch*

<table>
<thead>
<tr>
<th>Group</th>
<th>Sample No.</th>
<th>Class 1, 2 square inches</th>
<th>Class 2, 4.75 square inches</th>
<th>Class 3, 9.17 square inches</th>
<th>Class 4, 15.97 square inches</th>
<th>Average for group</th>
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The letters in the first column show the grouping of the samples by localities; the figures in the second column show the number of tests. The average crushing strength for all sizes of test pieces was 12,977 pounds per square inch. However, classes 3 and 4, which average 14,258 pounds, are probably more representative than classes 1 and 2.

The ores under the deepest cover, group F, are not the strongest, although some of the tests show strengths approaching the highest determined. The reduced strength is probably due to the numerous slips and incipient fractures produced by excessive strains in the ore.

An elaborate series of tests is under way to determine definitely the ultimate strength of the ores of the Big Seam and its relation to the character of the ore and the depth of cover.

**WEAKNESS OF ASSOCIATED FORMATIONS**

**BEDS OF "SLATE" AND SANDSTONE**

Beds of "slate" and sandstone, from a few inches to 2 and 3 feet thick, lie above the Big Seam. The slate as a rule is thinly bedded and in certain localities has numerous rolls and filled potholes. Along the line of Red Mountain, between points 4 1/4 miles north and 1 1/2 miles south of Red Mountain Gap, thinly bedded slate predominates for 12 to 15 feet vertically above the Big Seam; in other localities, particularly between points 1 1/2 and 8 1/2 miles south of Red Mountain Gap, the top rock is dominantly sandstone.

The slate lying directly above the Big Seam is commonly called draw slate, and is the most troublesome. The mines afford evidence that the top rock, particularly the draw slate, is more or less disturbed close to faulted areas, although not always in direct contact with the faults.

The sandstone that occurs interstratified with the slate is usually coarse grained and, although it has less strength than the ore, it is relatively tough and strong. As a roof rock it is much better than the slate.

**JOINT PLANES**

Joint planes or slips in the ore and the top rock are both an advantage and a source of weakness; but the advantage they give in mining is often more than offset by the damage they do the top rock. Two or more sets of slip planes crossing one another at angles of 60° and above effectively break otherwise strong top formations into beams and blocks 2 to 4 feet wide and of varying length. (See fig. 47.)

Slip planes may reduce the bearing strength of layers of top rock, when considered as elements of support or beams, almost to a nega-
tive value. Top formations spanning an excavation of 30 feet or more may be broken into six or more parts with very slight bond between the several parts, while in the case of so-called "mud slips":

![Figure 47.—Slip plane in top rock, and fallen mass of slate](image)

the slip planes may be filled with clay. Water also enters the mine through the mud slips, weakening the bond between the parts still further. (See fig. 48.)

**CROSS-BEDDING PLANES**

The cross-bedding of the top rock is an additional source of weakness. The effect is similar to that of the slips, but the bond is

![Figure 48.—Mud slip crossing stope](image)

usually much stronger; however, when the roof of an excavation begins to take weight the cross-bedding planes open and the rock breaks and falls. (See fig. 49.) The sandstone top rock and the ore
beds are cross-bedded widely. The slates are cross-bedded in places, with the result that their inherent weakness is increased.

In general the dip of the cross-bedding is south and southwest, but occasionally it is reversed, and it may vary markedly. The cross-bedding planes, therefore, usually cut the top rock along lines that parallel both the dip of the ore beds and the stope faces. This condition is more favorable to support than if the feathering out of the layers were parallel with the headings, as the width of the stopes determine the length of the beams that must be depended upon to support the roof of the stopes.

PROPS

Props placed at a and b (fig. 50) give maximum support, but when placed at c and d are of little or no value, in that they support only the feather edges of the cross-bedded layers. The difficulty is further augmented by the decreased angle of dip of the cross-bedding; but with thick beds and low angles of cross-bedding, the best conditions of support are obtained. When the top formations are thinly bedded and the angle of cross-bedding is high, falls of top rock quickly follow the mining of the ore, requiring the handling of much waste rock as the least of the disadvantages.

When props are set without due regard to the direction of dip of the cross-bedding, particularly when there are no visible lines of separation in the freshly formed roof to indicate the presence and direction of the cross-bedding, preventable falls may occur. This misplacing of props may be responsible for numerous falls in one part of a mine and not in another, due to the direction of the cross-bedding.
As a rule slate top rock is very brittle and breaks readily at the points or lines of support when taking weight. This characteristic of slate has given the impression that little pressure is exerted by the roof on pillars and props. Failure of props by breaking or brooming is rarely observed, and is easily accounted for by the fact that the roof breaks around the props, sometimes leaving a slender column of slate standing on them, but more often breaking and falling away, whereupon the props fall with practically no sign of having taken pressure. (See fig. 51.) The failure of pillars is, however, evident, and in many mines very pronounced, as shown later.

**EFFECT OF DIP ON SUPPORT**

The dip of the ore bed has a marked and often controlling influence on mining, handling of ore, and support of roof. The apparent dip, as usually found by compass and clinometer, is not given in this connection, the true dip being always recorded. The true dip was determined by a special instrument that obtained very accurate results. The direction of the dip was similarly taken and exact data were gathered for work requiring the application and direction of the dip. (See fig. 7, p. 16.) It should be stated, however, that all observations on dip were taken along the line of outcrop of the Big Seam on Red Mountain and on formations underlying the ore bed which have not been disturbed by mining operations.

Owing to the badly disturbed condition of the ore bed, due to folding and faulting, particularly at certain points along Red Mountain, the dip has a wide range, often varying from the horizontal to 60° or more with the horizontal. The average dip is about 20° and the mean direction is about S. 50° E.

When pillars are of normal size the dip affects support little, but with thin pillars, particularly when high, as when both benches of
the Big Seam are worked, they undoubtedly tend to overturn, as evidenced by diagonal cracks developing in line with the direction of the resultant force. Long narrow pillars, with their longer dimension parallel with the dip, which are ultimately broken into small stubs by robbing operations, are employed to advantage on the higher dips.

Props stand well on dips ranging up to 8° and 15°, beyond which they correspond more nearly to stulls in that they are set in hitches to prevent slipping on the footwall. The use of props in stopes driven in highly pitching beds is shown in Figure 52.
Where changes of dip are more or less abrupt the top and bottom rocks are very likely to have become weakened by bending; otherwise there is no serious difficulty in supporting the roof on high or low dips.

Aside from the occurrence of slips and cross-bedding planes, other conditions, in the nature of irregularities in formation, may and often do markedly influence support. In certain localities rolls are a common occurrence in the top formations and as viewed in part may be taken as cross-bedding. These irregular formations, when coupled with cross-bedding and thinly stratified slate and sandstone, make the roof weak and treacherous. Rolls of small lateral extent are designated as “pots” and “kettles,” and affect conditions of support only locally (See fig. 9, p. 19.)

**EFFECT OF IRREGULARITIES IN OCCURRENCE**

In the southern part of the district are more or less symmetrically shaped cylindrical masses of rock called filled potholes, composed largely of clay and fragments of ore and with diameters of 18 to 30 feet, which cut entirely through the ore bed. Their origin is attributed to the action of running water bearing sand, gravel, and boulders, which cut their way down through the previously formed ore bed; the openings thus made were later filled with débris. It is interesting to note that the larger dimension of many of the filled potholes is at the bottom of the ore bed, forming truncated cones, and that above the bed the materials of the potholes conform with those of the formations above the ore.

*Figure 52.—Ore bed dipping steeply*
Potholes do not cause any serious difficulties, except that occasionally they occur in line with the headings or close to them and must be cut through or the headings narrowed. No pronounced weakening of the top rock in contact with ore near potholes has been observed; they are, in fact, as strong as the ore and stand for long periods after exposure by mining operations without any signs of disintegration. (See fig. 10, p. 19.)

**EFFECT OF FAULTS**

The ore beds on Red Mountain are cut by numerous faults, some of large displacement, but by far the largest number have only several feet displacement. Faults affect support but little except locally, when the top formations may be badly disintegrated and weakened. Leaching of the lime in the top rock may be largely responsible for the weakness observed. The occurrence of several lines of faulting and displacement at intervals of a few feet and the presence of large quantities of water usually make support conditions bad, but in the next adjacent headings, both above and below, normal conditions may prevail. The fault planes are usually tight, and little water finds its way into the workings through them; however, in some instances much water enters directly through or near the fault planes. Faults in themselves may not change support conditions materially but, where faults are coupled with weak top rock and the presence of water, conditions extremely bad for support may prevail. (See fig. 53.)

Well-defined evidences in certain localities indicate that the occurrence of faults may have had a decidedly injurious effect upon the top formations by breaking the bond between them through the
differential action of the downward drag of the displaced part of the ore bed.

**TYPES OF SUPPORT**

**ORE LEFT IN PILLARS**

Ore left in pillars in the underground workings constitutes the best possible support. It is already in place and only has to be left unmined and in the proper position to give the support desired. However, the urgent demand for more ore and higher extraction tends to lead to reduction in the size of pillars. Workings could undoubtedly be given adequate support if one-third of the ore is left unmined; observations in abandoned workings under relatively slight cover and in the more recent workings under greatly increased cover show this to be true. In fact, it is now pretty well recognized that much timber was needlessly wasted in the early days of underground mining, when little or no support was needed but the pillars of ore. (See fig. 54.)

In a number of the large mines on Red Mountain 30 per cent of ore is left unmined, which, aside from giving ample support, also provides about the proper relation between height and width of pillars, namely, the height one-third the width. For example, in an area where the ore bed worked is 10 feet thick the pillar should be 30 feet wide, or if the bed worked is 7 1/2 feet thick the pillar should be 25 feet wide. Few signs of failure of pillars have been observed under such conditions of support and the roof stands remarkably
well. The average extraction is probably 60 per cent, 40 per cent of the ore being left as pillars; but, as mentioned before, the presence of slips, cross-bedding planes, and other irregularities may materially alter the conditions affecting support by reducing the strength of the ore.

Roof ore is an important supplement to pillars of ore as support. The thickness of the roof ore varies widely in different localities, but where ore is left as a temporary or permanent top rock it probably averages 10 to 12 inches, the range being from a few inches to 3 and 4 feet in thickness.

The roof ore serves a double purpose in support when left standing; it is strong and tough and acts as a buffer in mining, in that it safeguards the weaker draw slate immediately above from being injured and weakened by careless shooting. Roof ore, like ore in the pillar, is materially weakened by slips and cross-bedding planes, but it may serve a useful purpose as a temporary support for the weak top rock.

Barrier pillars are left between properties, and are 50 to 100 feet wide. They are supposed to be unbroken strips of unmined ore bed separating properties, but are usually cut at frequent intervals by headings.

**TIMBER PROPS**

**PROPS IN STOPEs**

Props have been used to supplement the pillars in support of the roof ever since underground mining was begun in the district.
They were formerly placed in the stopes in two rows 25 feet apart, and as an extra precaution the distance was sometimes reduced to 12 feet. Props were also placed close to the working face, so close in fact that many were broken and blown out, and rendered worthless for further use. At present props are not placed closer than 50 to 60 feet from the stope face, a practice which has been largely responsible for the abandonment of "pegging" of timbers. Iron pins were formerly driven into holes drilled in the roof, encircling the side of props opposite the working face. The top and bottom ends of the props were then held securely in place, and few timbers were displaced by flying rock or ore from shots.

![Figure 56.—Long props used when all of Big Seam is worked](image)

The normal and systematic use of props in stopes with average roof conditions is shown in Figure 55, while abnormally long props (22 feet or more) in a stope where both benches of the Big Seam were worked are shown in Figure 56. The use of props in connection with packwalls of waste from falls of top rock is shown in Figure 57.

It is claimed that setting timbers, wedging them up, and pegging are in themselves largely responsible for the failure of top rock. Much of the difficulty is now overcome by placing fewer props and omitting the pegging.

Extensive caving of top rock before the completion of work in a stope may require the use of numerous long props to maintain the haulageways and prevent falls. (See fig. 58.)
PROPS IN HAULAGeways

Special forms of support are employed in the haulageways, such as slopes, headings, and tunnels, but they are usually confined to localities where bad conditions of support exist. In numerous localities are to be found sets made of round and sawed timber spaced from 2 to 4 and 6 feet apart. Much lagging is used, often consisting of large timbers placed as crib work above the sets and occasionally reaching a height of 10 to 12 feet. (See fig. 59.) Steel rails are occasionally used as lagging, especially where the roof is smooth or can be made so for practically the whole width of the timbered passage.

Figure 57.—Pack walls of waste rock built between props

Where permanent haulageways are maintained through ground difficult to support, various combinations of methods are employed. Heavy timbers may be placed horizontally across the passage at a height of 7 to 8 feet, the ends being needled into the walls or supported on a row of iron pins. These timbers form the footing for sets usually provided with extra-long caps upon which a lagging of timber or steel rails is placed, making close contact with the roof.

Although many slopes have no other support than the slope pillars, experience has shown that ultimately permanent supports will have to be provided. Timbers in the nature of props and sets are used, the latter under more difficult conditions. The timbers are 12 by 12 or 14 inches and 14 by 14 inches and often are spaced 1 to 4 feet
center to center. The timber commonly used in underground work for all types of timber supports, including props, sets, and lagging, is "heart" pine.

**PROPS FOR LOADING STATIONS**

The support of loading-station excavations along the line of the slopes has been made somewhat more difficult with the adoption of skip haulage, the size of the stations having been enlarged materially, particularly in height, which is practically doubled. The large slope pillars have usually provided ample support for the excavations made for stations, although props are commonly employed for such temporary support as may be necessary. With less favorable conditions of top rock both wooden and all-steel sets are used. Although stations are occasionally concreted, the practice is not general.

**SUPPORT OF MANWAYS**

As manways are narrow they usually stand for long periods with little or no support. (See fig. 60.) Props are occasionally used where top-rock conditions are bad, but manways usually remain open throughout the normal life of a mine, because wide pillars are left on at least one side and occasionally on both sides to protect slopes and manways. As an additional protection, the disturbance due to caving ground in the stopes is overcome by causing the top rock to break in the stopes along a line paralleling the dip and near the first upset beyond the manway. This is accomplished by setting a row of props across the stopes from the heading track to the arch pillars above; the breaking of the roof along this line of resistance relieves the pressure on the slope side.
MASONRY WALLS

Before concrete was used, masonry walls were occasionally built as side supports of slopes, waste rock from the mine being readily accessible. These walls have stood well and have proved satisfactory where the top rock was strong enough to stand without intermediate support. Concrete was next employed and will be employed in future, being readily placed, strong, and fireproof.

CONCRETE

An attempt is seldom made to remove the supporting sets when slopes are concreted, but they are inclosed by the concrete, and if any timber extends through the concrete when the lining is completed it is sawed off flush with the surface of the concrete. As a result, when pressure comes upon the lining the inclosed timbers are occasionally pushed into the slope; this action is slow, however, and not particularly dangerous. Concreted slopes with single or double ways are employed, but two tracks are not usually placed in one passage—a precautionary measure to insure against accidents in hoisting. Slopes are occasionally concreted on the bottoms alone to support the haulage tracks, and are so used with or without concreting of the walls and top. Such concrete track supports may assume two forms—individual stringers for each rail or a solid bed

Figure 59.—Timbering in heading with weak roof
of concrete supporting and binding together both rails. When concrete stringers were used for each rail it was found that the tracks were not stable, both alinement and grade changing gradually under the strain of heavy loads and high speed of hoisting. When a solid bed support is given to the track the rails are bound together with old rails and bolted securely; the whole structure is then imbedded in concrete. Cushioning is seldom provided for the rails of the track, although it has been found desirable in general practice.

The concrete bed between the rails is usually dished somewhat, and the concrete on the outside of the rails is sloped outward so that ore falling on the roadbed may clear the tracks, reducing the risk of derailment.

The building of permanent concrete power and pumping stations has been the practice for some time, although concrete construction usually consists of walls in headings, stopes, or concrete floors leveled up and provided with adequate drains for special excavations in the ore.

**Figure 60.—Manway in iron mine**

**FAILURE OF SUPPORTS**

**PROPS**

As a rule props do not break under excessive weight of roof, the weakness of the top rock protecting them from failure through breaking and brooming. However, other wooden structures used as sets do fail under pressure of the top formations, because the caps and lagging prevent the top rock from falling, thus relieving the posts or legs of the sets from pressure.
TOP ROCK

Top rock fails along lines of slip and parallel with the strike of the cross-bedding planes. As the strikes of the cross-bedding and certain slip lines occasionally parallel the headings, an extremely bad condition of top rock may result, making the support of the roof for any great length of time very difficult or impossible.

Top rock almost invariably collapses and falls first along the line of upsets, which is partly if not wholly due to the extra area left unsupported, for it has been observed that usually where upsets are timbered top rock seldom falls. The same statement can also be made regarding the failure of roof ore, where the first breaks occur on the strike down the middle line of the upsets and usually along the slip planes. However, roof ore usually fails because drill holes have been too long or charges of explosive too heavy when ore is broken in the preliminary mining operations. It is to be expected that as the mines are deepened the condition of top formations will become worse, because they will be under strains beyond their limit of elasticity, and it will become more and more difficult to support the thinly laminated beds of draw slate.

PILLARS

The failure of pillars of ore has not been a matter of much concern; however, with increased depth of cover the breaking down of pillars will demand serious attention and may ultimately require more or less radical changes in mining practice.

That pillars are failing under pressure of cover is a fact beyond question, as extended observations in a large number of the mines have revealed. The only consideration that need be given to the matter in this connection is: What causes the failure of pillars?

Pressure of cover acting upon the pillars left in the mines for support of roof may manifest itself in several ways: 1, By the heaving of the bottom rock or footwall; 2, by the breaking off of wedge-shaped masses from the top and the bottom of pillars; and 3, by the spalling or slabbing of the body of pillars.

HEAVING OF BOTTOM ROCK

The weight of the top formations is transmitted through the pillars to the bottom rock, which being free to do so moves upward or swells in folds in the stope floors, often forming large cracks and seriously breaking up the slate bottom. (See fig. 61.)

BREAKING OF WEDGE-SHAPED MASSES FROM PILLARS

This upward movement, clearly evident some distance from the foot of the pillars, also occurs at and along the line of the pillars;
in fact, although the movement under the foot of the pillars is correspondingly less with distance from the face of the pillar inward, it is present and the force acting is even more powerful than at some distance from the pillar, where the result of the action is much more in evidence. The differential action of the pressure exerted through the pillar to the underlying formations forces off wedge-shaped masses of ore from the foot of the pillar, or if concentrated some distance from the face of the pillar may produce cracks extending from the bottom to the top of the pillar. With the loosening and final breaking off of the ore masses the pillars become smaller until their ultimate strength is greatly reduced and they collapse. (See fig. 62.)

**SPALLING OR SLABBING OF PILLARS**

The cause of the spalling and slabbing of pillars is not thoroughly understood at present, but indications point to expansion of the pillars under pressure.

Most of the pillars observed to be failing probably show more pronounced disintegration at the bottom or foot, which is often so intense that the action assumes the nature of mining or stoping.

The failure of pillars takes place irrespective of direction and occurs on all sides of pillars alike, but is much more pronounced on
the lower side and in upsets, being rarely observed on the upper side; the shape of the pillars seems to control the direction taken by the lines of failure. (See fig. 63.)

**CONDITIONS AFFECTING SUPPORT**

The conditions affecting support in the Red Mountain ore mines are fairly well known, and the changed conditions resulting from extension of workings and increased depth of cover are observable, but conditions at a depth of 2,000 feet or more are largely conjectural. There is evidence, however, that the conditions now existing in the operating mines are likely to become worse rather than to remain the same or improve.

In parts of the mines where small and inadequate support is provided and where robbing pillars has removed a large part of the support of the roof, there are unmistakable evidences of the failure of pillars, with the resultant collapse of roof, as shown by extensive falls of top rock. In other localities, even where an undue amount of ore has not been removed, pillars are wasting away, thus
slowly but surely threatening the stability of the mine workings: often such conditions exist where the cover is far less than 2,000 feet. If, therefore, the thickness of cover should be increased, as is gradually taking place where workings are extended, what will be the effect upon the conditions of support when covers are 2,000 to 3,000 feet thick? This question can be answered only in part and in an inadequate way by reference to conditions known to exist in the one or two workings that have attained such depths. It should be mentioned, however, that extreme conditions existing in any particular locality may be the result of a combination of conditions, and not attributable solely to pressure from depth of cover.

\begin{figure}
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\caption{Breaking of pillar under pressure}
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**EXPLOSIVE ROCK**

One possible and plausible theory advanced as a contributory cause of particularly bad conditions of top formations and weakened condition of ore is that stresses have been set up by faults that have not become fully adjusted, and unrelieved stresses still exist. The phenomenon of explosive rock has been observed in openings made some distance from the outcrop on Red Mountain and at considerable depths below the surface, indicating that conditions exist similar to those observed in other localities throughout the world, particularly in certain Michigan copper mines and the gold mines in India and the Transvaal.

Pieces of rock were observed to break from the face of the openings and were hurled with great force 8 to 10 feet or more. The size of the fragments varied from less than an inch across to masses that weighed several pounds. Pressure broke the various formations encountered into different sizes. Slate broke into relatively small pieces, fist size and under, sandstone was ground to a fine sand, and
limestone broke into the larger pieces, while ore spalled off in masses weighing several hundred pounds. The explosive action has not been apparent in the ore bed, but was exhibited to best advantage in the rocks above it when the mines were being opened. Lime and sandstone seemed to be most subject to explosion and exhibited to the best advantage this reaction to pressure. No explosive action or sudden change has been noted in the ore bed. The action in the ore, while definite and pronounced, is relatively slow; the great disadvantage is that masses of ore are loosened and fall without warning.

It is not altogether clear how the stresses act, for only the result of the release of the strains is evident. In general it may be said, however, that the pressure in the ore bed due to weight of cover is intensified, causing much more rapid disintegration of pillars than in the shallow workings. As discussed under "Support of workings," the breaking up of pillars makes itself felt and is most evident along the natural lines of weakness, such as slip planes; such action is intensified in those localities where conditions approaching or resulting in explosive action are observed. Especial care must therefore be taken to safeguard the workings against serious falls of top rock.

Although the fact is not definitely known, the expansion of the ore under pressure probably causes the explosive action observed; the spalling of pillars results from the same action but differs from explosibility in degree only. The successive breaking off of shell-like masses and splinters of ore shows that the action is continuous, the pillars disintegrating until they collapse because of reduction in size.

**CONTROLLING FACTORS IN SUPPORT**

When conditions affecting mine support and the failure of supports are considered it is well to bear in mind certain fundamental elements governing methods necessary to safe working conditions. These elements have been used as a basis in the preceding discussion, and are as follows:

1. In inclined workings the weight of the cover upon the pillars varies as the dip of the ore beds.
2. Relatively thick, strong beds relieve the pressure on pillars and maintain the position thereof until their limit of elasticity has been reached.
3. The width of underground excavations depends upon the strength of overlying formations and may be independent of depth below the surface.
4. Size of pillars and area of workings are directly proportional to depth of cover.
Extended investigations are being carried on by the Bureau of Mines to determine the strength of the red hematite ores and the causes of failure of pillars in the Birmingham district, which should definitely establish the facts and serve as a basis of calculations for future mining.

**HANDLING ORE IN MINES**

**METHODS OF HANDLING ORE**

No phase of mining is more important than the handling of ore; it is largely controlled by the occurrence of the ore, the dip of the ore bed being of first importance. The dip of the Big Seam varies widely along the outcrop and even more markedly in the direction of the dip. However, the average true dip of the ore bed is about 20°, presenting ideal conditions for handling ore in the stopes, the higher and lower dips requiring only slight modifications in the methods commonly employed.

The ore is usually mined in open stopes and moves by gravity from the face when broken by breast or overhand and underhand stoping to the level of the heading tracks, where it is loaded into the cars. If the bottom rock or footwall is smooth, little shoveling is required, particularly if stoping is correctly done and shots are carefully placed and fired. On fairly high dips ore moves to the track level more promptly and positively, and rough, uneven bottoms may have the advantage of retarding downward movement of the ore. When dips approach horizontal the ore must be handled by special methods.

Long, wide stopes present conditions for handling ore which materially change the methods used in the smaller and narrower stopes. In like manner the use of mechanical loaders in the mines has revolutionized ore mining and handling practice.

The actual handling of ore in the slopes is done by men and by mechanical loaders or scrapers. Loading ore into cars by mucking—that is, shoveling and handling the larger pieces of ore—is the common practice, although increasingly large tonnages are handled by mechanical loaders.

The handling of ore is in the transition period between hand and machine work. Human labor was fairly satisfactory and the supply abundant before the World War, but is now scarce, inefficient, and costly. All mining operations depend upon the loading of ore into cars and their delivery to the slopes; any slackening or lag means general slowing up and curtailment in the output, a condition reflected in higher cost of production.

Present mining conditions and the uncertain labor supply have stimulated the adoption of mechanical loading devices. The furnaces demand a steady output from the mines, and any material reduction
in the supply of ore may lead to serious complications. Machinery reduces human effort to a minimum and gives reasonable assurance of uniform production, but this advantage may not always be attained with decreased production costs. Operating expenses for mechanical loaders will undoubtedly be reduced until they approximate that of hand work, and will probably be much cheaper in time.

**HANDLING ORE IN SLOPES AND MANWAYS**

Differences in methods of handling ore in slopes and manways depend mainly upon the direction in which the passages are driven, that is, whether up or down the dip; slopes are usually driven up the dip or raised, while manways are driven with the dip or sunk. In slopes the ore moves away from the working face, and handling is materially facilitated thereby, whereas in manways gravity tends to hold the ore against the working face, thus hindering the advancement of the face.

The two general methods of procedure used in carrying forward the slopes and manways are shown in Figures 16 and 17. (See pp. 29 and 30.) In either method, at least one manway is carried in advance of the slope, the broken ore being transferred through the manways to the headings above, and thence to the main slope. The method illustrated by Figure 16 provides for the handling of all ore broken in the slope or manways on engine planes operating in the manways, while in the method shown in Figure 17, one manway may be driven by raising or sinking, when the ore broken in sinking is handled by engine plane and that broken by raising is transferred by gravity or scrapers to the crosscut and thence to the engine plane in the second manway. The ore broken in the manway in which the engine plane operates and the ore from the slope are handled by engine plane.

Engine planes operate in both manways, in the left manway from $e$ to $c$ and in right manway from $a$ to $b$, ore being transferred from $f$ to $g$ and through a crosscut to the engine plane $ab$. (See fig. 17, p. 30.) The ore is handled in metal cars with drop bottoms, operating on 42-inch-gauge tracks. The tracks are gradually elevated at the lower rib of the heading, where the ore is dumped into cars operating on the main heading tracks and transferred to the slope.

**CAR OR TRIP HAULAGE IN SLOPES**

There are two methods of handling ore on slopes in iron-ore mines—car (or trip) and skip haulage. When cars are operated on the slope tracks they are usually run in trains or trips of four to six cars, and an equal number of empty cars is returned to the mines on
the return trips. A head chainman rides both ways on the trips and has one or more helpers; however, more than two men seldom go to the surface on a loaded trip; the others leave it at some heading above the uppermost from which ore is loaded out and catch an empty trip coming in at the same point. The outgoing trips are commonly called "drags"; the empty trips are "slow drags," and are run to the surface when the men are going out at the end of shifts.

No ore bins or loading pockets are used with car or trip haulage on slopes, as it is practically impossible to spot or place small-capacity cars accurately enough to prevent undue spilling of ore on the tracks. As mine cars operate upon the heading and slope tracks in trip haulage the tracks must be of the same gauge. Frogs and switches are provided so that the cars can run from slope to heading tracks and from heading to slope tracks with equal ease, but to reduce danger of derailment to a minimum headings are usually staggered or offset, thus reducing the number of crossings one-half at any one point. This practice is further necessitated by the use of two sets of tracks in heading stations to provide track room for cars; the upper line of track receives the empty cars from the slope, while the lower track holds the loaded cars that are being collected for haulage up the slope. In certain smaller mines only one track is usually laid in the headings, simplifying track construction but making the handling of empty and loaded cars in the slopes difficult and troublesome.

Much difficulty is often experienced in placing empty trips and pulling loaded trips from headings on to the slope tracks, owing to the short radius of the track curves, although the chainmen become quite expert in getting derailed and loaded cars back on the tracks. Loaded cars are sometimes overturned, and the ore spilled on the tracks must be cleared away before hauling is resumed. All ore spilled on the slopes is removed from the tracks and shoveled to one side; it is loaded into cars at regular intervals, usually on Sundays, and sent to the top. Sometimes as much as 40 to 50 tons of ore is salvaged in a week. Overloading and trimming or racking of cars is largely responsible for the spilling of ore mentioned above. Ore is usually loaded out from below upward and thus the danger of accidents is greatly reduced.

Cars are joined with chains and clevices; the hoisting ropes are also attached to the cars by chains. The speeds of loaded trips are 800 to 2,000 feet per minute, averaging about 1,400 feet. The weight of rails on slopes for trip haulage is 60 pounds, on headings 30 to 40 pounds; the gauge of heading and slope tracks is 36 to 42 inches.
SKIP HAULAGE

When skips are used, ore is handled more efficiently and economically, there are fewer accidents and consequent delays, and little ore is ordinarily spilled on the tracks. Occasionally, owing to loss of control of gates in loading pockets and collapse of chutes, much ore may be dumped upon the slope tracks, but such accidents seldom happen.

Skips are built to hold 10 and 12 tons, but are seldom loaded to capacity, 7 or 8 tons being considered a fair load. The gauge of the skip tracks is 60 inches, and the rails weigh 60 to 90 pounds.

HAULAGE ROPE

Haulage ropes are $1\frac{1}{4}$ to $1\frac{1}{2}$ inches in diameter; rope speeds are 1,500 to 3,000 feet per minute, averaging about 2,300 feet. Because of changes in dip of the ore bed, with consequent variations of grade on slopes, which result in “pitches” and “flats,” the haulage ropes often alternately strike the top and bottom of the slopes and unless protected will rub on the track and top rock. The wear on haulage rope is often excessive; at one time it amounted to several cents per ton of ore handled, but the placing of rollers has reduced the loss to 1 cent or less per ton.

The wear on rope is included in the operating cost per ton of ore handled, but is emphasized to the casual observer by the grooves cut in top rock and foot of pillars. Long rollers at points on the roof of slopes where the ropes strike and rub and flanged rollers at frequent intervals on the bottoms of the slopes have done much to eliminate rope wear, but the rise and fall of the ropes as well as the lateral movement on long hauls make it virtually impossible to provide adequate protection; thus, in consequence, the haulage ropes are often rubbing on the supports of rollers, on the rails at crossings and switches, and worst of all on pieces of ore lying on the track bed. If tracks are well alined, loads heavy, and haulage engineers careful, much of the lateral movement of ropes may be avoided.

CHANGES NECESSITATED BY SKIP HAULAGE

With the adoption of skip haulage a radical change took place that necessitated much excavating, complete readjustment of methods, and the installation of new equipment. First, the level of the slope tracks had to be lowered to permit the loading of skips direct from the headings. To accomplish this the bottoms of the former slopes were lifted to a depth of 10 feet or more, the vertical distance between slope and heading tracks being at least 10 feet.

The station layout at the junction of headings and slopes consists of two lines of track extending 50 to 75 feet back into the headings
and provided with crossovers in order that empty and loaded cars may be shifted from one track to the other as occasion demands; the main purpose, however, is to expedite loading skips by handling loaded cars promptly. Directly over the main slope track and on a level with the headings is a heavy platform with large rectangular openings, through which the ore is discharged to the skip below. Each heading track terminates in a “horned” or “gooseneck” tipple, into which the loaded cars are pushed and by which they are dumped, the counterbalancing of the tipple returning it and the empty car to the horizontal. By this arrangement four loaded cars can be dumped at the same time, the empty cars being shifted on the crossovers and other loaded cars brought forward.

In the most recent practice at a number of the larger mines, particularly where mechanical loaders or scrapers are employed for loading cars in the stopes, 5-ton all-steel cars are operated in the headings on standard-gauge tracks (4 feet 8½ inches). Owing to the greater weight of a car and its contained load, air-operated tipples replace the hand-operated type, the air cylinders for operating the tipples being placed directly below the tipples on the slope level and protected by the heading-station platforms overhead. When 5-ton heading cars are used only two tipples are employed, one on either side of the slopes; however, a second track is provided and maintained with crossovers for shifting the cars from one side to the other of the slopes and for the accommodation of motors.

Chutes and guide boards extend from the lower side of the loading platform to within a few inches of the top of the skip to prevent spillage of ore in loading. Loading stations for skips at or near faults with 10 feet or more displacement may require special arrangement of platforms and tipples on the opposite sides of the slopes. The heading levels may be much higher on one side due to faulting, necessitating much longer chutes for the safe transfer of ore, while on the other side the chutes may be very short; however, the amount of excavation required and the arrangement for handling ore depend entirely upon the position of the slope, which in turn is definitely established by the amount of displacement of the ore bed.

**ROTARY DUMPS**

When slopes are definitely terminated by reaching the bottoms of basins in the ore bed, the dip flattening out or being reversed, the slope is usually extended into the underlying rock formations, and an ore bin of several hundred tons capacity constructed as a loading pocket for the skip. For prompt handling of cars rotary dumps
into which cars may be run singly or in groups of four or more and dumped are installed above the pockets. In a number of installations full trips of cars are run into the dumps without being detached from the haulage rope, dumped, and then returned to the mine, the entire operation taking only a few minutes.

At one mine a rotary dump is situated on the slope beside the foot of a vertical shaft through which all ore is hoisted to the surface, the ore being discharged into an ore pocket and thence into the skip operating in the shaft. Trips of cars are drawn up the slope into the dump, and when emptied and righted are returned to the workings without being detached from the hoisting rope. (See fig. 64.)

**MOTOR HAULAGE**

When the lay of the ore bed makes possible the employment of motor haulage, or when it may be desirable for efficiency and economy to drive rock tunnels connecting various parts of the workings that are difficult to reach by ordinary means, electric motors are installed. Expeditious handling of heavy trips of loaded cars and their prompt return to working places renders their use when possible practically indispensable. (See fig. 65.)

**HANDLING ORE IN STOPES**

**CAR HAULAGE**

Ore broken down at the face of the stopes is brought to the level of the heading track, where it is reduced by sledge or by explosives and shoveled into cars for transfer to the slopes. The ore is of such a type that relatively large masses can be broken up by well-directed blows of a sledge, and the miners acquire great skill in
using it. (See fig. 66.) The larger masses often weigh several hundred pounds, and are broken by "blockholing"; that is, by small charges of powder (one-fourth to one-half stick) in short holes drilled by jackhammer drills. If the masses are not actually broken by blockholing, they are so weakened that they are easily reduced by sledge.

**TRIMMING AND RACKING CARS**

The ore is loaded into cars by lifting the larger pieces by hand and shoveling finer ore that ranges in size from 4 to 6 inches to fines. When long hauls are necessary and only a limited supply of cars is available it is the practice to "trim" or "rack" the cars; that is, to load them above the sides as high as possible. Often a loose wall of large pieces of ore must be built around the top of the car to hold in the smaller pieces of ore. Trimming or racking ore cars can not
be said to be good practice, as it takes much time; however, the main objection is the extra effort required to lift the ore. (See fig. 67.)

The usual height of 2-ton wooden mine cars is 40 inches. To place a heavy piece of ore in such a car two moves are needed, the first above the knees, the second to the car top. Racking cars means elevating the ore still more in order to deposit it within the built-up ledge of ore. The height of racked cars makes them more unstable and consequently more difficult to handle on more or less irregular mine tracks. Loaded untrimmed cars have a capacity of 2 tons, and the same cars trimmed hold about 2½ tons.

To overcome the disadvantages of trimming and racking there is now being quite extensively used an all-steel car that is 32 inches high, but has a track gauge of 42 instead of 36 inches, the gauge for the higher wooden cars. The capacity of the lower cars is also 2 tons, the loss in capacity through reduced height being made up by increased width and length. The gauge of the recently installed 5-ton steel cars is standard, 4 feet 8½ inches. These cars are used in connection with the mechanical loaders.

**EFFECT OF MINING METHOD**

The mining method in use may facilitate transference of ore from the stopes to the heading tracks, where it is loaded into cars; when the ore bed dips slightly overhand stoping is preferable, but with higher dips underhand stoping is usually employed. (See figs. 23 and 24, pp. 36 and 37.)

**TOP AND BOTTOM ROCK**

The character and condition of the top and bottom rock also affect loading, because much hand picking may be required if the rock breaks and mixes with the ore. Waste rock is either hauled out of
the stopes or built into pack walls along the heading tracks where large quantities are stowed. (See fig. 57, p. 76.) The breaking down of draw slate is the source of much waste, but a no less important source is the slate parting between the upper and lower benches, which under the heaving of the footwall breaks and mixes with the ore. The labor required to break the waste rock into suitable sizes for handling and loading into cars and to stow it may constitute a large part of the expense incident to the handling of ore.

LOADING DATA

Under normal conditions a man will handle 10 to 15 tons of ore per 8-hour shift, three muckers in a stope handling 35 to 45 tons per shift. The muckers who handle the ore also tram the loaded cars to the slope station, the empty cars being returned by mule. On the basis of an 8-hour shift, the time usually consumed in actual handling ore by hand and shovel, or mucking, is 4 hours, and in tramming 2 hours, the remaining 2 hours being consumed by delays in handling cars, unloading, and other operations. The shovels used are square nosed, weigh about $5\frac{1}{2}$ pounds, and are $3\frac{1}{2}$ feet long; a shovel load averages about 20 pounds.

FILLING TRACKS

When headings are being driven and stoping done the ore is cut to hanging and foot walls, making a V-shaped cut at the bottom and next to the lower rib, which on high dips is pronounced. In order that the heading tracks may be laid along the line of the lower ribs much filling must be done to bring the tracks to the proper level and grade. Both ore and waste rock are used as ballast, but ordinarily ore is more commonly used; consequently large amounts of ore remain in the track beds after stopes are abandoned. In certain mines this ore is removed when the heading tracks are taken up; the amount of broken ore left in the mine is thus reduced.

TRANSFER OF CARS TO LOADING STATIONS

The loaded cars are transferred by gravity to the loading stations at the slopes in twos and threes, the shoveler riding them, and one at least sitting on a long lever that rests on the tread of one of the back wheels of the rear car and regulates the speed like a brake. Mules haul one or two cars at a time back to the working faces of the stopes. Hand tramming is occasionally done, but is only permissible for short distances and temporary work.

MOTOR HAULAGE

Although electric haulage is not applicable to heading work owing to the restricted range of operations, a motor might prove a good in-
vestment if it could be made to serve a number of headings by transference on a slope. Motor haulage is now confined to the haulage of ore from more or less distant points, such as between areas worked by separate slopes, which are connected by crosscuts in the rock and driven on a grade slightly in favor of the loaded cars. Motor haulage is occasionally used in practically horizontal areas, such as at the bottom of synclinal troughs, connection being made if necessary by rock cuts to the slopes or main haulageways. (See fig. 65, p. 91.)

HAULAGE IN LARGE STOPES

The usual procedure in handling cars, loaded and empty, on heading tracks and in stopes of ordinary width and length has been given in the foregoing paragraphs. The adoption of greater widths and lengths in working stopes has in turn necessitated more or less radical changes, modifications at least, in handling cars in the stopes; descriptions of the methods used in connection with methods of mining outlined before are given below.

With increased heading intervals and correspondingly wider pillars, the working of the pillars by large upsets materially increased the distance that the ore had to be handled in getting it to the heading track. Where the dip of the ore bed permitted, branch tracks were usually run from the main-heading tracks diagonally up the stope at least to the entrance of the upset. When dips are low and moderate the branch tracks are placed with little or no difficulty, but when dips are high and moderately high long branch tracks are required, the cost of which may be prohibitive. (See fig. 27, p. 40.)

HANDLING ORE BY CHUTES

When the distances through which ore had to be handled became too great and the stopes too narrow for advantageous use of branch tracks, shaking conveyors were employed and for a time extensively used.

Shaking chutes were usually employed on dips higher than those where cars were hauled by mules, but not as high as those where gravity planes operate. The sections of the metal chutes were 5 feet long and 18 inches wide, with round or flat bottoms having sloping sides; flanges at the ends made it convenient to fasten the sections with slotted lugs and keys. The chutes were suspended from rods or rods and chains, the upper ends of the suspension members being attached to eyebolts set in the roof or in props placed in rows up the stopes to the working face. It was found that when supports were long the chutes tended to swing too much, consequently the latter method of support was preferred. The slope given the chutes was
made adjustable by the suspension rods having hooks at the ends that were hooked into the links of short sections of chains. Where it was inconvenient to place props close enough together to accommodate the support of chutes, crosspieces were run between props and the supports attached to them. The chutes were actuated by the steam end of a No. 4 Cameron pump, the connecting rod being fastened to the last and lowest section of the chute. The rate of movement was about one stroke per second or 50 to 60 strokes per minute.

Shaking chutes have been abandoned and, like other methods of handling ore, have been replaced by equipment better suited to the handling of large tonnages under wide ranges of dip and greatly increased width and length of stopes.

**HANDLING ORE BY PLANES**

Where the heading interval reaches 80 feet, branch tracks may still be employed, but it is more usual to employ gravity planes, thus permitting the tracks to be laid directly up the stopes and taking full advantage of the dip.

To transfer empty cars from the heading tracks to the working faces and the loaded cars returned to the heading tracks, two lines of track are laid in the stopes, usually close together to begin with when the stopes are short, but often widely separated as stoping proceeds and the stopes lengthen. (See fig. 68.)

Cars operate on both tracks, empties being delivered to the working faces and loaded cars returned to the heading tracks. As the tracks are different distances apart, two pulleys must be provided in line with the center of each; one-half to five-eighths inch steel ropes pass over the pulleys to the cars operating on the tracks. The pulleys are inclosed in strap-iron clevices which are attached to iron pins or stubbing bars set into drill holes in the bottom rock.

At some intermediate point between the two pulleys is a brake by which the movement of the cars may be controlled. The brake is
usually placed near one of the pulleys in order that the lateral movement of the rope may be slight. It consists of three members; the two side numbers are round or sawed 4 by 6 inch timbers bound together and spaced 4 to 6 inches apart by 2½ by ½ inch strap iron placed close to the ends. A much longer piece of timber similar in size to the side pieces serves as a lever and is hinged to the side members by a bolt passing through all three parts, the lever operating between the side pieces. This combination of side pieces and lever is so placed that the rope passing over the two pulleys crosses the side pieces and lies between them and the lever, and can be firmly gripped by forcing down the lever. The whole structure is held rigidly in place on the stope floor by four iron pins set in drill holes placed near the ends of the side pieces.

The balanced or gravity planes have been used extensively in certain ore mines on Red Mountain, owing to the elasticity of the system, but are being replaced by mechanical loaders or scrapers.

Gravity planes may readily be adapted to use in stopes, where horses of rock occur or where it has been found desirable to leave pillars of ore, by a diverting pulley. Interference with the operation of planes by falls of rock may also be overcome by the use of a third or diverting pulley. Balanced planes may be employed in large stopes as shown in Figure 29 (see p. 41), the disadvantage, if any, being that the cars operating on the left track draw from the horizontal working face alone.

**HANDLING ORE BY MECHANICAL LOADERS OR SCRAPERS**

With higher dips the ore may be transferred several hundred feet or through several stopes by gravity alone, but the stopes must then be driven in line along the dip in order that the ore may move freely and unimpeded. In such work heading tracks are maintained at certain levels only, the intermediate tracks being abandoned. The success of this method is probably largely responsible for the development of large stopes in which mechanical loaders are employed and by which means it is possible to operate on moderate dips.

Stopes 100 by 200 feet are now being worked thus, giving heading intervals of 150 to 200 feet, which will undoubtedly be increased in localities where conditions of top rock will permit. Although the method of using scrapers for loading ore is practically the same in the various mines, the layout may be different, especially as to the size of stopes and the means of access to them. Workings in which scrapers are used may be designated as single or multiple passage or neck stopes, no limit being set for the number of openings that may finally be employed.
SINGLE-NECK STOPES

In a single-neck stope, all ore mined in a stope is discharged through one opening, through which the drag lines and scraper also operate. By shifting the drag-line pulley support along the face of the stope $ab$, Figure 31 (p. 44), all of the ore broken down in the stope can be reached. Two lines of car track are maintained in the heading and preliminary stope; a motor car with electrically driven drums for operating the drag line and moving the cars stands upon the track next to the lower rib, while the ore cars are handled on the track next to the stope. The scrapers move the ore from the stopes to and over apron chutes placed at the necks of the stopes, and discharge it into the mine cars. The loaded cars are run in trips of four or more to the skip loading stations at the slopes and are returned by ropes attached to a drum on the motor car. The

![Diagram of a single-neck stope](image)

**Figure 69.—Heading for scraper loaders**

full trip of cars is thus moved both ways as a unit, instead of the cars being returned singly as when hand or mule tramming is done.

As the mine cars are handled faster than when hand tramming is done and as heavier loads, including the motor car, are moved, it is both desirable and necessary that greater care be exercised in the construction of heading tracks, particularly with respect to the weight of rail, blasting, and grade. Where headings are driven for scraper work it is usually the practice to remove all of the ore in the headings to the line of the pillars, then to place the tracks at the proper level with respect to the loading chutes by cutting out the footwall and filling in the stopes next the lower pillar or rib. Although the relative amounts of excavation and filling vary somewhat, they are usually about equal. However, the amount of excavation and filling depends upon the dip of the ore bed. (See fig. 69.)
MULTIPLE-NECK STOPES

When multiple-neck stopes are worked the practice is similar to that given above, but greater width of stopes is possible, the motor cars and equipment being advanced from neck to neck as the stopes are successively widened by the cutting-out stoping operations that are carried on laterally.

HEADINGS AND PRELIMINARY STOPES

Scrapers are also employed to load out ore broken down in the headings and preliminary stopes, but the operating method differs from that employed in the stopes because the ore must be elevated to and above the car-top level. A long inclined chute of low grade is provided under which the mine cars are run and from which they receive the ore. The scrapers take their loads at the working face, which is practically horizontal, and convey it to and over the chute to the cars; their movement is controlled by a drum on the motor car stationed opposite a neck connecting with the stopes above. The motor car evidently has three functions, to operate the drag-line scrapers in the stopes, to move the empty and loaded cars between the skip loading stations at the stopes and the stope necks, and to operate scrapers in the headings.

LIMITATIONS OF SCRAPERS

Although mechanical loaders or scrapers are comparatively new in the Red Mountain ore mines, the degree of success attained in their operation has justified their continued application and their use will undoubtedly increase in future; however, they have not yet been applied to all conditions of dip and of top formations and they may be found to have definite limitations involving changes in method of mining and support. An important feature of the use of mechanical loaders is the reduction in amount and cost of development work. Furthermore, the output from a stope may be increased two to three times over that for hand work.

HANDLING ORE BETWEEN STOPES AND HEADINGS

The next step in the handling of ore is its delivery from the stopes to the heading tracks; that is, in cross headings and on planes.

In early attempts to mine lower bench ore short lines of branch tracks were run from the main-heading tracks, but beyond a certain distance these tracks had to be abandoned because grades were too steep; in consequence only a very small part of the available ore was obtained. (See fig. 32, p. 46.) Later, cross headings were employed both in robbing and in working the lower bench, especially the latter.
CROSS-HEADING TRACKS

When employed in working the lower bench, the cross-heading tracks are turned off the main-heading tracks at such points along the headings as seem desirable, and are run at grades that will permit cars to be operated, if the dip allows, along the middle line of the next stopes above, but on the footwall of the Big Seam. (See fig. 33, p. 47.) On high or moderately high dips the cross-heading tracks may not extend to the middle of the stopes above, but are placed on the footwall immediately below the former heading tracks. On steep dips cross headings obviously have limited application, but on an average dip of ore bed they can be employed to advantage.

Cross headings are widely used incident to the robbing of pillars; when thus used they seldom extend beyond the headings next above, but reach and continue along the center line of the pillars. (See figs. 39 and 42, pp. 33 and 55.) By the use of cross headings the larger part of the arch pillars can be removed, and only small remnants of the original pillars are left, formerly the upper and lower ribs. As this is the final operation in mining, the work is often carried beyond safe limits and caving may start before all the ore can be removed. It is not desirable, therefore, to extend the cross-heading tracks too far, but rather to run a new line every 200 or 300 feet along the stopes. Furthermore, the cross headings may be turned off the main-heading tracks either to the right or left, the grade of the tracks being readily established as the cross headings are advanced. (See fig. 70.)

It is only one step from the use of cross headings in robbing pillars to their application to mining the lower bench of the Big Seam and at the same time robbing pillars in both the upper and lower benches. In the latter work the cross-heading tracks may be ex-
tended in areas of low dip to the center line of the arch pillars in the next stope above the main-heading track from which it was turned, but when dips are high the grade is also too high; then the first arch pillar encountered and cut by the cross headings is split. The working of the lower bench and the robbing of pillars in the same lift are shown in Figure 42, page 55. (See also fig. 33, p. 47, for application to lower dips.)

Cross-heading tracks are often extensively used in reducing the size of pillars by slabbing from the lower side, which work may be extended with small heading intervals to the removal of practically all of the pillar. When this is done the cross-heading tracks are extended to and along the lower side of the arch pillar, if the dip of the ore bed will permit.

A special use of cross headings in robbing pillars may be observed in old stopes formerly worked by wide upsets that cut the pillars into long narrow columns extending from the top to the bottom of the slopes. Here the cross headings are turned off the main headings at moderate angles and driven diagonally across the stopes, breaking the pillars up into stubs. The cross headings may be run as close as 35 to 50 feet apart, and to maintain the grade the lower bench may be removed to a depth of 4 to 6 feet. (See fig. 71.)

**Switchbacks**

The use of switchbacks in splitting pillars has already been described, but their special application to steep and varying dips should be mentioned. In localities where the ore bed has been more or less disturbed by folding and counterfolding, resulting in high dips and wide ranges of strike, also where numerous fault planes, with relatively small displacements, cut the ore bed, switchbacks are most widely used as a means of reaching and handling the ore.
Often a cross heading is run at the highest permissible grade and switchback tracks are turned from it, thus permitting loaded cars to be transferred easily and safely from the more inaccessible areas to the line of a main heading below. (See fig. 41, p. 54.) Although switchback tracks are simple to operate much care is often involved in their layout, and particularly in filling and grading with timber barriers and waste rock. The so-called switchback tracks are really nothing more than a combination of cross headings, all connected but run in different directions.

**GRADES FOR TRACKS**

Grades for tracks in stopes, including headings, cross headings, shaking chutes, and balanced or gravity planes are approximately as follows: Headings, 2¹⁄₂ to 4 per cent; cross headings, 4 to 12 and as high as 15 per cent; shaking chutes, 8 to 12 per cent; and balanced planes, 10 to 35 per cent. Ore may be readily transferred by hand in moderately smooth stope floors from the upper to the lower rib on dips of 15 to 30 per cent, particularly when care is taken in placing shots, in the amount of powder used, and in order of firing.

**HILL TRACKS**

Cross-heading tracks and all others turned off the main-heading tracks are often called “hill tracks” and are run at such grades that loaded cars can operate upon them solely under the control of hand brakes. Hill tracks usually have fairly steep grades. (See fig. 72.)

**REMOVAL OF ORE ON STOPE FLOORS**

The movement of ore on stope floors is materially facilitated by mechanical means such as scrapers. By their use ore can be handled
on stope floors too rough and uneven for satisfactory handwork, as some must be, because great variations in the condition of stope bottoms will be encountered as the use of mechanical loaders is extended.

**A-HEADINGS AS A MEANS OF HANDLING ORE**

Although A-headings were used at one time rather widely as a means of handling ore in stopes with moderately large heading intervals, they are no longer in use.

Subheadings, driven from the manways about midway between headings and parallel with them, traversed the stopes and cut the pillars, dividing the relatively large ones into two parts, which were then attacked and greatly reduced in size by slabbing until small pillars remained which were spaced some distance apart and symmetrically arranged in pairs in the stopes. (See fig. 40, p. 53.) The ore removed in driving the A-headings, with that slabbed off the pillars later, was loaded into small—usually 1-ton—cars and delivered to certain points where it was dumped and conveyed by gravity on the stope floor or by chutes to the headings below. The tracks in the A-headings were of slight grade and usually of moderate length, but they did not prove altogether efficient or economical and were abandoned.

**DRIVING HAULAGeways**

Driving haulageways to any given grade in the ore mines is not as difficult as it might appear, and as it would be with other occurrences of ore. The section on “Character and occurrence of ore” stated that throughout much of the field the parting between the upper and lower benches of the Big Seam is slight, in certain localities amounting to nothing more than paper-thin separation of the ore. Furthermore, the upper part of the lower bench may often be a fair grade of ore, as good, in fact, as much of the upper bench. In driving to grade it is unnecessary to confine the work to the upper bench, but excavations may be extended into the lower bench as far as required, which would not be desirable or possible if the footwall was hard rock. In other localities, where the parting between the upper and lower benches attains a thickness of 8 to 10 inches and occasionally 24 to 28 inches, removal may sometimes be found desirable, but as it is usually broken by shooting and the heaving of the bottom, little or no difficulty is experienced in making the necessary excavations.

When it is desired to run a track along the upper rib in stopes when robbing is begun and a large amount of waste rock is available, track beds may be built up to the proper grade, although longer
lines are required. In fact, the whole stope floor may be raised several feet, permitting tracks to be laid and cars run virtually on the horizontal at any point across the full width of the stope. This practice means an extensive handling of waste rock that is hardly warrantable, even with a large amount of waste rock available.

**EFFECT OF VARIATION IN DIP OF BED ON HANDLING OF ORE**

Secondary folds sometimes occur in the ore bed, warping it upward or downward, the ore bed in consequence either rising above or falling below the line of inclination of the parts lying above and below the fold. Several methods are employed to handle ore economically and efficiently in the development of such irregular areas.

Slopes and manways may be driven through the rock, irrespective of whether the folds are anticlinal or synclinal, and connection made with those parts of the ore bed that lie beyond the folds. By this procedure the handling of ore on the main slopes is not hampered, and only slight changes in grade result unless faulting has occurred with the folding.

**USE OF ENGINE OR GRAVITY PLANES**

On either anticlinal or synclinal folds, engine or gravity planes are commonly employed to handle the ore mined in the disturbed areas. When the folds are anticlinal and the elevated parts or "hills" have almost equal slopes on both flanks, engines at the crests of the hills pull the loaded cars up the inclines one at a time, and without stopping or being disconnected from the haulage rope the cars move on and down the opposite side of the elevations; deflecting pulleys above the tracks make possible efficient and expeditious handling of cars. When, as occasionally happens, one flank of the elevated part is steep, the ore mined in the successive headings may be run down the stope floors by gravity to the loading stations, the cars running directly to the main slopes. Gravity planes may also be employed, which will transfer the ore down either side of the elevations to headings connecting with the main slopes. When depressions or troughs occur in the ore bed, the method employed may depend largely upon the inclinations, but the usual practice is to use engine planes, which raise the ore in cars to the level of the main slopes.

**ROCK TUNNELS**

Rock tunnels may be employed to connect the ore bed on either side of wide troughs, or the main slopes may follow the ore bed to the bottom of the troughs, terminating there. In the latter circumstance
car haulage is employed in rock cuts above the slopes, where ore pockets are located with dumps placed above them, the ore from the pockets being discharged into skips operating on the main-slope tracks below.

CONNECTING SEPARATED PARTS OF ORE BED

The handling of ore under conditions that result from faulting of the ore bed may, in a general way, be covered by the methods employed in anticlinal and synclinal folds, which have been referred to in the foregoing paragraphs. Connecting the ore bed above fault planes with the displaced parts below the faults eliminates the slopes as means of handling ore direct from the headings, thus entailing the use of engine and gravity planes to transfer ore from stopes to the first headings that make connections with the slopes. (See fig. 73.) In the section two methods of connecting the parts of an ore bed separated by a fault are shown, the rock
slope \( ad \) being the most common method of procedure with large displacements, while the rock slope \( ec \) is probably the most satisfactory method with smaller displacements. Under the former conditions the part of the ore bed lying between \( a \) and \( b \) is usually worked by headings driven off the main slope, but should the rock slope be driven and in operation before all of the ore is mined, an engine plane would be established at \( h \) and ore raised from \( i \) and \( p \) to \( h \) and handled on heading \( k \) to the main slope at \( n \). Similar practice would be followed in the section between \( c \) and \( d \), except that gravity planes would transfer the ore from \( j \) and \( o \) to \( m \) and thence by heading to the main slope at \( n \). If the rock slope \( ec \) was used, all ore would be handled directly on the main slope except that part that might be unworked after the rock slope was completed, or main-slope haulage might be maintained in the section \( ab \) by the use of a switch track (see fig. 74), thus permitting cars or skips to be used with equal facility on the main slope and on the track in section \( ab \).

**Figure 74.**—Vertical switch track in split slope

**COMBINED SKIP AND TRIP HAULAGE**

Two slopes can also be used as the main haulageways, one of the slopes handling cars in trips delivering ore to an ore pocket which in turn discharges in skips operating on the main slope. (See fig. 75.) From \( a \) and \( b \) the main slope follows the ore bed and from \( b \) to \( c \) is a rock cut; the ore bed continues along the line \( bfd \). An ore pocket, \( e \), is cut in the footwall of the ore bed, which receives ore from cars operating in \( hd \) through a rotary dump above the pocket at \( f \). An electrically driven hoist for slope \( hd \), situated at \( g \), handles
all ore mined in the extension of the ore bed beyond the trough abf. In some respects this occurrence of ore and method of handling is similar to the development and working of synclinal troughs mentioned before, but illustrates the more usual practice of following the ore bed with the slopes to the lowest point rather than the employment of rock tunnels for maintaining the slopes on fairly uniform grades between the upper parts of the bed. The absence of definite information on the presence of faults and other irregularities may be largely responsible for the practice of continuing slopes to fault planes and bottoms of troughs, although economic conditions probably have the most influence. In general, then, it may be said that between the working faces in the stopes, as on headings and other passages connecting with the main slopes, all ore is handled by direct tramming or haulage, or by engine or gravity planes. Much ingenuity has been shown in meeting conditions arising from the occurrence of folds and faults, but the added expense of special installations is no small item in the cost of operating the mines, and with increased complications in methods of handling ore, production must naturally slow down and lag.

MULE HAULAGE IN MINES

As mentioned before, empty cars are hauled to the stope face by mules and the loaded cars returned to the slope stations by gravity. A mule pulls one car on tracks with 2½ to 4 per cent grade, and for short hauls may negotiate grades as high as 8 to 12 per cent. On the usual heading tracks of moderate grade a mule can pull two cars, but the most common practice is probably one car per trip.
CARE OF MULES

In the smaller mines the mules are usually taken in and out of the mines daily. The mules travel the manways with the men, and where considerable water is present, particularly near the surface, the footing is usually bad. In deeper mines the mules are kept underground in stables.

Stables are built in old stopes near the slopes to insure adequate ventilation and the removal of obnoxious odors from the workings. Many stables are well constructed and equipped with such conveniences as feed troughs for hay and grain, running water, plank floors, pipe partitions, good drainage, and electric lights, and kept sanitary by weekly flushing. If mules are well cared for when not at work they can give good service, which is not always possible when they are removed from the mines daily.

POWER-DRIVEN EQUIPMENT FOR HANDLING ORE

The haulage equipment for ore mines is extensive as to types and makes; the wide range of equipment found in the mines where conditions are similar is largely due to the method of haulage employed; that is, whether motor haulage is used in the mines or trip or skip haulage on the slopes. Motor haulage in slopes and stopes has already been briefly discussed. Eight-ton General Electric motors are used, the motor-generator sets being in the mines. Skip haulage requires more powerful engines operating at higher speeds than can be employed with trip haulage. Folds and faults may necessitate various types of hauling equipment which are driven by compressed air or electricity, the former being preferable as an aid to ventilation and the latter more convenient.

Hoists used in main slopes are direct acting, geared with single and double cylindrical drums 8 to 10 feet in diameter, and provided with friction clutches of the axial plate and rim types. They are driven by steam and electricity, requiring from 700 to 1,800 horse-power engines or motors. Slide and Corliss valve engines are used in the steam-driven installations, while with electric installations alternating-current motors are employed, and geared to the drums; 44,000-volt, 3-phase, 60-cycle current is supplied to the mines, and is stepped down to meet the consumers' requirements. The principal makes of hoists used in the districts are Nordberg, Wellman-Seaver, Morgan, Hardie-Tynes, and Ottumwa Iron Works.

Ore is usually handled on short or auxiliary slopes by single-drum hoists driven by compressed air and electricity. The rope is ordinarily one-half to five-eighths inch in diameter.
SURFACE HANDLING OF ORES

The handling of ore on the surface is necessarily controlled by the methods employed underground, trip haulage requiring one type of tipple, while skip haulage requires an entirely different form. With car or trip haulage the cars run from the slope to the rock house or tipple tracks, the elevation of which permits the ore to fall into a crusher and thence by chute to the railroad cars. The
elevation of the tracks for the incoming and outgoing cars is such that they are handled largely by gravity, thus reducing the amount of hand work done upon them in dumping and shifting to the different tracks. The rock houses or tipples for trip haulage are long and comparatively low to permit the trips of cars to run upon them from the slopes and be disconnected from the haulage rope. (See fig. 76.) The skips operate on tracks that are supported above the ore pockets and are part of the rock house proper. The grade of the skip tracks on the rock houses is practically the same as that of the slopes, although it is usually maintained at an angle of about 16°. (See figs. 77 and 78.)

![Figure 78.—Skip and dumping rails above ore bins at rock house or tipple](image)

Ore dumped from the skips is received by a pocket which feeds to crushers and thence by chute to railroad cars. As the rock houses are usually built on fairly steeply sloping hillsides, ore is readily moved through the pockets and crushers without increased elevation. Pockets for waste rock raised in the regular skips or in special bottom-dumping sinking skips are usually placed ahead of the ore pockets. Cars, operating on tracks laid on the ground directly below the pocket, run to dumps along the mountain side some distance from the rock houses. Some of the more recently built rock houses have been so constructed that ore and waste can be rapidly and economically handled at the same level, thus making possible haulage of cars by locomotives instead of by hand.
Gyratory crushers are universally used, Nos. 7½, 8, and 10 being in most common use. The standard sizes of receiving openings, sizes to which ore is crushed, and capacities are as follows:

<table>
<thead>
<tr>
<th>Number</th>
<th>Opening</th>
<th>Size of product</th>
<th>Capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td>7½</td>
<td>15 by 56</td>
<td>2½ to 5</td>
<td>80 to 180</td>
</tr>
<tr>
<td>8</td>
<td>18 by 68</td>
<td>2½ to 6½</td>
<td>110 to 250</td>
</tr>
<tr>
<td>10</td>
<td>24 by 84</td>
<td>3½ to 6½</td>
<td>210 to 450</td>
</tr>
</tbody>
</table>

From 50 to 150 horsepower is required to operate these crushers; it is supplied by alternating-current motors.

**DRAINAGE**

**CONDITIONS AFFECTING DRAINAGE**

Under normal conditions the amount of water entering the underground workings of the ore mines on Red Mountain is small. The ore is hard, dense, and remarkably free from solution cavities and other inlets, such as watercourses, connecting the surface or other sources of water supply with the workings. Although natural conditions favor the exclusion of water from the mines, increasing amounts enter the underground workings from year to year and will ultimately become a serious menace to mining.

No particular location or area of minable ore can be said to be situated more disadvantageously than other areas with respect to wetness. However, the relation of the ore bed to the elevation of the ground probably has some effect upon the influx of water, as the ground-water level is affected thereby. The numerous gaps in Red Mountain cut porous strata that occur above the Big Seam, and serve as outlets for the water that accumulates in the strata and workings above the level of such outlets. Although the level of the ground water depends upon the elevation of the outlets and the distance between them, the resistance of the intermediate ground to the movement of water through its formations tends to maintain the level of ground water above that of the outlets. The level of outlets roughly establishes the zone of ground water, and corresponds in unmined ground to the zone of soft or leached ores. Along the Big Seam on Red Mountain, this zone extends down the dip an average distance of 300 feet. Although little authentic information is available regarding the limits in distance of ground water from the surface, available data indicate that ground water probably extends 800 to 2,000 feet, the distance varying according to the condition of formations and the lay of the ground.
SOURCES OF MINE WATER

Water enters a mine in the following ways:
1. Surface or meteoric water enters at the outcrop or through breaks reaching to the surface as a result of falls of top formations.
2. Water comes from porous formations that are tapped by subsidence cracks.
3. Water enters through fault planes and other lines of weakness, such as slips and solution cavities.
4. Water enters through exploratory drill holes.

SURFACE WATER

Much water seeps into the mines through open cuts and caved areas along the line of outcrop. The amount of rainfall largely determines the influx from this source, although precautionary measures may reduce the amount very considerably.

WATER FROM POROUS FORMATIONS

The individual members of the geological formations overlying the Big Seam are usually strong, tough, and impervious to water, but one or more formations are, however, by far the most important source of large quantities of water. The Fort Payne chert bed is the principal water-bearing stratum and lies immediately above the Clinton formation. Directly above the chert bed is the Hartselle sandstone, which, while contributing to the source of mine water, may be of minor importance compared with the Fort Payne chert.

The chert bed has a high lime content that is soluble, and it has been extensively leached by percolating waters; watercourses and reservoirs of great capacity have thus been formed. Although the extent of the leached zone is not definitely known, it follows the crest of Red Mountain for the entire distance through the mining district, and ranges in width from several hundred to two thousand feet or more.

The contribution of the Hartselle sandstone to the reservoir capacity is known much less definitely, but is probably not inconsiderable. Unless otherwise disturbed and broken the series of impervious slate beds inclosing the water-bearing strata effectively seal off the water and prevent its entry into the mine workings. Any movement of the rock strata that weakens or breaks the inclosing impervious beds at once makes it possible for water to reach the workings in varying amounts, depending upon the degree of the rupture. The inclosing slates must therefore be largely relied upon to protect the mines from influx of water. How the stability of the
water-bearing beds can be maintained is a question that is receiving the serious consideration of the mine operators and demands prompt and decisive action if the mines are to be worked economically and efficiently.

WATER ENTERING THROUGH FAULT PLANES

When the number of faults that intersect the ore beds is considered, it is evident that the ultimate drainage of the workings would be very difficult if they admitted water to the mines readily and the reserve of water was large. Fortunately, however, the known fault planes are relatively tight and seldom admit much water. This condition of affairs is particularly observed in faults that occur some distance from the outcrop, or where the weight of the cover is enough to keep all cracks and fissures tightly closed. When, however, one or more minor faults occur very near a major fault plane, even though the amount of displacement of the beds is small, the strata have often become so weakened that large amounts of percolating waters find their way into the workings along the disturbed and faulted zone. (See fig. 53, p. 72.) Fault planes may also be open enough to admit into the workings large amounts of water, which are very difficult, if not impossible, to control effectively.

Slips may be responsible for the admission of larger quantities of water, although the amount coming from individual slips may be slight. Slips that bear much water and consequently show deposits of lime and clay are called "mud slips." (See fig. 48, p. 67.)

Cavities of considerable width and length produced by solution along slips are not uncommon. Moreover, solution cavities or caverns of large cross-sectional dimensions and lengths are sometimes encountered. Occasionally such caverns are filled with water and when broken into by mining operations this water may seriously affect the work until drained off.

WATER ENTERING THROUGH PROSPECT HOLES

Much difficulty has been experienced from water that enters the underground workings through prospect drill holes; the water will be under high pressure and may also have large volume, thus making direction and control doubly difficult. Failure to mark the position of holes or losing track of such locations often makes it difficult to guard against mining operations breaking into old drill holes.

PREVENTING ENTRANCE OF WATER

Careful investigation of the sources of mine water shows that preventing the ingress of water is fully as important as removing water that has gained ingress, and the means employed to safeguard
workings against the entrance of water are fully as much a part of
dewatering or drainage as the ejection of water from the workings.

The simplest phase of this problem is to prevent the entrance of
surface water, which ordinarily is readily done by digging inter-
cepting ditches to divert the water from the outcrop and from breaks
in the surface after top formations are caved. In addition, crevices
and fissures resulting from subsidence of the surface are filled with
well-tamped dirt, which may halt the ingress of water temporarily,
but will shift when further movement occurs.

Placing intercepting ditches is usually comparatively simple, but
when, as is often true, the surface or open-cut workings are large,
with numerous breaks, intercepting ditches may be difficult to place
and inadequate for protection.

Intercepting and diverting ditches are widely used to handle
surface water along the outcrop of the Big Seam on Red Mountain.
Water must not only be kept out of the open cuts, but conducted to
natural lines of drainage by flumes or concrete-lined ditches; when
the natural lines of drainage can not be utilized ditches or flumes
must be run to proper grade.

DRAINAGE OF SURFACE WATER

Of the numerous ravines occurring on the slopes of the mountain,
those on the Shades Valley side particularly are deep and in many
places cut through or nearly through the formations overlying the
ore beds. This proximity of watercourses to the ore beds and the
disintegration of the formations often permit water to reach the
workings in fairly large amounts. In extreme cases, specially con-
structed drainage ditches are both desirable and necessary. Even
where much work must be done on account of unfavorable natural
conditions the expenditures are fully justified and the resulting cost
per gallon of water handled is practically negligible. Carefully
planned systems of drainage ditches are extended from the outcrop
along the side of ravines, grades being maintained that will insure
prompt discharge of water and yet will not permit the earth sides of
the ditches to be cut out and destroyed. The intercepted water is
discharged far enough from the outcrops or broken ground to insure
its safe run-off and thus reduce to a minimum the chance of its
entrance into the underground workings.

UNDERGROUND DRAINAGE

Meteoric waters that enter the workings despite all efforts to ex-
clude them are caught and impounded as close to the surface as pos-
sible to reduce the lift necessary to return them to the surface. For
expeditious handling of surface water the system of intercepting and diverting ditches is extended to the underground workings. Water from all sources, such as from fractured top rock, faults, and slips, is collected and conducted to sumps near the slopes where the pumps are located. (See fig. 79.)

Ditches are seldom dug to collect mine water until it attains some volume, therefore the stope floors may be very wet from the run-off from one stope to another. Water may be caught in ditches cut in the stopes along the headings or, when a large enough amount occurs at any one place, in catch basins, usually of concrete, from which the accumulated water is piped to sumps. (See figs. 80 and 81.) Four to eight inch pipes are generally used to conduct water from catch basins to sumps and are preferred to ditches, since the expense of placing and of upkeep is less than that of ditches, and the loss of water is reduced to a minimum. However, concrete-lined

ditches are often used. Furthermore, lines of drainage may be shifted at will when pipe lines are employed, but when ditches are cut to some depth and to grade, the system must be considered more or less permanent until a change is absolutely necessary.

In mines where large volumes of water enter at certain points it is often desirable if not necessary to provide water-tight stoppings in headings and stopes, thus transforming the dead ends of the stopes into reservoirs where large quantities of water may be impounded. Pipes lead from the stoppings to larger and more permanent reservoirs or to sumps at lower levels. An advantage in the use of such reservoirs is that sudden influxes of water may be controlled, the drain pipes equalizing the run-off. Without such equalizing reservoirs, catch basins and drainage ditches would overflow and flood the workings, often when this would seriously affect working conditions in the mines.
Catch basins and auxiliary reservoirs are extensions and enlargements of the system of mine drainage begun on the surface, which in the mine workings has been extended to take care of and handle water entering the mines from all sources. In those mines, therefore, where large volumes of water enter from the surface through subsidence cracks and along faults and slip planes, the catch basins, reservoirs, pipe lines, and ditches, all of which are auxiliary to the main sumps, must be correspondingly large in number and in size.

The main sumps are placed at points well below the main sources of water and are usually 1,500 to 2,000 feet from the surface.

Below the level of the main sumps the water that escapes from the intercepting systems above, plus water from other sources, such as faults and slips, is intercepted at intermediate points between the main sumps and the end of the workings and pumped back to the
main sumps. The water that accumulates in the sumps at the foot of the slopes is in like manner raised to the main sumps. The water impounded in the main sumps accumulates from all sources and is raised to the surface by pumps, usually electrically driven. (See figs. 82 and 83.)

The main sumps are located in headings and stopes provided with stoppings of reinforced concrete. In order that leakage may be reduced to a minimum and at the same time that the stoppings may be strong enough to withstand moderate heads of water, the walls, tops, and bottoms of the stopes are channeled a foot or more deep, the concrete and reinforcement being placed so as to make good connection on all sides. Occasionally it is not found necessary to do more than build a dam, behind which the water is impounded, across the stopes, to a height of 10 to 15 feet. (See fig. 84.)
Should stopes that have breakthroughs cutting the pillars be used as sumps, all such openings must be closed by concrete stoppings built like stoppings for reservoirs and main sumps. Much care is usually required to place stoppings in breakthroughs, as the ore around the openings is ordinarily somewhat shattered and weakened by the heavy charges of explosives used. In spite of the utmost care, much water may leak through the body of the pillars, slip planes providing a way for water impounded above to escape.

Before the almost universal use of concrete for underground constructions, particularly dams and stoppings, brick was commonly employed and proved quite satisfactory but rather expensive. (See fig. 85.)

Auxiliary sumps in the headings along the line of the slopes must lie below the level of the floor or in specially constructed chambers cut in the slope pillars. As pumps are often set in the headings

![Diagram of making sump out of old stope](image)

**Figure 84.—Making sump out of old stope**

next to the ribs to maintain free way for haulage, the bottom of the headings and slope pillars may be excavated to get enough room for sumps. A more usual arrangement, however, is to make all the excavations in the slope pillars immediately below the headings, mounting the pumps and pipes upon timber or concrete supports. By this procedure there is no interference whatever with the handling of ore or supplies in the headings.

**GROUP DRAINAGE SYSTEMS**

In the discussion given above the drainage system of a single mine has been considered, while in fact mines are seldom operated singly unless a group of closely related operations may be considered as one large working, with separate slopes the individual parts. If such a drainage system was applied to a group of mines the main
Iron-ore mining in birmingham district

Sump and pumping plant would be placed near the slope that is most advantageously situated with respect to transportation, power plant, sources of water in the mines, and other controlling conditions underground. The collection and distribution of inflowing waters in the various parts of the respective workings or mines is accomplished as explained before by the use of intercepting ditches and catch basins; however, pipe lines (see fig. 81, p. 113), owing to ease of placing and flexibility, are almost universally used to conduct water from the various and often distant parts of the workings to the main sumps. Even in a small group of mines it may be necessary to pump water from more than one large sump, owing to the occurrence of faults and folds in the bed, and thereby approach more nearly the unit system described before.

Figure 85.—Brick stopping for holding back mine water

Unfavorable Conditions Affecting Drainage

The foregoing discussion gives the sources of mine water, with the conditions affecting its entrance into the mines and its collection and expulsion therefrom. The conditions encountered may be grouped into two classes, those inherent in the mines and those resulting from working the ore bed.

Natural Conditions

The conditions existing in the mines by virtue of the occurrence of the ore need no further comment. Although the amount of water entering the mines through natural channels is often large, the amount is inconsiderable when compared with the influx of water from other sources.

Conditions Resulting from Mining

Conditions that affect drainage as a result of working the mines are represented by fissuring and caving of top formations and move-
ments of the bottom rock or footwall of the ore bed. Owing to the badly broken condition of the top rock along the line of the outcrop and the wide extent of the surface workings, vertically and horizontally, it is extremely difficult to collect and divert more than a small fraction of the water seeking entry to the mines on the surface. The bottoms of old open-cut workings have been filled and graded, and the accumulated water conducted from one point to another until it could be discharged beyond and below the workings and thus excluded from the mines. Breaks and subsided areas that result from caving top rock in the mines and occur some distance from the outcrop are surrounded by ditches and the water is conducted to artificial or natural lines of drainage. The main difficulty in this work is the extremely broken and porous condition of the ground near the outcrop, which is continually subsiding and opening up new channels for the entrance of water to the workings. The exclusion of waters from caves and breaks on the surface is a continuous operation, and as the water usually enters during stormy and wet weather source and amount are not fully appreciated, and the importance of the problem is not given merited consideration.

On the other hand, the continuous and often large flow of water through breaks and caving top rock in the mines themselves is more evident and is consequently given more serious consideration. Water from the surface and other sources, such as slips and faults, may contribute to the volume apparently coming from caved ground, and thereby complicate the problem of mine drainage in caved and caving ground.

CAVING OF TOP ROCK

Caving of top rock, unless extensive enough to extend to the water-bearing formations above—the Fort Payne chert and the Hartselle sandstone—has no effect upon mine drainage, but when caving and the resulting breaks reach the water-bearing beds a continuous flow of water follows, the amount of water entering the workings depending upon the lateral extent of the ground disturbed. The strike and dip of the formations near the breaks are factors controlling the amount of water involved, in some instances concentrating the flow of water to the breaks and in others draining it away from them. (See figs. 86 and 87.)

DISTURBED FORMATIONS

A disturbed condition of both top and bottom formations and of ore is also often responsible for minor amounts of inflowing waters, but has to do mainly with the transfer of water from one point to another rather than with being a source of additional supply. The effect of weak, loosely bonded top formations upon the collection and
control of mine water is shown to good advantage in the impounding of water in slopes back of presumably water-tight stoppings. Numerous reservoirs, constructed by building stoppings as discussed before, have proved to be worthless, owing to the fact that when the reservoirs are partly filled further accumulation of water is impossible because it escapes through the top and bottom rocks, particularly the former, the water going through and between the strata and past the stoppings. Under such conditions the thickness and strength of stoppings have no effect in retaining the water, nor is the depth to which the stoppings are set into the walls and top and bottom rock of any material importance in holding back the water and preventing leakage.

![Figure 86.—Extensive caving of top rock, allowing admission of water](image)

Aside from the loss of water, the entrance and passage of water for considerable distances between the bedding planes of the top formations from the reservoirs may cause falls of top rock by hydraulic pressure. The extreme wetness of the roof and the more or less forcible discharge of much water from cracks and bedding and slip planes show that water is under pressure.

Fractures in the bottom rock or in the footwall of the stopes by heaving of the bottom occur very commonly, and while not affecting drainage to the extent that breaks in the top rock do, they often form watercourses and permit leakage and loss of control of water that could otherwise be readily handled. Although broken bottom rock may be responsible for loss of water by leakage it can not be considered of serious consequence.
Thin and badly fractured pillars, particularly those in which slip planes are numerous, may cause the loss of large quantities of water by leakage when they serve as the sides of reservoirs or sumps. The application of cement, as gunite, to the stope walls would probably reduce materially the amount of water entering the mines and prevent the movement of water through ore and inclosing strata, even when breaks result from caving ground.

**PUMPING EQUIPMENT**

A wide variety of pumping machinery is employed in the iron-ore mines, depending upon the purpose for which it is used. Such equipment may be classed as follows: 1. Gathering pumps for handling water from isolated basins or pools to sumps, usually of small capacity; 2. small sump pumps, employed in lifting water to the main sumps from main slope sumps and auxiliary sumps below the main sumps; 3. main sump pumps, employed in ejecting the water from the mines.

The gathering pumps are usually of the direct-acting or centrifugal types designed to operate under heads of 200 feet or so. Cameron, Aldrich, and Worthington makes are commonly used. Their capacity ranges from 50 to 100 gallons per minute, and they are driven by compressed air or by electricity.

The second type of pumps may be designated as station pumps, and are of the centrifugal and multiplex types, although some direct-

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Figure 87.—Water entering mine through cave, as seen through upset
acting plunger pumps are used. Double-suction single-stage pumps directly connected with motors are usually employed. Triplex, quintuplex, and single-acting plunger pumps, motor driven through gears, are also employed as auxiliary pumping units. Such pumps usually operate under heads of 100 to 500 feet, and have capacities of 100 to 500 gallons per minute. Aldrich, Worthington, Allis-Chalmers, Prescott, and Cameron pumps are common makes used.

Most main-station pumps are driven by electric motors. Practice favors the centrifugal type, single and multistage, direct connected. Aside from the single-stage type, the four and six stage types are common. The capacity of these pumps is 800 to 1,800 gallons per minute. All centrifugal pumps are driven by alternating-current motors ranging from 100 to 600 horsepower and operating under heads up to 800 and 1,000 feet. Plunger pumps, also motor driven, are occasionally used. The principal makes are those just mentioned. The discharge pipes are 6 to 10 inches in diameter and are laid in the manways, often on concrete piers. In the multistage type one stage is added for about each 125 feet additional head. Much pumping is still done with compressed air as power.

GENERAL CONDITIONS

In the foregoing pages the conditions that affect mine drainage have been stated, and the means employed to free the underground workings from water have been discussed. Aside from the influx of water through breaks and caves extending to the surface or to certain water-bearing formations overlying the ore bed, the amount of water entering the workings is of relatively slight importance. Most surface water can be excluded with due care and attention, therefore the main problem is the reduction in the amount of water coming from caved ground. Solution of the drainage problem, therefore, depends largely upon the method or methods of mining employed, which in turn depend upon the support of workings. In the discussion of "Suggested changes in mining practice," methods applicable to existing conditions of support and drainage are outlined.

VENTILATION

CONDITIONS AFFECTING VENTILATION

As ore mining on Red Mountain has developed gradually from open-cut work to underground workings of large extent, the need of ventilation has not been felt as it would had the mines been developed by vertical shafts when operations were begun. Moreover, the use of compressed air in drilling has helped to delay the application of positive methods of supplying fresh air to the underground
workings, as the drills discharge some air at the working faces where it is most needed. It is now well recognized that the natural movement of air, even though assisted by air discharged from drills and hoists, is very inadequate at best in large mines and that ventilating equipment must soon be provided in the more extended workings.

After the outcrop of the Big Seam was mined to a depth of cover beyond which it could not be removed economically, slopes were driven into the face of ore exposed, marking the beginning of underground mining. This work was carried on for some years before the limit of the soft or leached ore was reached, but as such ore usually extended about 300 feet from the outcrop the slopes were evidently very limited in extent.

VENTILATION OF SHALLOW WORKINGS

While the soft ores were being mined numerous slopes were driven and headings turned off close to the surface, so close, in fact, that many of the upsets broke through. Upsets in pillars formed by subsequent headings, combined with short heading intervals and breakthroughs to the surface, provided large open spaces in the workings through which air might circulate freely if there was enough “motive column” to set up a movement. Fortunately, however, for operations then as well as at present, the difference in elevation between certain points on the outcrop is enough to provide a fairly strong movement of air in workings of no great vertical depth. This condition still exists, but to a much more limited extent than before, owing to numerous and extensive falls near the surface that have effectively closed many openings and have largely reduced or eliminated the motive column. That there is still much movement of air through the caved ground is made evident, especially in cold weather, by clouds of vapor coming from certain areas and often with some force.

VENTILATION OF DEEP WORKINGS

As the workings were extended many hundreds, often thousands, of feet, and the air currents were variable on account of surface atmospheric changes, more and more reliance was placed upon the exhaust air from drills and hoists. Extension of the workings laterally and on the dip of the ore bed has made it possible to relieve the situation somewhat by connecting headings at such levels as to furnish air to the lower workings; the full benefit of differences in elevation in producing motive columns is thus obtained, but indiscriminate holing through between headings may scatter and dissipate the air currents and lose the advantage gained. Conditions in any one mine, therefore, are seldom such that the full effect of
natural ventilation may be felt, particularly as there can be no great
difference in elevation of openings; consequently the weight of the
air column is not enough to produce a positive movement of air.
This difficulty may be and often is partly overcome by increasing
the vertical distance between the slopes and manways by driving
rock tunnels to the surface, but owing to the close proximity of these
two passages and the numerous crosstubes connecting them, any ad-
vantage that will arise from increasing the motive column is lost by
short-circuiting of the air between them.

In order that the best results may be obtained under the given con-
ditions, two or more mines are connected, the slopes and manways of
each serving as upcasts and downcasts; with such an arrangement
relatively strong air currents may be brought to but seldom very
far beyond the connecting passages. The strength of the movement
of air currents depends largely upon the difference in elevation of
the slopes of the two workings, which may amount to 100 feet or
more, according to the elevation of the outcrop from which the
slopes are driven. Under such conditions the movement of air cur-
rents is obtained by natural means alone and is therefore subject
to variations in strength and to reversals, according to the season. In
fact, so nearly balanced are the forces acting that reversals may take
place between day and night, and occasionally between parts of the
same day.

**EFFECT OF OUTSIDE TEMPERATURE**

Motive columns are also affected by the difference in temperature
between the outside air and the mine atmosphere, the greatest dif-
ference naturally producing the greatest effect. During the pro-
longed hot summers of this region it is observed that there are
relatively long periods when the difference between outside and
mine air is slight, often disappearing and consequently eliminat-
ing the motive column. This is particularly noticeable in idle mines
where the air movement can be observed to vary, often several times
an hour, giving periods of high and low slack air of some length
between reversals. Although the motive column is stronger during
the winter months, it is not as large as under similar conditions in
colder climates.

**EFFECT OF HAULAGE**

Another factor that affects ventilation but slightly is the move-
ment of cars and skips in the slopes, which sometimes assists and
sometimes retards the otherwise free movement of air currents. The
effect of the rapid movement of large skips is felt appreciably for
great distances on either side of the slopes, the air being pushed on
ahead of the moving body and drawn in behind it, thus tending to
set up eddy currents in the headings near the slopes. However, it is doubtful whether any benefit results from the movement of cars and skips incident to hoisting, except in the less extensive workings.

VENTILATION OF REMOTE WORKINGS

Beyond the connecting passages between two sets of workings conditions often become so bad that full reliance can not be placed upon air escaping from drills and hoists, but a more reliable supply of fresh air, which is particularly necessary in development work, must be provided. Small blowers or fans are occasionally used to force air into dead ends through canvas or metal piping, but the use of such blowers is limited to their capacity to deliver air against the resistance of long lines of piping. Danger of damage to the piping from blasting does not permit its installation close to the working face, rendering its use somewhat less effective. If a supply of fresh air was available for the blowers they could much improve the condition of the atmosphere at the working places, but to furnish this is not always possible, consequently vitiated air is often delivered to the advanced workings, where conditions are already bad.

Only one mine in the district has equipment to provide fresh air to the mine workings, but lack of provision for distributing the incoming air to the working places decreases very materially the benefits that should be derived from such an installation. The difficulty of building and maintaining properly constructed stoppings and partings is largely responsible for failure to extend the system into the workings and for the loss, to a large extent, of the advantages possible.

ATMOSPHERIC IMPURITIES

Factors that contribute to poor conditions of mine atmosphere are ore and rock dust and decaying timber. Dusts will always be present until water drills are universally employed, a change in practice that is coming about slowly but surely. Large amounts of fairly siliceous dust are thrown into the air during drilling and shoveling. The slow movement of air in the stopes prevents prompt removal of the dust as produced, leaving it in the air to be breathed. Decaying timber diminishes the oxygen content in mine atmospheres by increasing the carbon dioxide content and perhaps by forming a very small percentage of carbon monoxide. The diminishment of oxygen is usually so slight that it may be ignored, but the carbon monoxide and carbon dioxide, if present in large enough quantities, will cause serious trouble and may have deleterious effects upon the men.

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SUMMARY OF CONDITIONS AFFECTING VENTILATION

Conditions in iron-ore beds worked on Red Mountain are very similar to those in coal beds of the Birmingham district, particularly as to thickness, dip, and character of associated formations. The methods employed in working the ore bed are also similar, as is evident from the descriptions given in the present report. Although the ventilating practice for coal mines could be employed to advantage in the ore mines, practically no attempt has been made to adopt extensive and systematic ventilation in these latter mines.

Natural ventilation is universally employed, with but slight attempt to regulate and control the movement of air currents. With entire reliance upon natural ventilation, and with underground rock and water temperatures around 70°F., in most mines, though it is somewhat less in some shallow properties and greater in the deeper properties, and with outside air temperature around 70°F at some time of the day during nearly every month of the year, circulation of air is very inadequate in the mines at almost any time; and when inside rock and water temperatures are approximately equal to outside air temperature there is practically no movement of air in the mines. Even when there is enough difference between outside air and inside rock and water temperatures to induce air to flow into the mines, practically no fresh air reaches the working faces, as it is short-circuited through abandoned open workings and returned to the surface without reaching the active workings.

The main and in fact the entire supply of fresh air at most of the working faces is compressed air escaping from drills. The total amount available in a large mine is usually less than 5,000 cubic feet per minute, but not all of it is in use, as the drills are operated only about 25 per cent of the time. The escape of compressed air removes but a small part of the fumes of explosives and dangerous dust, and is wholly inadequate to overcome the harmful influence of stagnant air, with temperatures 70°F. and above and the relative humidity 90 to 100 per cent.

The workings are usually dry and where dry drilling is done large quantities of very finely divided dust, which is breathed by the miners, are formed and thrown into the air. Blasting is done in most mines at fixed periods, usually at the end of shifts; yet in some of the smaller mines it is allowed throughout the afternoon. This practice keeps the fine dust stirred up and suspended in the air. Aside from the dust variable quantities of poisonous fumes from explosives remain in the mine atmosphere and injure the health and efficiency of miners. Air analyses at working places show abnormally high CO₂, CO, and dust content, though usually the oxygen content is not abnormally depleted.
The air in some of the mines has, during working periods, enormous quantities of fine dust containing 20 per cent or more of silica, which ultimately harms the miners who breathe it. Silica is the most harmful constituent of insoluble mine dusts.

Even small amounts of the poisonous gas carbon monoxide, formed so frequently at working faces of these mines by blasting, are a decided menace to the health of the workers if not promptly removed. Other investigations have shown that in stagnant air with high temperature (70° or over) and high humidity, men can not exert themselves to the full without injury to health. The main remedy for unfavorable air conditions is the circulation of pure fresh air to remove gases and dusts and to provide comfort against high temperature and humidity; the only positive method of insuring this condition is the circulation of air in mines by mechanically driven fans on the surface, supplemented with doors, brattices, regulators, and tubing to control and direct the air currents.

Although there is not a large amount of combustible material in the mines, there are materials combustible locally, such as explosives, lumber, shacks, and timber cribs, which might become ignited. Should this occur the lack of control of air currents as well as lack of fire protection or fire-fighting equipment and training might prove disastrous to life and property.

SUGGESTED IMPROVEMENTS

Remedial measures which would probably insure adequate conditions of ventilation in the ore mines within safe and healthful limits are as follows:

1. Mechanically operated fans, with capacities of 25,000 to 50,000 cubic feet per minute, should be installed at the surface. The fans should be reversible and provided with fireproof housings, each mine to have its own fan.

2. Brattices, doors, stoppings, regulators, and over or under casts should be provided, as well as other devices and equipment to force air to working faces, as in gaseous coal mines.

3. For the ventilation of single drifts, crosscuts, or headings, fan and tubing units should be used.

4. All dry drilling should be discontinued and no blasting should be done when men are on shift.

5. Dry muck piles should be sprinkled several times during each shift, and water sprays should be installed at places underground where ore is dumped.

6. Abandoned stopes should be sealed by loose rock walls gunited on one side or by board, canvas, or other stoppings, and between working headings all upsets except the last one near the face should be kept closed.
7. All combustible material should be removed from places where men gather, or if it must remain it should be fireproofed by guniting or kept under lock and key away from possible ignition.

8. Fire doors should be placed at the portals of all openings reaching the surface, as well as at all openings to explosive magazines, underground stables, and oil houses.

9. Open bonfires or forges should not be allowed underground.

10. Electric installations underground should be made as carefully as similar installations in houses and offices on the surface and the housings of electric switches, motors, and other underground machinery should be carefully fireproofed. Buckets of sand and fire extinguishers or water lines and hose and nozzle should be readily available for prompt use in case of fire.

11. Plainly worded exit signs should be placed at strategic points underground and a stench system for warning miners of fire should be installed. At least two easily accessible exits from each mine should also be available.

12. It would be advisable that each mine employing 100 or more men underground should have one man employed full time on ventilation, fire prevention, and safety work.

If in the future development of ore mines vertical shafts are adopted, fans installed, and air distributed to the workings by special methods, difficulties that are progressively becoming worse will be eliminated, with consequent improvement of working conditions. As the mines in Shades Valley will probably be developed considerably in the next decade, improvement of conditions in the mines evidently will soon become imperative and more positive methods will have to be adopted.

The difficulties experienced in ventilation, as with drainage and support, are increasing rather than diminishing as workings are enlarged, and all contribute to the necessity for changed mining methods. The transition from present practice to improved future practice will be slow, as in the past, but will acquire a permanence that is somewhat lacking at present.

**MISCELLANEOUS PRACTICE**

"Miscellaneous practice" may include lighting, signaling, sanitary conditions, fire hazards, and concentration of high-silica ores, all important parts of mining operations.

**LIGHTING**

All slopes and stations in the actively operated parts of the mines are lighted by electricity, the wires usually being attached to iron

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supports fastened to the walls or roof. The lights are seldom protected against breakage, but are ordinarily placed high enough to be out of reach. Lighting wires are commonly extended into the headings for special purposes only, such as to supply illumination for stables, pumping and power stations, and powder magazines.

Acetylene lamps are generally used by miners and are useful and efficient in all mining operations, their lightness and convenience making them particularly adapted to the work. Large reflectors permit light to be concentrated on walls and roof, giving good results for mining and inspection. In some mines the carbide is supplied by the companies, and in others the miners provide both lamps and carbide; in any case the expense is slight, although much carbide is used and some is wasted.

In mines where haulage is by trip, the riders or "chainmen" use oil torches which give a fair amount of light and are more dependable so far as keeping a light is concerned.

**SIGNALING**

Virtually all mines have telephones conveniently placed at stations along the slopes. The telephones are for the convenience of officials only and are used solely to facilitate direction of operations underground.

The movement of cars and skips on the slopes is directed and controlled by a two-wire signaling system, usually operating on a bell-ringing transformer system. With trip haulage the head chainman gives the signals, while with skip haulage the man in charge of the loading station directs the placing of skips, both telephones and bell signaling being essential to the full control of the skips. The practice in signaling is to use a "bridge" wire for making contact between the two wires of the system, but it is not uncommon for the operator to pull the two wires together, the upper one usually receiving the greater strain. In consequence the wires tend to sag and may accidently make contact, thus confusing signaling.

**SANITARY CONDITIONS**

Stations in virtually all mines have drinking water supplied by pipe lines provided with faucets. The source of this supply is water from top rock broken by caving, consequently the water is not contaminated by drainage from the workings. Special reservoirs or catchpools impound the water; from these it is distributed through the mines by pipes. This water is also used as a supply for the miners' village near the mines.

Most of the conditions affecting health in the ore mines are fair. One marked exception is that no regular and systematic disposal of
waste is provided, as parts of virtually all of the mines are used by the miners as privies and no provision is made for the removal of the excrement. This practice results in disagreeable odors and the pollution of the mine water.

**FIRE HAZARDS**

Few serious fires have occurred in the ore mines of the district, although the timbering in a number of slopes has burned out without loss of life. Such fires have emphasized the importance of lining the slopes with concrete, which has been done in a number of mines. The amount of timber standing in the stopes of the more recent workings is only large in those localities where large stopes are worked. In old abandoned workings is to be found much timber which, though well rotted, might become a fire menace. However, the greatest source of danger from fire exists in the headings near the slopes, where large quantities of timber and lumber have been piled indiscriminately. These piles of refuse timber, although badly decayed, are fairly dry, due to circulating air currents, and therefore would burn readily. Since workings are very long, parts being inaccessible to the surface by foot, and positive means of controlling ventilation are lacking, a serious condition would exist, even though only limited amounts of timber should become ignited and burn or smolder.

**CONCENTRATION OF HIGH-SILICA ORES**

The problem of beneficiating high-silica ores has been mentioned under various headings in this report, and its relation to the various phases of mining has been emphasized. Successful and economical treatment of the large tonnages of low-grade, high-silica ores in the Birmingham district would add greatly to the reserve supply of ores available for furnace use. The treatment of these ores has received much attention in the past and is now being attacked vigorously. It is hoped that satisfactory results will soon be obtained, when the full importance and value of the additional reserve will have become apparent.

The mining of low-grade ores of the Big Seam, now carried on largely in a desultory way and in a few localities only, will assume a position of first importance and necessitate changes in practice that will permit the extraction of ores from both benches of the Big Seam as well as the removal of lower-bench ores in the old workings. However, mining the whole thickness of the Big Seam at one operation will be simple compared with the reclamation of the ores of the lower bench in abandoned workings where extensive caving has occurred.
OBJECTIONS TO MINING LOWER-BENCH CKRES

Mining high-silica ores and ferruginous sandstone of high iron content must be part of future mining practice, therefore it should receive consideration in this connection. The following objections have been given to the mining of the lower-bench ores:

1. The work of mining would be increased, owing to the necessity of sorting and handling much slate that would probably mix with the ore more than when the upper bench alone was mined.

2. Falls of top rock would probably be more numerous, as the length of time required to mine out all the ore would be greater than at present where operations are confined to the upper bench.

3. The cost of timbering would be prohibitive, longer, larger, and more timbers being required.

4. Large quantities of water would have to be handled because of more extensive caving.

5. Mining would be more expensive from enlarged operations.

REBUTTAL

It is doubtful whether the amount of slate to be handled in the new work would be any greater than when the upper bench alone is mined, as the parting in certain localities is now largely loosened and disposed of by stowing in old stope, Where the lower bench has changed to thin beds of ore, ferruginous sandstone, and slate, the expense of sorting out the waste and disposing of it might be prohibitive, but such ores would probably not be mined and do not need to be considered in this connection.

Where mining is done in new territory the time of mining need not be prolonged to such an extent that falls of top rock would result. Caving of top rock would be largely under the control of the miners and would depend upon the size of openings in the preliminary and final operations, which would be particularly the case should caving methods be employed in removing the ore. The mining of previously worked territory would require great care, especially when areas have caved or are overlaid by water-bearing formations; under such conditions the amount of ore reclaimed would be small.

Should timber be employed as permanent support the cost would no doubt be excessive, but if caving methods applicable to these ores were used, timber would be used as temporary support only. The amount needed and the size of the individual units would not have to be large.

The same considerations that affect the mining of the upper bench of the Big Seam would apply in the working of the entire thickness, as far as ingress of water is concerned. It is the width of opening rather than the thickness of bed that affects the area caved. Extren-
sive caving under areas of water-bearing formations should not be tolerated.

Mining costs should decrease rather than increase with increased production from a given area, as the breaking of ore from high faces is more readily accomplished than from low faces. Further, with caving methods the cost of breaking ore is extremely low, as is also the cost of support, although rapid and systematic work is necessary to prevent loss of ore.

SUGGESTED CHANGES IN MINING PRACTICE

GENERAL CONSIDERATIONS AFFECTING WORK

In the mining methods discussed in the foregoing pages, including development, working, support, drainage, and ventilation, previous and recent practice are described and the methods employed in the transition between the two. It now remains to outline briefly possible future practice based upon past experience in the mines and upon conditions that it is reasonable to assume will exist as workings are extended. However, the conditions under which work will be carried on when the operations have reached greater depths are not wholly unknown.

The conditions observed in the operating slope mines embrace a wide range in lateral extent and in depth of cover, and show the changes that are occurring progressively as the ore bed takes on more weight from superimposed formations. It is but a step, therefore, to apply to mines approaching considerable depth the conditions in mines that have already attained considerable depth of cover; the facts in the former case are limited, while in the latter case they are numerous. In view of the known facts and those indicative of probable future conditions, tentative suggestions relating to later practice are offered below. It goes without saying, however, that changes in practice should be made and will probably be necessary soon, particularly in certain localities.

What is needed is not more mines but better operated and equipped mines—more efficient, low-cost producers; and it is with these considerations in mind that the following suggestions are made.

The problems confronting future mining operations are threefold: Subsidence of the surface, caving and fracturing of top rock, and percentage of ore recovery. No one problem can be said to be more important than the others, for in one locality one problem may assume serious proportions, while in another locality another problem may transcend the others in importance and thereby demand special consideration.

Subsidence of the surface will probably affect residential property only, and will be confined to a comparatively narrow strip of land
lying on the eastern and southern slope of Red Mountain, particularly the part nearest Birmingham. It is probable that less than 50 per cent of the ore bed has been worked underground in this locality, consequently in future the surface can be given any protection possible. In localities where mining has already been done, surface subsidence can only be controlled at great expense.

Outside of the residential districts the land has little value, except for grazing, consequently subsidence will not be serious, and since the land is owned or controlled by the operating companies, damages to the surface should not involve legal complications. Moreover, it is hardly probable that subsidence will extend much beyond the slope of Red Mountain, because heavy beds of lime and sandstone occur that will check any pronounced and serious irregular settlement of the formations that overlie the ore bed.

**CONTROL OF INFUX OF WATER INTO MINE**

Although top formations will fail ultimately where the ore bed is worked, and such failure will extend a great distance vertically, the most extensive fracturing and caving will probably occur within the zone of leaching of certain formations above the ore bed, and will correspond to but extend further from the outcrop than the leached zone observed in connection with the ore bed. Extensive caving has, however, occurred in a number of localities, the breaks in the top formations reaching to and disturbing certain water-bearing formations, particularly the Fort Payne chert.

The distance from the outcrop of the ore bed on Red Mountain to where the chert bed ceases to be porous enough to hold water in any large amount seriously affects the problems of mining and drainage. Information as to the extent of the leached zone of the Fort Payne chert bed is not available, but should this zone prove extensive adequate support in that area will have to be given serious consideration. Preliminary mining operations have probably extended over the larger part of the leached area of the Fort Payne chert formation and, except in localities where robbing has taken place, there is reasonable security against tapping the water-bearing beds by caving and fracturing. Beyond the leached area of the Fort Payne chert bed the water problem is not a menace. Continued and extensive robbing of pillars in the old workings means more and larger breaks, which will tap the water-bearing formation and admit an increasing volume of water which must be pumped from the mines. Protection must be provided or a limit will be reached beyond which the mines can not be freed of water economically. The protection furnished must be one of two forms or both, as conditions dictate, namely: For virgin ground, preliminary or one-stage mining alone;
and for previously worked ground, no robbing of pillars at all or extremely careful work if any is done. In other words, all mining must be limited to a definite percentage basis of the ore. If this practice is followed to the lower limit of the leached zone of the Fort Payne chert bed, ample protection will probably be provided against any material influx of water through caving ground.

Extensive breaks in top rock can be limited and damages already done can probably be remedied to a large extent by grouting the known breaks or by isolating the caved areas by walls of concrete. Although these methods are relatively expensive, they would greatly reduce the present operating cost and avoid materially increased difficulties. The first method—grouting—might sometimes prove to be more satisfactory, because the top rock in some localities permits water to pass through it and around dams and barriers built in the stopes and headings.

CHANGES IN DEVELOPMENT

Although there is little prospect of any material change in the near future in the method of developing the iron-ore mines, it is not unlikely that a change from slopes to vertical shafts will become increasingly desirable and will ultimately be required. Standardized equipment and practice, already adopted in many of the mines and to a more limited extent in mines of different companies, although highly desirable and commendable, hinder the development of new methods. This is particularly true in developments where a change from skip to cage haulage would mean practically scrapping the present slope equipment. However, the ease of handling ore underground, greatly increased production, and economical operation of vertical shafts should be so evident as to remove all doubt as to the desirability of such a change.

ADVANTAGES OF VERTICAL SHAFTS

The principal advantages of using vertical shafts in developing the iron-ore-bearing areas beyond Red Mountain are as follows:

1. Geological conditions are probably more favorable some distance from the outcrop. Although faults and other disturbed conditions of the ore bed are known to occur, conditions are probably better than on or close to Red Mountain.

2. Even though geological conditions are no better than on Red Mountain, the more symmetrical development of the mines would be reflected in improved working conditions.

3. Shorter hauls would be possible, with increased tonnage and decreased costs. Underground haulage systems could be installed more easily.
4. The upkeep of shafts would cost much less than that of slopes.
5. The wear on hoisting rope would be much less than with slope haulage. The speed of hoisting could be greatly increased.
6. The cost of keeping the mines free of water would be reduced greatly or obviated.
7. Ventilation would be ample and positive.
8. Labor could be handled more efficiently and less time lost.
9. Mine equipment could be centralized.
10. The number of loading stations would be greatly reduced and ore handled more efficiently.

CHANGES IN MINING

The problem of the future working of the mines then resolves itself into the adoption of mining methods that will permit the maximum extraction of ore with minimum loss from caving. The most serious problems are still to come, although their nature is fairly definitely known; the problems regarding conditions intermediate between the present and future require immediate consideration, and their successful solution should insure successful solution of the more serious future problems. The mining of Red Mountain ores is now in a transition period between definitely known and only partly known conditions, with a water menace threatening.

A wide range of conditions of support occurs in the district (see "Support of mine workings"), therefore it is impossible to outline methods that might always be applicable so a general statement will suffice. Future workings will be under greatly increased cover and the pressure of cover will increase progressively; this increase will make itself felt much more positively than similar increases in the past, the accumulated load being vastly greater. In consequence, stopes of sizes formerly considered safe may be beyond the safe working limit and areas of pillars known to have an ample factor of safety may prove inadequate for future work.

The present practice, fairly extensively followed in the larger mines, of leaving 30 per cent of the ore unmined in the pillars, although apparently ample may have to be increased very materially, even to 50 or 60 per cent or more in future preliminary work. The final removal of ore by drawing the pillars may increase the extraction to 85 per cent or more, but it must be so removed as not to jeopardize adjacent workings by the caving and crushing of the pillars.

The practice that permits caving of top rock in large stopes may prove satisfactory when covers are comparatively light, but in the deeper workings will hardly be permissible even with wide barrier pillars, as crushing will undoubtedly take place and follow down the
dip, breaking the barrier pillars and spreading in all directions. The destructive action of squeezes crushing wide pillars left as protection and the ultimate wrecking of extensive workings are too well known to require repetition. In the Pennsylvania anthracite fields barrier pillars 150 to 200 feet wide have proved ineffective to stop or even check for any great length of time the crushes that developed on and swept down the dip of the beds. Isolated squeezes and influxes of water through slip and fault planes might be successfully controlled by inclosing pillars in connection with concrete stoppings.

**APPLICABILITY OF CAVING METHOD**

As areas of high pressure are approached some good method of procedure in working the mines will have to be adopted. The controlling factor in making the choice should be experience—in other districts where similar conditions prevail; at the same time the method adopted must be elastic enough to meet varying conditions and be economical. As the pillars and top rock will be difficult to support it would be logical to choose a method that would permit their collapse early in the operations so that costs of even temporary support would be relatively slight. In brief, a caving method might be adopted to advantage, preliminary workings, such as headings and stopes, being carried narrow to reduce the need of support; where possible under roof ore, the final operation would remove the pillars and roof ore and permit the top rock to cave. By carrying the stoping faces in a direction that cuts across the slip planes at an angle of 45° or thereabouts, the greatest strength of pillars and top rock would be developed, and, by forming the face into a straight line to prevent concentration of pressure at any one point, the caving of top rock could best be controlled.

If the character of ore in the lower bench of the Big Seam changes or beneficiation of the high-silica ores of the lower bench is successfully accomplished, a new condition will have to be met in that the whole thickness of the Big Seam will be workable. In this event the caving method suggested above would be modified to meet the new condition. Preliminary work and the removal of pillars would be done in the lower bench, the upper bench serving as roof to such workings; and caving would follow drawing or robbing of pillars. The superior strength and thickness of the upper bench, compared with the weaker slate roof, would tend to insure the successful working out of some such method.

If the upper bench or the combined upper and lower benches of the Big Seam were mined by a caving method, mining would have to be done by retreating from the limits of the property and across the line of dip to insure against crushes starting and getting out of
control. However, the work could probably be done best by development in panels, the inclosing pillars serving as a protection in the earlier stages of mining. The details of such a method, if applicable at all, would have to be worked out after the conditions existing in the ore and top rock were fairly well understood. There is no doubt, however, that a method will have to be devised that can be widely employed and controlled, that permits a relatively high percentage of extraction, and that is economical in labor and supplies.

CHANGE IN VENTILATION

Satisfactory ventilation in the average iron-ore mine has about reached its limit, while in many mines the limit has been exceeded and unsatisfactory conditions exist. No mine in the district can be said to have a complete system of positive ventilation. Blowers are inadequate to supply air to the advanced workings and dead ends, because air so supplied has usually been rendered unfit for further use by its passage through the workings before delivery to the advanced workings. Carefully planned ventilating systems are needed in all of the large mines if satisfactory conditions are to be provided for working; however, positive exhausts or pressure systems could be installed at low cost in the present slope workings. See also the recommendations near the end of the section on "Ventilation."

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PUBLICATIONS ON METAL MINING

A limited supply of the following publications of the Bureau of Mines has been printed. Requests for publications available for free distribution should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list of all publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION

**Bulletin 48.** The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1914. 50 pp., 3 pls., 7 figs.

**Bulletin 80.** A primer on explosives for metal miners and quarrymen, by C. E. Munroe and Clarence Hall. 1915. 125 pp., 15 pls., 17 figs.

**Bulletin 204.** Underground ventilation at Butte, by Daniel Harrington. 1923. 131 pp., 3 pls., 42 figs.


**Miners' Circular 11.** Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.


**Miners' Circular 17.** Accidents from falls of rock and ore, by Edwin Higgins. 1914. 23 pp.

**Miners' Circular 19.** The prevention of accidents from explosives in metal mining, by Edwin Higgins. 1914. 16 pp., 11 figs.

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| INDEX |
|-------|-------|-------|
| Abbott, C. E., acknowledgment to .................. | Catch basins, for drainage, use of ............ | Page |
| Acetylene lamps, for miners, use of ................ | view of ......................................... | 114 | 115 |
| A-headings, for handling ore, use of ................ | Caving method, use of ........................ | 129 | 115 |
| in mining pillars, use of ........................... | .................................................. | 102 | 136 | 137 |
| diagram showing ...................................... | Chutes, for handling ore, discussion of .......... | 53-54 | 94-95 |
| Alabama, production of red iron ore in, data on .......... | shaking, use of ............................... | ........ | 94-95 |
| Alice mine, mining of hard ores at, history of .......... | Cleavage planes. See Slips. .................. | 22 | 2 |
| Allison, V. C., work cited ................................ | Coffin, W. H., acknowledgment to .............. | 7 | 2 |
| Alumina, in hard ores, amount of .................... | Compressed air, for drills, supply of .......... | 128 | 59 |
| curve showing ........................................ | Concrete, as support, use of .................. | 13 | 78-79 |
| Ball, Edwin, acknowledgment to ........................ | Crook, F. H., acknowledgment to ................ | 2 | 2 |
| Barksdale, Jelks, acknowledgment to .................. | Cross-bedding, of top rock, effect of .......... | 2 | 2 |
| Big Seam, dip of, diagram showing .................... | 20-21, 67-68, view showing ................... | 16 | 62 |
| location of .......................................... | Cross-bedding planes, effect on support of .... | ........ | 63 |
| lower bench, mining methods in, description of ......... | workings ......................................... | 10 | 62 |
| obligations to abandonment of ....................... | curve showing .................................. | ........ | 63 |
| suggested alteration of ................................ | Cross-heading tracks, diagram showing .......... | 23-24 | 100 |
| open-cut working of, description of .................. | grades for ..................................... | ........ | 101 |
| robbing entire thickness of, methods for ............ | on high dips, use of, diagram showing .......... | 4-5 | 53 |
| thickness of .......................................... | use of .......................................... | 54-55 | 99-100 |
| upper and lower benches of, simultaneous mining of ... | views of ....................................... | 10 | 99 |
| diagram, description of .............................. | Cross headings, for working lower-bench ore ... | 50-50 | 101 |
| view of ................................................ | diagram showing ................................ | 49-50 | 47 |
| upper bench, mining methods in, description of ........ | Cross-bedding planes, effect on support of .... | ........ | 55-55 |
| Birmingham district, early history of ................ | workings ......................................... | 22-25 | 110 |
| present mining methods in ............................ | curve showing .................................. | 2 | 33 |
| Blasting, holes for, drilling of ..................... | Cross-heading tracks, diagram showing .......... | 23 | 111 |
| equipment for .......................................... | development, discussion of .................... | 57 | 110-122 |
| in Big Seam, methods of ................................ | underground, discussion of .................... | 55-60 | 113-114 |
| Bottom rock, effect on loading method ................. | views of ....................................... | 92-93 | 114, 115, 116 |
| Bowron, C. E., acknowledgment to .................... | Drainage systems, group, feasibility of ....... | 2 | 9 |
| Branch tracks, arrangement of, diagrams ............ | Drifts, development by, description of .......... | 49 | 117-118 |
| showing ................................................ | Drill bits, sizes of, lists of ................ | ........ | 6-7 |
| in wide upsets, use of ................................ | Drills, mounting of ................................ | 40 | 59-60 |
| Breakthroughs, driving of, description of ............ | types used .................................... | 50 | 57 |
| irregular, view of .................................... | Drilling, in Big Seam, methods for ............ | 51-52 | 58 |
| value of ................................................ | Egy, W. L., work cited ........................ | ........ | 55-60 |
| Burchard, E. F., work cited .......................... | Electricity, for lighting mines, use of ....... | 37 | 128 |
| Bureau of Mines, advisory committee, acknowledgment to | Ellis, E. E., acknowledgment to .............. | 10, 11 | 128-129 |
| tests of strength of red iron ore by ................ | Engine planes. See Gravity planes. ............ | .... | 2 |
| Butts, Charles, work cited ........................... | Explosives, for hard ores, types used .......... | 10, 11 | 60 |
| Cahalan, J. J., acknowledgment to ................... | Fairburn, C. T., acknowledgment to .......... | ........ | 2 |
| Calcium oxide, in hard ores, amount of ................ | Fault planes, effect on support ............... | 2 | 73-73 |
| Carbon dioxide, in mine air, occurrence of .......... | Ingress of water through ........................ | 13 | 73 |
| Carbon monoxide, in mine air, occurrence of .......... | 126 | 112 |
| Cars, racking, in stopes, method of .................. | view of ......................................... | ........ | 18 |
| trimming, in stopes, method of ...................... | Faults, in ore bed, description of ............. | 126, 127 | 72 |
| view showing .......................................... | Fire hazards, in mines, elimination of .......... | 91-92 | 17-18 |
| view showing .......................................... | Folds, in ore bed, description of ............... | 91-92 | 128, 130 |
| view showing .......................................... | Forbes, J. J., acknowledgment to .............. | 92 | 17 |

141
INDEX

Fort Payne chert, cAVING of .................................. 133
Ingress of water from ........................................... 111
Effect on support of workings ................................. 61
Formations, dip of, effect on support of .................. 62-64
Porous, ingress of water from ................................ 111-112
Strength of, effect on support ................................. 61-62
Water bearing, effect on support ............................. 61
Geobegan, L. E., acknowledgment to ...................... 2
Geology, of Birmingham district, discussion of .......... 10
Goodrich, Levine S., works of ................................. 3
Gravity planes, for handling ore, use of .................... 95-96
diagram showing ................................................ 95
In folded areas, use of ........................................... 103
In large stopes, use of ........................................... 41
diagram showing ................................................ 41
Gulf States Steel Co., Inclined shaft of, description of . 32
diagram of ......................................................... 32
Hand stripping, on Red Mountain, description of ....... 4-5
Handling, of ore, at surface, discussion of ............... 108-110
between stopes and headings, method of ................. 98-103
by chutes, discussion of ...................................... 94-95
by planes, diagram showing ................................ 95
discussion of ..................................................... 95-96
by scrapers, discussion of .................................... 96-98
in mines, discussion of ....................................... 85-107
Harrington, Daniel, acknowledgment to ................. 2
Hartsville sandstone, ingress of water from .............. 61, 111
Haulage, car, in slopes, discussion of ...................... 86-87
In stopes, discussion of ....................................... 90-94
Combined skip and trip, description of ..................... 105
diagram of ......................................................... 105
Effect on ventilation ............................................. 124-125
In large stopes, procedure for ................................ 94
Skip, in slopes, discussion of ................................. 88-90
Surface handling of cars for ................................ 108-109
Trip, surface handling of cars for ............................ 108-109
See also Haulage, car.
See also Wagon haulage.
Haulage methods, for hard ores, description of ......... 7-8
Tracks for, view of .............................................. 8
For soft ores, description of .................................. 4-6
Motor, in slopes, discussion of ............................... 90
View of ............................................................ 91
In stopes, discussion of ....................................... 93-94
Mule. See Mules.
Power-driven, discussion of .................................. 107
Haulage ropes, data on ......................................... 88
Haulageways, driving, procedure for ....................... 102-103
Headings, driving of, arrangement of holes for .......... 35-36
Diagrams of ....................................................... 36, 37
With scraper loaders, description of ......................... 43-44
Handling ore in, by scrapers .................................. 98
In slope mines, arrangement of .............................. 30
Driving of .......................................................... 27-28
Hesslip, Joseph, work of ....................................... 3
Hickory Nut bed, location of .................................. 10
Hillman, Daniel, work of ...................................... 3
Ida bed, location of ............................................. 10
Impurities, atmospheric, effect on ventilation .......... 125-126
India, gold mines of, explosive rock in ................... 83
Iron, metallic, in hard ores, amount of .................... 13
curves showing .................................................. 14, 15
Iron ore, Big Seam, strength of .............................. 65-66
Ironton bed, location of ....................................... 10
Thickness of ...................................................... 10-11
Joint planes, advantages of ................................... 66
disadvantages of ................................................ 66
Katz, S. H., work cited .......................................... 128
Kettles, in top formations, effect of ......................... 18
View of ............................................................ 19
Lacey, W. M., acknowledgment to ......................... 2
Lighting, mine, equipment for ................................. 128-129
Lime, in hard ores, amount of ................................. 13
curves showing .................................................. 14, 15
Loaders, mechanical, handling ore by, discussion of ... 96-98
Loading, data on .................................................. 93
Manways, arrangement of, diagram showing .............. 38
Construction of, view of ....................................... 79
Driving of, arrangement of holes for, diagrams of .... 34, 35
Methods for ........................................................ 34-35
Handling ore in, discussion of ............................... 86-90
In slope mines, advancing, diagram of .................... 29, 30
diagram of ........................................................ 27
Driving of .......................................................... 26-27
Masonry walls, as support, use of .......................... 78
McCalley, Henry, work cited ................................... 10
McElroy, G. E., work cited ..................................... 125
Meriwether, F. V., acknowledgment to .................... 2
Michigan, copper mines of, explosive rock in .......... 83
Mine dusts, harmful constituents of ......................... 126, 127
Mine workings, support of, discussion of ................. 61-85
Mining, description of ......................................... 33
Red iron ores, Birmingham district, history of .......... 2-9
Mining methods, changes in ................................. 39-45, 132-137
Motor haulage. See Haulage, motor.
Mud slip, crossing stope, view of ............................ 67
Mules for haulage, use of ....................................... 106-107
O'Brien, Frank P., operation of coke ovens by ........... 3
Oil torches, for miners, use of ................................. 129
Open-cut mines, mining methods at, description of .... 4-6
Ores, compressive strength of, tests of ................... 21
Hard, character of .............................................. 11-12
Constituents of, variation in ................................ 13
curves showing .................................................. 15
Early mining of, description of .............................. 7-8
Grade of ............................................................ 12-13
Modern mining of, description of ........................... 8-9
Physical properties of .......................................... 14
Structure of ........................................................ 16-17
High silica, concentration of ................................. 130-132
In pillars, as roof support, discussion of ................. 73-74
Soft, mining of, description of ............................... 4-7
<table>
<thead>
<tr>
<th>Ore beds, dip of, effect on handling ore</th>
<th>103-106</th>
</tr>
</thead>
<tbody>
<tr>
<td>effect on support</td>
<td>69-71</td>
</tr>
<tr>
<td>diagram showing</td>
<td>69</td>
</tr>
<tr>
<td>variation of</td>
<td>17</td>
</tr>
<tr>
<td>number of</td>
<td>10</td>
</tr>
<tr>
<td>parts of, connection of</td>
<td>104-105</td>
</tr>
<tr>
<td>diagram showing</td>
<td>104</td>
</tr>
<tr>
<td>view of</td>
<td>105</td>
</tr>
<tr>
<td>thickness of</td>
<td>10</td>
</tr>
<tr>
<td>Ore cars, tracks for, view of</td>
<td>8</td>
</tr>
<tr>
<td>Penobscot, W. J., acknowledgment to</td>
<td>2</td>
</tr>
<tr>
<td>Phillips, W. B., work cited</td>
<td>10</td>
</tr>
<tr>
<td>Pillars, failure of, discussion of</td>
<td>80-84</td>
</tr>
<tr>
<td>views of</td>
<td>82</td>
</tr>
<tr>
<td>robbing of, methods of</td>
<td>50-55</td>
</tr>
<tr>
<td>diagrams showing</td>
<td>51, 52, 53, 55</td>
</tr>
<tr>
<td>slabbng of, description of</td>
<td>51</td>
</tr>
<tr>
<td>diagram showing</td>
<td>51</td>
</tr>
<tr>
<td>splitting of, description of</td>
<td>52-53</td>
</tr>
<tr>
<td>reasons for</td>
<td>54</td>
</tr>
<tr>
<td>Pipe lines, for drainage, use of</td>
<td>114, 115</td>
</tr>
<tr>
<td>view of</td>
<td>115</td>
</tr>
<tr>
<td>Planes, balanced</td>
<td>See Gravity planes</td>
</tr>
<tr>
<td>Porosity of hard ores, curve showing</td>
<td>14</td>
</tr>
<tr>
<td>Pots, in top formations, effect of</td>
<td>18</td>
</tr>
<tr>
<td>view of</td>
<td>19</td>
</tr>
<tr>
<td>Potholes, effect on support</td>
<td>71-72</td>
</tr>
<tr>
<td>in top formations, effect of</td>
<td>18</td>
</tr>
<tr>
<td>view of</td>
<td>19</td>
</tr>
<tr>
<td>Props, failure of, causes of</td>
<td>79</td>
</tr>
<tr>
<td>for manways, use of</td>
<td>77</td>
</tr>
<tr>
<td>for support, use of</td>
<td>70</td>
</tr>
<tr>
<td>views showing</td>
<td>71, 72, 73, 74, 75, 76, 77, 78</td>
</tr>
<tr>
<td>steel, for loading stations, use of</td>
<td>77</td>
</tr>
<tr>
<td>timber, for loading stations, use of</td>
<td>77</td>
</tr>
<tr>
<td>in haulageways, use of</td>
<td>76-77</td>
</tr>
<tr>
<td>in stopes, use of</td>
<td>74-75</td>
</tr>
<tr>
<td>Prospect holes, entrance of water through</td>
<td>112</td>
</tr>
<tr>
<td>Pumps, mine, types of</td>
<td>121-122</td>
</tr>
<tr>
<td>Red Mountain, East, occurrence of ore on</td>
<td>17</td>
</tr>
<tr>
<td>open-cut workings on, view of</td>
<td>4</td>
</tr>
<tr>
<td>Republic Iron &amp; Steel Co., organization of</td>
<td>3</td>
</tr>
<tr>
<td>Reservoirs, for drainage, use of</td>
<td>114, 115</td>
</tr>
<tr>
<td>Richardson, A. S., work cited</td>
<td>125</td>
</tr>
<tr>
<td>Rock, explosive, discussion of</td>
<td>83-84</td>
</tr>
<tr>
<td>See also, Bottom rock, Top rock</td>
<td></td>
</tr>
<tr>
<td>Rock houses, description of</td>
<td>108-109</td>
</tr>
<tr>
<td>views of</td>
<td>108-109</td>
</tr>
<tr>
<td>Rock tunnels, for handling ore, use of</td>
<td>103-104</td>
</tr>
<tr>
<td>Rolls, in top formations, effect of</td>
<td>18</td>
</tr>
<tr>
<td>Roof ore, as support of draw slate, view of</td>
<td>64</td>
</tr>
<tr>
<td>thickness of, effect on support of works, curve showing</td>
<td>63</td>
</tr>
<tr>
<td>Rotary dump, use of</td>
<td>89-90</td>
</tr>
<tr>
<td>view of</td>
<td>90</td>
</tr>
<tr>
<td>Sanitation, mine, measures for</td>
<td>128-130</td>
</tr>
<tr>
<td>Sandstone, above Big Seam, weakness of</td>
<td>66</td>
</tr>
<tr>
<td>discussion of</td>
<td>96-98</td>
</tr>
<tr>
<td>in large stopes, use of</td>
<td>41-45</td>
</tr>
<tr>
<td>limitations of</td>
<td>98</td>
</tr>
<tr>
<td>Seams, effect on blasting</td>
<td>58</td>
</tr>
<tr>
<td>Shafts, inclined, development by, description of</td>
<td>32-33</td>
</tr>
<tr>
<td>diagram showing</td>
<td>32</td>
</tr>
<tr>
<td>vertical, advantages of</td>
<td>134-135</td>
</tr>
<tr>
<td>development by, description of</td>
<td>31-33</td>
</tr>
<tr>
<td>development of Big Seam by</td>
<td>24-25</td>
</tr>
<tr>
<td>Shannon slope, description of</td>
<td>32</td>
</tr>
<tr>
<td>diagram of</td>
<td>32</td>
</tr>
<tr>
<td>view of</td>
<td>50</td>
</tr>
<tr>
<td>Signaling, methods for</td>
<td>129</td>
</tr>
<tr>
<td>Silica, in hard ores, amount of</td>
<td>13</td>
</tr>
<tr>
<td>curves showing</td>
<td>14, 15</td>
</tr>
<tr>
<td>Singhwald, J. T., work cited</td>
<td>1</td>
</tr>
<tr>
<td>Skips, development of slopes for haulage by, description of</td>
<td>30-31</td>
</tr>
<tr>
<td>in top rock, amount of, effect on support of workings</td>
<td>62</td>
</tr>
<tr>
<td>curve showing</td>
<td>63</td>
</tr>
<tr>
<td>Sledges, for breaking ore in stopes, view of</td>
<td>91</td>
</tr>
<tr>
<td>Slips, effect on blasting</td>
<td>56</td>
</tr>
<tr>
<td>Slip planes, in Big Seam, direction of, diagram showing</td>
<td>20</td>
</tr>
<tr>
<td>effect of</td>
<td>18-20</td>
</tr>
<tr>
<td>view of</td>
<td>21</td>
</tr>
<tr>
<td>in top rock, effect of</td>
<td>66-67</td>
</tr>
<tr>
<td>view of</td>
<td>67</td>
</tr>
<tr>
<td>weakening of top rock by</td>
<td>62</td>
</tr>
<tr>
<td>Slopes, advancing of, diagram of</td>
<td>28, 30</td>
</tr>
<tr>
<td>arrangement of</td>
<td>30</td>
</tr>
<tr>
<td>diagram showing</td>
<td>38</td>
</tr>
<tr>
<td>development by, description of</td>
<td>6-7</td>
</tr>
<tr>
<td>development of, for skip haulage, description of</td>
<td>30-31</td>
</tr>
<tr>
<td>for trip haulage, description of</td>
<td>25-30</td>
</tr>
<tr>
<td>general plan of</td>
<td>28</td>
</tr>
<tr>
<td>development of Big Seam by, description of</td>
<td>26-31</td>
</tr>
<tr>
<td>driven across slope planes, view of</td>
<td>57</td>
</tr>
<tr>
<td>driving of, arrangement of holes for, diagram showing</td>
<td>36, 37</td>
</tr>
<tr>
<td>methods for</td>
<td>25-28, 33-34</td>
</tr>
<tr>
<td>handling ore in, discussion of</td>
<td>86-90</td>
</tr>
<tr>
<td>Sloss-Sheffield Steel Co., organization of</td>
<td>3</td>
</tr>
<tr>
<td>Songo mine, vertical shaft at, description of</td>
<td>31</td>
</tr>
<tr>
<td>Specific gravity, of hard ores, curve showing</td>
<td>14</td>
</tr>
<tr>
<td>determination of</td>
<td>15-16</td>
</tr>
<tr>
<td>Stopes, common arrangement of, diagram showing</td>
<td>38</td>
</tr>
<tr>
<td>development of, by scraper loaders</td>
<td>44-45</td>
</tr>
<tr>
<td>diagram of</td>
<td>44</td>
</tr>
<tr>
<td>handling ore in, discussion of</td>
<td>90-98</td>
</tr>
<tr>
<td>multiple neck, handling ore in, by scrapers</td>
<td>98</td>
</tr>
<tr>
<td>preliminary, handling ore in, by scrapers</td>
<td>98</td>
</tr>
<tr>
<td>single neck, handling ore in, by scrapers</td>
<td>97</td>
</tr>
<tr>
<td>wide, adoption of new methods in</td>
<td>39-45</td>
</tr>
<tr>
<td>gravity planes in, use of</td>
<td>41</td>
</tr>
<tr>
<td>diagram showing</td>
<td>41</td>
</tr>
<tr>
<td>scraper loaders in, use of</td>
<td>41-45</td>
</tr>
<tr>
<td>diagram showing</td>
<td>44</td>
</tr>
<tr>
<td>timbering of, view of</td>
<td>42</td>
</tr>
<tr>
<td>Stope floors, ore on, removal of</td>
<td>101-102</td>
</tr>
</tbody>
</table>
INDEX

Stoping, overhand, arrangement of holes for, diagram showing.......................... 36
conditions determining use of ....................................................... 38
underhand, arrangement of holes for, diagram showing.......................... 37
conditions determining use of ....................................................... 38
Stoppings, brick, for holding back water, discussion of................................. 117
view of ...................................................................................... 118
Strong, J. E., acknowledgment to .............................................................. 2
Supports, failure of, discussion of ................................................................... 79-84
See also Concrete; Masonry walls.
Surface equipment, for iron mine, view of...................................................... 9
Switchbacks, on high dips, use of, diagrams showing........................................ 53, 54
use of ...................................................................................... 100-101
Telephones, for signaling, use of ................................................................. 129
Temperature, outside mines, effect on mine ventilation.................................... 124
Tennessee Coal, Iron, & Railroad Co., manufacture of iron by................................ 3
Thomas, H. J., acknowledgment to ............................................................... 2
Timbering, in headings, view of ........................................................................ 78
in old stope, view of ............................................................................. 73
in steep ore bed, view of.............................................................................. 71
views of ...................................................................................... 72, 73, 74, 75, 76, 77, 79
Tipples. See Rock houses.
Top rock, caving of, discussion of .................................................................... 80
effect on drainage......................................................................................... 119
view of ...................................................................................... 120
nature of, effect on loading method.............................................................. 92-93
Track, filling of, method of ................................................................. 93
grades for............................................................................................. 101
See also Branch tracks, Cross-heading tracks; Switchbacks.
Tram lines, at open-cut mines, haulage by, description of................................. 5-6
Transvaal, gold mines of, explosive rock in .................................................... 83
Trips, development of slopes for haulage by, description of.............................. 25-30
general plan for ..................................................................................... 28
United States Geological Survey, work cited .................................................... 22
Upsets, driving of, arrangement of holes for, diagram of.................................... 38
failure of top rock along line of, discussion of.................................................. 80
Ventilation, mine, changes in, suggestions for, discussion of................................... 137-128
Wagon haulage, in open-cut mines, description of............................................... 4-5
Water, entrance of, prevention of.. 112-113, 133-134
mine, sources of....................................................................................... 111-112
surface drainage of..................................................................................... 113
entrance into mines..................................................................................... 111
Weigel, W. M., acknowledgment to .............................................................. 2
Woodward Iron & Steel Co., first operations by.................................................. 3
vertical shafts of, description of...................................................................... 31-32
Workers, in slopes, protection of ...................................................................... 29-30
Workings, deep, ventilation of ......................................................................... 123
remote, ventilation of..................................................................................... 125
shallow, ventilation of.................................................................................... 123